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METALLURGICAL AND CHEMICAL ENGINEERING

A Monthly Journal being the Incorporation of
Electrochemical and Metallurgical Industry
and Iron and Steel Magazine.

VOLUME VIII

From January to December 1910

436833
3. 7. 45

NEW YORK
ELECTROCHEMICAL PUBLISHING CO

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GENERAL ALPHABETICAL INDEX

*Entries from the Synopsis of Periodical Literature are indicated by S. (Synopsis); from the Analysis of Current Electrochemical, Chemical and Metallurgical United States Patents by P. (Patents); from the Digest of Electrochemical United States Patents Prior to July, 1902, by D. (Digest). Illustrated articles are indicated by.**

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Metallurgical and Chemical Engineering

VOL. VIII. NEW YORK, JANUARY, 1910 No. 1

Metallurgical and Chemical Engineering

An incorporation of Electrochemical and Metallurgical Industry—Iron and Steel Magazine

Published Monthly by the
ELECTROCHEMICAL PUBLISHING COMPANY
239 West 39th Street, New York

PITTSBURGH OFFICE, 413 Lewis Block.
EUROPEAN OFFICE, Hastings House, Norfolk St., Strand, London, Eng.

J. W. RICHARDS, PH.D.....*President*
E. F. ROEBER, PH.D.....*Editor and Secretary*
C. E. WHITTLESEY.....*Treasurer*

Yearly subscription price for United States, Mexico and United States dependencies, $2.00; for all other countries, $2.50 (European exchange, 10 shillings, 10 marks, 12.50 francs).

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Entered as Second-Class Matter at the Post Office at New York, N. Y., under the Act of Congress, March 3, 1879

NEW YORK, JANUARY, 1910

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Ourselves.

Seven years ago this journal was founded as "Electrochemical Industry," to be the exponent of electrochemistry and its industrial applications. It has grown as these branches have developed, has put forth its best endeavors to represent every phase of this development, and will increase rather than relax in its future service to this great field of professional and industrial activity. Owing to the overlapping of, and the interrelations between, the different branches of chemistry and metallurgy, however, it was early necessary to enlarge the scope of the editorial policy so as to include more broadly the industrial applications of chemistry and metallurgy, and more particularly those of an engineering nature. This evolution first found expression in the change of name from "Electrochemical Industry" to "Electrochemical and Metallurgical Industry" in 1905, and a further step was the absorption of the "Iron and Steel Magazine" in 1906. The change of name now announced to "Metallurgical and Chemical Engineering" is the final consequence of the logical development of the past seven years, for the new title defines precisely the field which this journal has been more and more making its own in its past career. The adoption of the new name brings us privileges and lays upon us obligations—the privilege of addressing more generally than before all men interested in the engineering features of metallurgy and chemistry, and the obligation of living up to our responsibility to cover as completely as possible all important phases of these broad fields.

* * *

The work to be done is well defined and the field is not covered by any other journal. We do not address the purely mining engineer on the one hand nor the purely analyst on the other. In brief, all our efforts will be concentrated on making the best possible journal to serve the metallurgical and chemical sides of engineering. There is a natural community of interest between the metallurgical and the chemical engineer. Managers and engineers of ore concentration plants, of cyanide works, of metal smelters, of refineries, as well as manufacturing chemists and chemists engaged in engineering practice in industrial establishments of all kinds, are all more or less interested in the same processes—in crushing and grinding, concentration and separation, drying and evaporating, filtration, calcination, combustion, furnace operation, fusion, welding, in the measurement and control of temperatures, in chimney draft and forced draft, and in a hundred other lines of work. By making the word "Engineering" part of our new title, we wish to emphasize the fact that this journal is not, on the one hand, a trade paper, nor, on the other hand, a periodical representative of pure science. While neither commercial news nor scientific articles or abstracts will be absent from our columns, they will be secondary to the record and discussion of what is primarily needful in the work of the engineer or in the conduct and management of metallurgical and chemical works. Naturally, in these days when electrochemistry is extending more and

more in its scope and in its relations with other branches of metallurgy and chemistry—we need mention only the electric refining furnace in the steel industry and the fixation of atmospheric nitrogen in the manufacture of fertilizers and explosives—it will be our duty and our privilege to be as jealous an exponent of electrochemistry in all of its phases as ever. This is our birthright, which we do not relinquish by change of name. But along the broader lines above sketched, we feel convinced that this journal can be made a powerful catalytic agent in the advance of metallurgical and chemical engineering; and we sincerely hope that in our efforts to the attainment of this end, we may count on the support of all interested in the cause in which these pages are enlisted.

Again the Problem of Electrode Losses

The interesting article by Dr. J. Forssell of Sweden, published in this issue, had been written before the author had seen the article of Mr. Hering in our October issue and the papers on the same subject, presented at the last meeting of the American Electrochemical Society, and published in our December issue. This explains the omission of references to these publications in Dr. Forssell's article. While the treatment of the subject by Dr. Forssell has various original features, it is particularly interesting on account of the way in which he takes account of the heat leakage from the electrodes into the wall. He lumps this together with half the Joulean heat loss and introduces into his formulas a coefficient which may vary from ½ to 1 according to conditions. The method is feasible. But to get a simple conception of what really happens, it is perhaps preferable to consider the matter in the following way, the results being easily proven by analysis, at least under simplifying assumptions. We can consider the whole heat loss as a superposition of three phenomena. Firstly, we have the simple heat conduction, due to temperature difference at the two ends; this is the heat H that would flow if there was no electric current. Secondly, we have the Joulean heat i^2r. Thirdly, we have the heat leakage L from the electrode into the surrounding wall. The Joulean heat flows off impartially to both sides, ½ i^2r into the furnace and ½ i^2r into the water jacket outside. The heat L which is given off from the electrode to the surrounding walls is also derived impartially in equal halves from both sides, ½ L from the interior of the furnace and ½ L from the water jacket. The heat which flows from the furnace into the hot electrode end is, therefore, $H - ½ i^2r + ½ L$. The heat which goes into the water jacket is $H + ½ i^2r - ½ L$. But this latter quantity is not the whole electrode loss, since the latter includes that heat which is given off from the electrode to the furnace walls. If this goes completely to the outside, the total electrode loss is $H + ½ i^2r + ½ L$. Further advance must now be made experimentally. It would seem advisable to determine the "constants" for carbon and graphite under conditions which exclude heat leakage into the wall (by some such method as Mr. Hering's, described in our December issue) and then to determine afterwards the effect of leakage for various furnace designs and electrode constructions.

The Situation in the Iron and Steel Industry

The iron and steel industry has closed a spectacular year. The trade movement has shown remarkable swings as to prices and as to tonnage. With three years of heavy demand and gradually advancing prices, followed by a 15 months' campaign to maintain prices of finished steel products in the face of greatly reduced demand, the market broke Feb. 18, 1909. Perhaps it is more accurate to say that on that date the decision was reached by officials of the United States Steel Corporation, in conference with representatives of some of the important independent interests, to throw the market open, and that thereafter different portions of the price structure collapsed one at a time. Within two days of the decision steel bars suffered a reduction of $4 a ton and plates and shapes a reduction of $6 a ton. One business day later, Feb. 22, steel pipe was reduced five points, or about $9.50 per ton, and two days thereafter sheets dropped $4 to $5 a ton. On March 15 tin plate was cut 25 cents per box. Wire prices were moderately well held to give jobbers an opportunity to distribute their stocks, and on May 1 cuts were made of $7 a ton on nails, $8 on plain wire and $10 on barb wire. In steel pipe, tin plates and wire products the initial cuts were found sufficient to meet the situation, while in bars, plates, shapes and sheets the first cuts were followed by shading which established new low points. There was no cut in standard sections of rails. The subject was canvassed at the time the market on other finished steel products was thrown open, and it was concluded that no large business would follow a reduction. In succeeding months individual manufacturers stood ready to make a sharp cut in rails, if it appeared that large orders could be secured thereby, but it was evident that the railroads were not prepared to buy rails heavily. In no line did it require a long time to develop a bottom. Bars, plates and shapes were trending upward within a little over two months from the time of the original cut; tin plates and pipe did not advance until long after their reductions, but in each case there was but one reduction, the market being held firm thereafter. The lowest prices on wire products were open for a single fortnight, the reductions being made May 1 and the first advances May 15. Sheet products were slowest to find a stable basis, but there was double delay in that case; prices of sheets were reduced Feb. 24, but there was a long delay before the steel mills made their lowest prices on sheet bars; when the minimum on sheet bars was finally reached the independent sheet mills loaded up and fresh cutting occurred in their finished product, and it was not until July that sheet prices showed a discernible tendency to advance.

* * *

Of the wisdom or unwisdom first of the price-maintenance movement and then of its abandonment, *quot homines, tot sententiae*. The game was played with such pomp and circumstance that the sudden abandonment could not but shock the aesthetic senses of the onlookers. Those who had been nearer the stage had heard the creaking of the machinery and had felt that the play was being carried on partly to amuse the audience and partly to keep the players out of mischief. If the undertaking was as serious as its sponsors maintained during its progress, the disruption of the forces should have been regarded as a calamity, but the event has proved what was maintained by some observers at the time, that a definite break would be followed by a rapid recovery. Price maintenance, as has been pointed out before, was no new thing in the steel industry, on many occasions there had been efforts in that direction. The trappings, however, were new. The break has proved to be but

an incident in history. Substantially the entire reduction in finished steel products was recovered by the subsequent advances, except in the case of pipe. Most finished steel products closed the year at prices $1 to $2 a ton below the opening prices, but the closing prices were firmly maintained, while the opening prices were largely nominal, and subject to shading. Merchant steel pipe, however, is a conspicuous exception, as it experienced but a small recovery.

* * *

No less spectacular than the movement in prices has been the movement in tonnage. It strained the capacity of existing plants to make 23,000,000 tons of pig iron in 1905; next year under similar strain the increased capacity produced 25,300,000 tons; in 1907 the industry, still pushed with increased capacity, was making for an output of about 27,350,000 tons and had reached a rate closely approximating 28,000,000 tons in October, but the panic of that month caused a sudden slump and the calendar year's output was only 25,781,361 tons. In January, 1908, the rate of output had dropped below 13,000,000 tons, yet at that time there were new furnaces in course of erection with an annual capacity of 3,500,000 tons. Where was the hope that capacity would again be fully employed, in any reasonable period of time? The year 1908 gave no answer, for its total output fell below 16,000,000 tons, but the year 1909 showed the wonderful resources of this prince-and-pauper industry. It opened with a rate of 21,500,000 tons, showed an average rate of 22,000,000 tons during the first half, and closed with a rate of nearly 32,000,000 tons. With the present rate of production the physical capacity has been closely approximated. There are a number of old furnaces not yet in blast, and several new ones shortly to be completed; apparently there is perhaps 2,000,000 tons of capacity left to come in, but practically the present rate is a fictitious one. Many blast furnaces were relined and otherwise repaired during the depression, and the efficiency of the furnaces as a whole is at present abnormally high. It is normal for close to 10 per cent of the furnaces to be out from one cause or another, but at the moment the idle capacity, including furnaces not completed, is less than 10 per cent of the capacity in blast. It is improbable that the existing plant could steadily make 32,000,000 tons a year. With the new furnaces, some nearly completed and some only projected, a capacity of 33,000,000 tons or more is easily in sight.

* * *

We have been dealing with pig iron. Of the 25,781,361 tons made in 1907, the record calendar year, about 18,000,000 tons went into steel. The gain in pig iron, from a maximum rate in 1907 of nearly 28,000,000 tons a year to a rate of nearly 32,000,000 tons a year at the close of 1909, was almost wholly in steel-making iron. The foundry trade lagged behind in the industrial revival, as it usually does, the trade dictum being that the foundry industry is six months behind the steel industry in such cases. That practically the entire gain should be in the steel industry is remarkable, as the percentage of gain is increased, but it is still more remarkable to observe that rail production has fallen behind, and consumption of plates and shapes in the steel-car industry has certainly made no gain over 1907. The gain in the total, plus an amount to cover these defections, must be found in other lines of consumption. It does not seem that there has been a gain in bridges and buildings. There must be an enormous gain, on the other hand, in the light lines, pipe, wire, sheet products, merchant mill products, etc.

* * *

When the steel industry of the United States confined itself almost exclusively to the Bessemer process, pig iron production was a close index to finished steel production. A given tonnage of Bessemer pig iron being produced, the metalloids were eliminated and a little iron lost, most of the scrap made in casting and rolling went back into the converter, and the finished Bessemer steel was regularly a small percentage below the original pig iron production. Nowadays the case is complicated by the open-hearth furnace, with its scrap and ore additions, some of the scrap being old material but the bulk of it new material produced in Bessemer and open-hearth steel works. We know that in 1909 practically the same tonnage of pig iron was produced as in 1907, and that there was considerably more steel-making pig iron and considerably less foundry iron and other grades. We know, too, that the Bessemer steel works did not do as well in 1909 as in 1907, and hence we are perfectly safe in concluding that, the acid open-hearth being a minor issue, the basic open-hearth furnaces consumed a great deal more basic pig iron in 1909 than they did in 1907, but what good does that conclusion do us as an index to steel production? In past years the production of basic open-hearth steel ingots has been roughly double the production of basic pig iron. If each additional ton of pig iron in 1909 was used with an additional ton of scrap, the gain in steel production was very large, especially since there was less new scrap to be drawn from Bessemer plants. Old material, however, was scarce in 1909, as shown by market prices compared with pig iron, and it may be that some of the additional pig iron production was for the purpose of making up for the shortage in scrap, in which case we should find a much smaller gain in steel output. The official statistics will be awaited with unusual interest. Perhaps the best that can be said at this time is: In 1907 we made 25,781,361 tons of pig iron, the calendar year record, and in 1909 we made substantially the same tonnage; in 1906 we made 12,275,830 tons of Bessemer steel ingots and castings, the record; in 1907, 11,667,549 tons; in 1909, probably considerably less than 11,000,000 tons; in 1907 we made 11,549,736 tons of open-hearth steel ingots and castings, the record; in 1909, probably more than 13,000,000 tons; in 1907 we made 17,664,736 tons of finished rolled steel, the record; in 1909, probably half a million tons either way from 19,000,000 tons.

* * *

The pinch of an insufficient scrap supply noted in the foregoing has not been without its effect in the selection of processes for the future. Late in the year the Bethlehem Steel Company, having before it alternative propositions of adding to its Saucon plant of ten 50-ton basic open-hearth furnaces either 10 similar furnaces or two 20-ton Bessemer vessels for duplexing, chose the latter; the Jones & Laughlin Steel Company experimented with duplexing by combining its Bessemer and open-hearth equipment, originally built with no such idea, and the United States Steel Corporation, which originally made authorizations for the Gary plant, involving an open-hearth ingot capacity more than double the blast-furnace capacity, has been considering duplexing and the addition of blast furnaces without corresponding open-hearth additions.

Perkin Medal.

At the meeting of the Perkin medal committee on Monday, Dec. 13, at the Chemists' Club in New York City, in which delegates of the Society of Chemical Industry, the American Chemical Society and the American Electrochemical Society participated, the Perkin medal for 1910 was awarded unanimously to Dr. Edward G. Acheson, of Niagara Falls. The formal award will be made at the January meeting of the New York Section of the Society of Chemical Industry.

Boston Meeting of the American Chemical Society.

The forty-first meeting of the American Chemical Society is to be held in Boston in the last week of December. It will begin on Monday, Dec. 27, and last to Friday, Dec. 31. Special arrangements have been made for the entertainment of the visitors, while the program for professional papers to be presented before the different Sections contains not less than 111 papers. Of these there are 60 to be presented before the Division of Industrial Chemists (Mr. A. D. Little, chairman). Before the same section there will also be presented a symposium of papers on the chemistry of paints.

Of lectures of general interest we note the report by Dr. F. W. Clark for the International Committee of Atomic Weights, a paper by Dr. T. W Richards on methods employed in precise chemical investigations, a paper by Dr. Louis Kahlenberg on the past and future of the study of solutions, and the address of the retiring president of the American Chemical Society, Dr W R. Whitney, on "Some Chemistry of Artificial Light."

Dr. C. L Parsons, Durham, N. H., is the secretary of the society

Annual Meeting of the Western Association of Technical Chemists and Metallurgists.

The fifth annual meeting of the Western Association of Technical Chemists and Metallurgists will be held at Boulder, Golden and Denver on January 6, 7 and 8, 1910. There is an interesting program of papers and a number of visits have been arranged by the local committees to various industrial plants, including smelters, beet sugar factories and breweries. The various laboratories of the State University and the State School of Mines will also be visited.

Mr H C Parmelee, 1510 Court Place, Denver, Colo., is the secretary of the association.

Philadelphia Meeting of the American Institute of Chemical Engineers.

The second annual meeting of the American Institute of Chemical Engineers was held in Philadelphia from Dec. 8 to 10 and the attractive program, which was printed in our last issue, was successfully carried out. The attendance at the meetings and numerous excursions and at the dinner in the Hotel Walton was very satisfactory.

The address of the retiring president, Dr S P Sadtler, on the relation of the chemical engineer to the conservation problem is printed in full on page 9 of this issue, while abstracts of some of the papers presented will be found in the Synopsis in this issue. Of other papers presented we note one by Mr E R Taylor on the electric furnace for smelting iron ore, which covered in a new form essentially the same ground as his Electrometallurgy paper published in our last issue

The following officers were elected for next year: President, Dr Chas F McKenna; vice-presidents, Dr F W Frerichs, Dr Edward G Acheson and Dr Eugene Haanel, secretary, Dr John C Olsen, Pratt Institute, Brooklyn, N Y; treasurer, Wm M Booth; auditor, H D Reynard; directors, G B Adamson, D Wesson, Dr F Goldmann, L Reuter, Thorn Smith, H F. Brown, Dr. W. M Grosvenor, R K. Meade, Dr S. P. Sadtler.

Dr. Chas. F. McKenna, the newly elected president of the American Institute of Chemical Engineers, is a new Yorker by birth, 48 years of age, an M.A. of St. Francis Xavier's College and a Ph.D. of Columbia University. He was chairman of the committee of six which organized the Institute and has been first vice-president since its organization.

His earliest work in chemistry was with the Havemeyer Sugar Refining Company and the Cambria Steel Company.

In 1887 he designed, erected and operated a plant for the manufacture of lime for the purification of gas, and this led him first into the study of the utilization of lime wastes and later into the field of limes, mortars and cements, in which he has won a high reputation. In the manufacture of Portland cement he has acted for many of the manufacturers in the study of their raw materials and design of plants.

In the beginning of his work as a consultant and chemical expert he was associated with Dr. Gidson E. Moore and established a testing laboratory of which he is still the head and which has aided him to attain a position of authority upon materials their qualities and processes of treatment.

He has been a chemist to several metallurgical and manufacturing concerns and is frequently consulted by the authorities of his own city, while his devotions to the interests of chemistry, chemists and chemical education are well known and generally recognized.

The New Chemists' Club Building.

With great pleasure we are able to state that the preliminary arrangements for the erection of a new large Chemists' Building, worthy of the industry which it is to represent, have been successfully concluded.

The capital of the Chemists' Building Company is to be $250,000, of which $210,000 have been subscribed by nearly 100 individuals or companies. The remainder of the money for the erection of the building will be raised by a mortgage

The directors of the Chemists' Building Company are Dr Morris Loeb, president; Dr Chas. F. Chandler, vice-president; Albert Plant, treasurer; W. H. Nichols, Jr., secretary, Dr Leo H. Backeland.

As already stated, the building is to be erected at 50 to 54 East Forty-first Street, a plot 56 x 100 ft.

The six lowest stories will be devoted to the purposes of the Chemists' Club, while the upper five floors will be used for laboratories.

At the entrance to the club rooms will be a reception room, office and coat room, back of which is a large hall, 50 ft. square, which runs over two stories The front of the second story will be devoted to billiard room, etc. The third story will contain a large social room and the restaurant The front of the fourth story will be devoted to the library, and the back to the board room, a dark room and a chemical museum. On each of the next two floors will be found to bedrooms for club members

It is hoped that the library will be made a thoroughly complete one, and house not only the books of the Chemists' Club, but those of the American Chemical Society and those of the Perkins Fund as well.

The Museum will be devoted to the collection of articles of scientific interest, rather than such as are adapted to popular exhibitions It is also hoped that it will contain specimens of all kinds of chemicals to be used for research in about the same way as a circulating library, that is to say, specimens can be taken out for experiments, or for comparison, on the condition that they be returned or replaced within a given period. The meeting room will be let out to scientific societies and others.

The laboratory floors are to be provided with all the mains necessary to supply the individual laboratories with gas, electricity, water, etc. They are also to contain ventilating flues,

and such provision for hoods that these can be introduced at minimum cost. Indeed, they could probably be provided at once to any tenant who made a lease before the completion of the building. These facilities have been so designed as to make a subdivision of each floor into four individual laboratories with adjacent offices, but, at the same time, it will be easy for a single tenant to make satisfactory use of a whole floor, or half a one.

While the rentals of the laboratories must, necessarily, provide an income for the building, they will be placed at as low a figure as possible, in order that commercial chemists as well as scientific investigators may obtain laboratories at a reasonable cost under far better conditions than those prevalent elsewhere. The advantage will be a fireproof laboratory, with the same improvements that can be found in a modern college building, a central library within reach and those benefits which usually go with the occupancy of a building devoted to a single profession.

Of course, these laboratories can also be rented out for work in cognate sciences, such as physics and biology.

The stockholders have agreed to limit their dividends to 3 percent, and any surplus will be applied to reducing the capital indebtedness of the company. At the same time the Chemists' Club has an option to purchase the entire stock of the company at par within the next 50 years. This will make it possible for the club to purchase the building whenever the capital has been sufficiently reduced by the operation of the provision just stated.

On the other hand, should the club for any reason fail to prosper, the stockholders will have the building, which is always applicable for business purposes, since it is in a central location, and will be constructed in a perfectly satisfactory manner for conversion into ordinary offices.

Municipal Chemistry.

The Department of Chemistry of the College of the City of New York (Prof. Chas. Baskerville, Director) offers during the Spring Semester a course of thirty lectures on the chemistry of daily life.

These lectures will be open to three classes of hearers:

1. Senior students of the college who have complied with the requirements of the department. Laboratory practice (no fees) involving the examination of water, milk, foods, gas, paints and cements, will be had under the direction of Associate Professor Moody.

2. Employees of the city who have studied sufficient chemistry to pursue the laboratory work. The attainments of each applicant will be passed upon individually. The expense involved in apparatus broken and chemicals used must be met by the hearer.

3. A limited number of auditors composed of citizens of New York City will be admitted on applying for a seating to the Director of the department. The last are not admitted in the laboratory part of the course.

The lectures will be given as per printed schedule, at 4 p. m. in the Doremus Lecture Theatre, Chemistry Building, 140th Street and Convent Avenue, Plaza entrance.

All communications should be addressed to the Director, Dr. Chas. Baskerville.

The course of lecture is as follows:

Feb. 4th, Prof. Charles Baskerville, Sanitation.

Feb. 9th, Dr. Wm. P. Mason, Drinking Water and Disease.

Feb. 10th, Dr. Wm. P. Mason, Sources of Municipal Water Supply.

Feb. 11th, Dr. Wm. P. Mason, The Purification of Polluted Water.

Feb. 15th, Dr. Thomas C. Darlington, Milk.

Feb. 18th, Dr. Harvey W. Wiley, The Purpose, Method, and Extent of Food Adulteration.

Feb. 19th, Dr. Harvey W. Wiley, The Remedy of Food Adulteration and Relation of Chemistry thereto.

Feb. 25th, Mr. Bayard C. Fuller, Food Inspection.

March 1st, Mr. Bayard C. Fuller, Spoiled Foods.

March 4th, Dr. Virgil Coblentz, Drugs and their Adulteration.

March 8th, Dr. Virgil Coblentz, Methods for Detecting Adulteration.

March 11th, Dr. Virgil Coblentz, Habit Inducing Drugs.

April 1st, Dr. Allerton S. Cushman, Streets and their Construction.

April 2d, Dr. Allerton S. Cushman, Modern Road Construction.

April 5th, Hon. William H. Edwards, Street Sanitation.

April 8th, Mr. Edward D. Very, Disposal of Ashes and Light Rubbish.

April 12th, Dr. Edward D. Very, Disposal of Garbage.

April 15th, Dr. Edward D. Very, Disposal of Putrescible Materials.

April 19th, Dr. Arthur H. Elliott, Manufacture of Gas.

April 22d, Dr. Arthur H. Elliott, Means of Testing the Properties and Quality of Gas.

April 26th, Dr. Arthur H. Elliott, The Smoke Problem.

April 29th, Dr. Herbert R. Moody, Ventilation.

May 3d, Dr. Thomas A. Storey, The Chemistry of Personal Hygiene.

May 10th, Mr. Maximilian Toch, Paint and Painting.

May 13th, Mr. Maximilian Toch, Corrosion of Metals and its Prevention.

May 17th, Mr. Maximilian Toch, Cement and Concrete.

May 20th, Dr. A. A. Breneman, Combustibles and the Causes of Fires.

May 24th, Dr. A. A. Breneman, Methods of Extinguishing Fires.

May 27th, Dr. N. L. Britton, City Parks, Gardens, and Playgrounds.

The Iron and Steel Market.

December has been a very quiet month in the iron and steel trade, but the market has shown a firm undertone and the lull is regarded as merely the seasonable one which is practically inevitable at this time. No sudden resumption of activity is expected, but a gradual resumption of buying is anticipated as the first quarter is well entered upon, and producers are holding prices of finished steel products firmly, with the tendency to ask higher prices for first half than for first quarter deliveries.

The high pace of production reached in November continued through December. No important additions to capacity were made in the month outside of the blowing in of a few blast furnaces here and there. The iron and steel industry had gotten up very close to the limit of its capacity in October and November, and looked rather for some decrease in output in December on account of weather conditions, but these have been fairly favorable, and the month's statistics are likely to show the best output of the year.

Price changes in December were unimportant, the principal finished steel products being untouched, with the exception that at the close of business Saturday, December 12, a general advance of $1 a ton was made in wire products. These prices apply to jobbers, with whom it is customary to make contracts requiring specifications within 60 days. Specifications on contracts placed at previous prices were not heavy, and the advance was presumably made chiefly with a view of stimulating specifications. Prices had already been made to manufacturers using plain wire at a $2 advance on contracts for deliveries over the first six months of the new year, so that the advanced price to jobbers is $1 a ton below the price made to manufacturers, and another $1 advance to jobbers is expected January 1 or January 15.

Specifications have been heavy on contracts for finished steel products expiring December 31, this applying particularly to steel bar contracts. Such contracts carried prices below the present level, and buyers were anxious to escape having any of the contract tonnage cancelled by the mills at the time limit.

The finished steel trade enters the new year with fully three months of actual operations specified on books in the case of steel bars, and nearly as much specified in plates and shapes. Sheets and tin plates are sold under definite contract involving full half, and perhaps more than half, the prospective production in the first half. Wire and pipe are sold less far ahead, as it is contrary to the policy of the mills to sell for more than about 30 to 60 days ahead.

As a result of the disorder attending the attempt to start the Ætna-Standard sheet plant, of Bridgeport, O., of the American Sheet & Tin Plate Company, it seems to have been decided to abandon all efforts to start the Wheeling district plants. June 30, on the expiration of the annual scale, the members of the Amalgamated Association walked out of all the sheet and tin-plate mills of the American Sheet & Tin Plate Company which had hitherto been union, with the exception of the Guernsey sheet plant, at Cambridge, O., the company having announced that it would operate on the open-shop principle thereafter. Subsequently operations were started at all the sheet and tin plate plants affected, with non-union labor, with the exception of the plants in the Wheeling district, comprising the Ætna-Standard sheet plant at Bridgeport, O., 23 mills; the La Belle tin plate plant at Wheeling, 10 mills, and the Laughlin tin plate plant at Martin's Ferry, O., 23 mills. Wheeling has been a stronghold of union sentiment, and no effort to start operations was made until December 1, when a few mills at the Ætna-Standard plant were started with old workmen, who had requested the management to give them employment. This brought about a small riot, with the calling out of 1500 troops. Cecil A. Robinson, manager in the Wheeling district, has resigned, effective January 1, while D. A. Barrett, superintendent of the Laughlin plant, has been transferred to the Morewood tin plate plant at Gas City, Ind., containing eight mills, which have operated only about one month in the past four years. It is intended to put the Morewood plant in operation, while there appears to be no intention of trying to start any of the Wheeling district plants, at least in the near future.

Statements have been made regarding the building by the Steel Corporation of a large sheet and tin plate plant at Gary, Ind., but it is possible that the announcements at this time are merely an incident of the labor difficulty, since the plans for 50 tin plate and 50 sheet mills for Gary were drawn up more than a year ago. Through the dismantling of a number of sheet and tin plate plants by the American Sheet & Tin Plate Company and the building of a large number of independent plants the Steel Corporation has lost much in relative position, and would not entirely restore its former relations in the industry by building even a 100-mill plant at Gary. The growth of tin plate and sheet consumption has been very rapid, and there is no reason to suppose that demand would not be found for the additional output.

Coke.

Asking prices for Connellsville furnace coke have yielded slightly in the month, to $2.75 or $2.80, but not enough to induce the non-covered furnaces to come into the market on contracts for the new year. During the spectacular advance from less than $2 in July to $3.10 in September a number of contracts were made, two or three small ones being understood to have been made at the top price, but there remains large uncovered requirements. Consumers' expectations seem to be that there will be no collapse, and that there may be a decline. Meanwhile, if they do not make contracts, they can buy prompt coke from time to time.

Pig Iron.

The local pig iron market has been quiet, but prices have been well held. Bessemer is $19, valley, for any delivery, but no important sales have been made since those late in November whereby two subsidiary steel interests took a total of 28,000 tons at this price for the first quarter delivery. Moderate sales of basic have been made for first quarter delivery at $17.25, valley, there being a little tonnage reported as available at $17 for immediate shipment. Foundry iron is $17 to $17.25, valley, depending on delivery, with forge about 75 cents less. Prices delivered at Pittsburg are 90 cents higher than valley. Owing to the offering of speculative Southern iron at $14.50, Birmingham, and then at $14, but only for immediate shipment, the Southern furnaces have yielded from their $15 price and have been ready to do $14.50 for first-half delivery. This price has brought out a moderate amount of business.

Finished Materials.

Prices of finished steel products are very firmly held as below. In cases where a range is indicated, the higher price is for the more extended delivery. Prices are at Pittsburg unless otherwise noted:

Standard steel rails, $28 for Bessemer, $30 for open-hearth, f.o.b. mill, except Colorado.

Plates, $1.55 to $1.60 for tank quality.

Shapes, $1.55 to $1.60 for beams and channels, 15-inch and under, tees, and angles 2 x 3 and larger.

Steel bars, $1.45 to $1.50, base.

Iron bars, $1.70 to $1.75; Chicago, $1.60.

Wire nails, $1.85 base; plain wire, $1.55; galvanized barb wire, $2.15.

Black sheets, 28-gage, $2.40; galvanized, $3.50; blue annealed sheets, 10-gauge, $1.75; painted corrugated roofing, $1.75 per square; galvanized corrugated roofing, $3.

Tin plates, $3.60 for 100-lb. cokes.

The Faraday Society.

The forty-seventh meeting of the Faraday Society was held on Tuesday, November 30, 1909, at the Institution of Electrical Engineers in London. Mr. James Swinburne was elected president.

A paper by Dr. Henry J. H. Sand dealt with the **electro-analytical determination of lead** as peroxide.

This investigation was carried out with the object of elucidating the cause of the discrepancy between the statements of various experimenters regarding the behavior of an electrolytic lead peroxide deposit in drying at 200°. All authors agree that the peroxide deposit retains water at this temperature, but whereas Hollard and Bertriaux give an analytical factor of 0.853, most other investigators find a factor of approximately 0.864, the theoretical factor being 0.866. Incidentally, the effect of varying conditions on the coherence of the deposit was also studied.

It was found that at 200° a lead peroxide precipitate is capable of absorbing moisture from a damp atmosphere, and an increase of as much as 1.7 per cent of the weight of the deposit has thus been obtained. On heating in a dry atmosphere at the same and higher temperatures, the peroxide loses its water exceedingly slowly.

The most satisfactory results were obtained by depositing the peroxide in such a manner that it is practically free from water owing to electric endosmose on leaving the bath. The conditions are a high temperature and a large current density (with a rotating cathode). The deposit was dried by means of alcohol and ether in the identical manner employed for metal precipitates.

A paper by Mr. Arthur Jacquerod discusses the influence of dissolved **gases** on the **electrode potential** in the system of silver, silver acetate, aq.

Mr. Arthur Jaquerod also contributed a paper on **ionisation** in aqueous solutions in lead acetate and cadmium acetate.

A paper entitled "The **calorimetrical analysis of hydrated salts**" was communicated by Prof. F. G. Donnan and Dr. G. D. Hurst. The authors point out that the interpretation of the heats of solution of hydrated and partially dehydrated salt given by Thomsen in his Thermochemische Untersuchungen is in various cases either erroneous or unsatisfactory.

It is shown that Thomsen's data for sodium carbonate indicate," when correctly interpreted, the existence of only the

hydrates with 1, 7 and 10 mols. water per mol. anhydrous salt. The authors' experiments confirm this result.

In the case of copper sulphate, neither the experiments of Thomsen nor those of the authors indicate more than the existence of the hydrates $CuSO_4 . 5H_2O$ and $CuSO_4 . H_2O$, though the hydrate $CuSO_4 . 3H_2O$ is known to exist.

This calorimetric method may be employed as an aid in the discovery of the existence of salt hydrates, though it is probably neither so easy nor so reliable as other well-known methods.

The Aluminium Situation.

According to a communication published from trade circles in the *Frankfurter Zeitung* and noticed in the London *Electrical Review*, the hopes of an understanding among the French producers of aluminium, after months of preparatory work, had recently to be abandoned. The inland understanding between the French firms, who produce a very large part of the total output of aluminium, would naturally have formed the basis for a far-reaching combination. But from the beginning the allotment question or the regulation of the output of the individual works seems to have been incapable of solution.

It is quite obvious that the powerful undertakings which during the period of high prosperity carried out what were doubtless exaggerated extensions now object to renounce their comprehensive schemes in relation to production. Nevertheless, they will find themselves, if they also wish to improve the price of the metal, in the situation which they have themselves created through the excessive straining of the (late) syndicate prices.

The price of aluminium improved from the time of the appearance of the first syndicate rumor in June, 1908, but since the recent definite abandonment of this community of interests consumers have manifested a natural reserve. This circumstance is aggravated by the accumulating offers of intermediaries who seem, owing to the continued circulation of syndicate rumors, to have supplied themselves with large quantities at the lower prices prevailing until recently. But as their speculative purchases were not so quickly realized as they had hoped for, they are inundating the market with cheap prices, which again appear at about 1s. 3d. per kilogram (14 cents per lb.) and lower for crude aluminium.

Apart from this, consumers are deriving advantage from the situation. There is no doubt that the consumption of the metal is increasing and must in all probability continue to progress if the price is kept within normal limits. These may be regarded as from 1s. 5d. to 1s. 7½d. per kilogram (16 to 18 cents per lb.) for crude ingots or plates with an extra of from 5d. to 5½d. (10 to 11 cents) for the rolling out of sheets. On the basis of this charge for sheets, the manufacture of aluminium vessels and various articles is remunerative, but the overstepping of this maximum would reduce the consumption. In conclusion, it is stated that the present average price of 1s. 3d. per kilogram (14 cents per lb.) for crude aluminium only represents a small profit, but it no longer directly results in a loss.

All this refers, of course, to Europe. For this country the protective tariff is to be taken into account.

CORRESPONDENCE.

Electrical Smelting of Copper Ores.

To the Editor of Metallurgical and Chemical Engineering:

SIR—The electric furnace has acquired a footing in the iron and steel industry, but as yet little has been done in the field of copper smelting. As an accessory to a matte-smelting blast furnace or reverberatory, electric smelting is, however, capable of being employed to advantage in many cases.

As a general proposition, the fuel energy required to increase the temperature of one unit of mass by one unit of temperature increases greatly with the initial temperature of the mass, and to raise the temperature of a given mass from 1500 to 1600 deg. requires more fuel energy than is required to heat it from 100 to 1100 deg., and vastly more than is required to heat it from 500 to 600 deg. In general it increases in a geometrical progression as the temperature increases. Heat generated in a mass by electrical resistance is, on the other hand, cumulative, and except for loss by conduction and radiation it takes no more current and costs no more for 100 deg. increase of heat at 1500 deg. initial temperature than at 1000 or 500 deg.

This points to a field where electrical heat can be used to advantage, viz., for carrying the summit of the smelting load under conditions where fuel heat does the work of smelting up to the point where an electric current can act upon the material to generate internal heat, the electric current then finishing the operation. In other words, to an adjustment of conditions such that the smelting by fuel heat will proceed to the point where there will be formed a semi-fluid mass capable of flowing and forming a bath, when the completion of the smelting will be done by electric heat.

In the smelting of sulphide ores we are dealing with material of a complex nature, composed of elements or constituents having different temperatures of fusion, and the constituent of lowest smelting temperature fuses first and the others in the order of fusibility.

As in the freezing of molten alloys and mixtures where the element of highest freezing temperature solidifies first and the elements of lower freezing temperature remain as land-locked bodies within the chilled mass until the temperature falls to their respective freezing points, so in the fusion or smelting of ores containing components having different fusion temperatures, the constituents of lowest smelting temperature fuse first and drain down into the bath, carrying downward particles of unfused constituents of a higher fusion temperature or leaving a spongy mass composed of such constituents. In straight fuel smelting in a blast furnace conditions require that the fusion of all elements, those having a high-fusion temperature as well as those of a low smelting point, be accomplished at or above the plane of the tuyeres. Below that plane the space is filled with incandescent coke, which, being in a non-oxydizing atmosphere, remains as a support for the charge. At the plane of the tuyeres everything except the coke melts and drains down through the incandescent coke bed into the bath. The result is that in straight fuel smelting the consumption of coke is largely in excess of theoretical requirements, for it must be sufficient to make a free-flowing fluid of all material and preserve the coke foundation. In straight pyritic smelting the problem appears to be complicated by the small margin between the fuel energy of the contained sulphur and iron and the heat requirements of complete fusion, and further, probably, by the formation of strata of material of high fusion temperature below the tuyere plane in lieu of the incandescent coke foundation before noted.

The electrical heating of the bath in a matte smelter will permit of the cutting down of the coke charge to the point where the coke will do the smelting of the more fusible material and the electric current the balance, or if the fuel value of the contained sulphur and iron approximates to that required for the basic smelting, the use of coke can be dispensed with. This can be secured by a hearth of a trough character with electrode terminals, and also a fore hearth with a flowing tap, so as to maintain in the trough hearth a bath of constant level or cross-section. The hearth should so discharge into the forehearth that there will not be an accumulation of matte. The current, of course, is of large amperage and low voltage, the resistance of copper slag indicating a working voltage of from 5 to 7 volts. Estimates indicate that 1 electric horsepower will give a control of 200 deg. (Fahr.) per minute per ton of ore capacity per day, so that 1 electric horsepower will heat 1.4 pounds matte-slag 200 deg. per minute.

This should suffice, for it is to be borne in mind that the elec-

[illegible] is cumulative, and is added to material already close to the fusion point; that the time or capacity element is the one to be affected by the quality of the current; that if the volume of current is small it simply takes a longer time for it to supply the deficiency for smelting a given quantity of semi-fluid material than it will take when the volume of current is larger. In other words, that the electric current is a governor supplying just what heat is needed for each unit of product to give a uniform temperature.

With the coke charge reduced to the amount simply required for the basic fusion and all consumed at or above the tuyeres, or in the case of pyritic ores eliminated entirely, the result will be that the space below the tuyere plane will be occupied by material of an open, spongy character of progressively varying fusibility with that of highest fusibility at the bottom. The fused products of the fuel smelting draining down into the bath will, with the excess of heat they carry down, complete the smelting of matter of the lowest smelting temperature met with.

Elements of lowest fusibility will be smelted out in advance of elements of higher fusibility, and consequently the material resting on the bath and pressed down into the bath and extending up to the plane of the tuyeres will be of varying fusibility with that of highest fusibility at the bottom. The electric heat is therefore expended directly on the material of highest fusibility, while fuel smelting starts with the material of lowest fusibility and works up in the scale of temperatures, so to speak, to the elements of higher fusibility. Under proper adjustment of current and fuel, fuel smelting and electric smelting will meet at the point where, everything considered, the result is most economical.

If accessory electric smelting is to be used as a control for pyritic smelting so as to make it practical, the current need not necessarily be so very heavy, for, being cumulative, the electric heat will in time supply the balance required.

Naturally, we look to hydro-electric power, where it is obtainable, as the source of current supply. But accessory electric smelting will probably permit of the use in many cases of comparatively cheap fuels, oil, producer gas, waste, heat, etc., in lieu of high-priced coke.

Electric smelting can not only be employed in connection with blast-furnace smelting, but it can be used in reverberatories. The adaptation to reverberatory practice of electric smelting as above outlined calls for a new type of reverberatory, one in which there can be maintained two long banks of ore or charge with a trough between, this trough having electrode pockets or connections at both ends and a forehearth with a flowing tap. The reverberatory has rows of charging holes in the roof next to the sidewalls and the charge, banked against the sidewalls and protecting them, presents a crest of summits and saddles with slopes leading down to the central trough into which the semi-fused material drains.

In a reverberatory furnace a large amount of waste heat is available for current generation, and thus the entire heat energy can be utilized, either primarily as fuel heat or secondarily as electric heat.

STORY B. LADD.

Washington, D. C.

Electric Smelting of Iron Ores at Domnarfvet and at Nykroppa, Sweden.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—In order to show to what extent the Iron & Steel Associations of Sweden are putting faith and confidence in the work and results already accomplished in electric smelting of iron ores by the use of the "Elektrometall" Company's electric smelting furnaces, it is interesting to learn that the "Jernkontoret" at a meeting in November last, appropriated the sum of [illegible] kronor toward the immediate construction of an experimental electric furnace plant at Trollhättan, of the above type and of [illegible] daily capacity.*

As already has been mentioned in the *Electrochemical and Metallurgical Industry*, a 700-hp electric furnace of this type was in successful operation at Domnarfvet for some time, and, according to the very complete report† of Lars Yngström in the transactions of the "Jernkontoret" (Part 9, 1909), these results have demonstrated that the question of electric production of iron from ore "is now economically as well as practically solved."

The furnace which the Swedish Iron & Steel Association is to erect, therefore, ought not to be considered as a further experiment in this direction, but as an "experimental plant" to which any iron manufacturer or mine owner can send his ore and have the same treated for obtaining information as to its suitability for such a process, the purpose for which the resulting metal would be best adapted, and the cost of manufacture, as compared with iron produced from the same by means of charcoal or coke.

The business arrangements arrived at between "Jernkontoret" and the "Elektrometall" are reported in the Swedish daily press to comprise the following principal parts:

The "Elektrometall" shall without further cost execute and furnish all necessary detailed drawings and specifications of building and furnace appliances; also during the construction period furnish all required advice and information.

The "Jernkontoret" shall defray all expenses in connection with the construction of the plant and have full charge and final decision as regards the same. The Jernkontoret reserves the right of exercising the patent rights for their own use or to dispose of such rights to others, in consideration of a first payment equal to 2 krone per electric horse-power for each particular size furnace or furnace type erected, besides a royalty of 1.25 krone per metric ton pig iron produced according to the "Elektrometall" Company's patents during the life of the same.

From this, however, is excepted the present experimental plant, for which no further charge or royalty shall be paid.

As regards power for the present plant, the "Jernkontoret" has made an arrangement with the Swedish "Hydro-Power Commission" whereby during two years they will, for this particular purpose, obtain up to 3000 hp at Trollhättan for the annual sum of 10,000 krone (equal to about $3) per horse-power year.

In the petition to the Government the "Jernkontoret" gives some interesting information on the results obtained from the late electric smelting experiments at Domnarfvet and Nykroppa that during said experiments charges of different kinds of iron ores and briquettes have been used (both high and low in P-contents); that the coal consumption has been brought down to about one-third of that of the ordinary blast furnace practice; that the maximum efficiency of the furnace so far has been only 58 per cent (whereas that of the ordinary blast furnace can be considered to be about 80 per cent), but that there is good reason to expect a still further reduction in heat losses for the furnace proposed, by changes in the water coolers about the electrodes and in adopting thicker furnace walls. Mention is also made of the superior quality of some of the electric pig iron made from the Tuolluvara ore at Nykroppa and used in one of their 10-ton O H furnaces. This was white iron, which usually is cold and sluggish, but which here proved to be the very reverse, the iron being hot and liquid, and allowing even a higher percentage of ore charge than the ordinary. The cold drawn pipes made from the steel products proved also to be of superior quality.

Owing to the successful results obtained from the Swedish experiments a large Norweg an electric smelting plant has also been decided on, to be constructed in accordance with the principles embodied in the "Elektrometall" Company's patents.

E. A. SJOSTEDT.

Sault Ste. Marie, Ont., Canada.

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*This was already briefly referred to in an editorial note on page 583 of our December issue.—E.

†A translation of this report will be found on page 11 of this issue.—E.

Conservation and the Chemical Engineer.*

BY SAM. P. SADTLER.

We have heard much in the last year or two concerning the Conservation of our Natural Resources and we shall, I feel certain, hear much more in the next few years, as the facts elicited from the preliminary studies of the subject come to be understood by the public at large. The importance of the subject will grow correspondingly as the matter is studied by the thoughtful citizen and his appreciation of it will in time be reflected in the activity of the statesmen at Washington in the proposing of remedial measures.

Conservation let us note, however, represents the third stage in the history of the development of Natural Resources.

The first stage is Exploration or discovery. This is the era of the prospector and has given us in this country some famous episodes. We need only recall the discovery of gold in California in 1849 and the way in which it operated to attract adventurous spirits from the older parts of our country, or the repetition of the same story with the discovery of the rich gold deposits on the Yukon and at Nome in Alaska.

The first discovery of rich petroleum deposits of Western Pennsylvania in the early sixties brought, similarly, multitudes of prospectors or "wildcatters," as they came to be known locally, and this experience has been repeated also from time to time as great petroleum gushers or powerful gas wells have been reported in various sections of the country, resulting in the opening of new fields, as in Ohio, Indiana, Kansas, Texas and Oklahoma.

The second stage is Exploitation, when these lavish gifts of nature are worked with a view mainly of increasing production and usually in a wasteful way with no thought of the exhaustion of the supply.

As illustrations of this stage we need only cite the way in which our coal mines have been worked. In Bulletin 394 of the U. S. Geological Survey (Papers on the Conservation of Mineral Resources) we find the statement that "it has been estimated that the actual loss or waste sustained through coal left in the mines in conducting mining operations amounts to 50 per cent of the quantity produced and marketed." Worse than this, when the Anthracite Coal Waste Commission made its report in 1893 they estimated that "for every ton produced 1½ tons were lost."

One of our most valuable gifts of nature is the natural gas, which is associated more or less directly with petroleum. It is a fuel of the greatest value, being nearly pure hydrocarbon in its composition. Yet we find in the same Bulletin of the Geological Survey before referred to the following: "As to the amount of natural gas which is being wasted daily, no accurate statistics have been attempted and the judgment of Dr. I. C. White, State Geologist of West Virginia, may well be accepted to the effect that no less than 1,000,000,000 cubic feet of gas are wasted every twenty-four hours. Of this, undoubtedly the larger part is wasted in the production of oil." This waste, Dr. Day of the Geological Survey says, "practically equals the annual consumption of natural gas reported for 1907. This waste should furnish light for half the urban population of the United States."

The exploitation of our once great timber resources and the resulting denudation of great tracts of land and the evils to the soil which have followed in the train of this ruthless waste have been so graphically portrayed by the U. S. Forestry Bureau and others recently that I need not more than allude to this case of reckless extravagance.

It is not my intention, however, to take up this evening the question of the conservation of our natural resources and the absolute need thereof, as so ably developed in recent reports of the "National Conservation Commission," the Forestry Bureau, and other official publications, nor yet the part played by the chemist in this Conservation of Natural Resources which has been so ably reviewed by Dr. Bogert in his recent Presidential address before the American Chemical Society.

What I would like to do is to indicate that these same three stages of exploration or discovery, exploitation or effort at production, and finally conservation, are to be seen in the history of every great chemical industry and to point out, that, while it is the part of the chemical engineer to aid in the exploitation step, what is still more important is his part in counselling and indicating how the wholesome influence of conservation can be applied, so as to broaden and extend the scope of the industry, to maintain and add to its remunerative character and to give it stability and promise of permanence.

Let us illustrate this view by examples and take first the great petroleum industry to which reference has already been made in speaking of the era of exploration when the petroleum fields were first outlined by the work of the prospector and driller. The second stage of exploitation began to draw upon the help of those who were chemical engineers in fact if not by title. The development of the distilling and refining processes came first. Starting with the old cheese-box still with its circle of heating grates under it, there was a change, following upon the discovery of the "cracking process" as applied to crude oil, to the present form of cylindrical still with its movable cover to regulate the chilling of the vapors during the latter part of the operation. This simple feature in distilling enables the refiners to get 75 per cent or more of burning oil from the crude petroleum instead of the 45 per cent of a normal fractional distillation, while producing a residuum which can be advantageously distilled for paraffin oils.

The proper control of the acid and alkali treatment of the crude distillates, the change from the old forms of presses for paraffin scale to the modern filter press, the introduction of bone-black filtration for reduced oils and residuums all contributed to develop and expand enormously the several parts of the industry to which they were applied. With these improvements in large scale methods went the inauguration of improved testing and analytical methods, so that uniformity of product so essential for sound business development was secured.

Let us turn now to the newer evidence of the work of the chemical engineer in the way of conservation as illustrated in this same industry. The collection and utilization for fuel purposes of the uncondensed gas from the distillation of crude oil is one of the important economies that has been generally adopted. The working up of the sludge-acid and recovery of not only sulphuric acid but of valuable side products is now a feature of the larger refineries. The introduction of clay filtration to improve the quality of the heavier grades of oils is an important advance as well as a step of economy. The production of gas oils from the less valuable crude oils has also become an important industry as these are of great value for gas-making and gas-enriching. The more thorough utilization of residues is also a feature of recent years. From these residues are now made excellent road oils for the aid of the good roads movement. The petroleum pitch is all utilized also partly for electric light carbons and partly for fuel purposes. Most promising of all, however, is the result, not as yet fully attained, but most certain of solution in the immediate future, viz., the utilization of crude petroleum of the lesser valuable kinds and residuums in internal combustion engines for the development of power.

A second typical industry is that of coal distillation. We have already spoken of the wastage in the mining of coal. It is not necessary here to speak of the corresponding waste in its utilization as fuel. We will speak solely of the distillation of coal. This treatment may be carried out from either one of two points of view, and the exploitation in each case has been pushed with great energy, utilizing all available chemical and engineering skill. The first is the distillation for the manufacture of illuminating gas. While the mechanical side of this process has advanced steadily, particularly after the introduction of regenerative firing and mechanical stoking, the chemical side did not

*Presidential address delivered at the second annual meeting of the American Institute of Chemical Engineers in Philadelphia, Dec. 8, 1909.

advance so rapidly. While the coal-tar is no longer thrown away, unless it be in very isolated localities, the proper chemical utilization of this tar has lagged behind both in this country and in England. On the other hand, the true conservation of this valuable side-product and the development of its possibilities has advanced in a notable degree in Germany, the home of the coal-tar industry. Here the research chemist and the chemical engineer have gone hand in hand in the building up of a great industry or rather two industries based upon the utilization of the coal-tar, the manufacture of the coal-tar dye colors, and the manufacture of synthetic medical preparations possessing valuable therapeutic characteristics. Besides these most important and highly developed illustrations of conservation, we have, however, some minor utilizations of coal-tar or products from the same that deserve mention. Thus the manufacture of creosote oils for the preservative treatment of wood, the roofing-pitch and tar-paper manufacture and the use of pitch in the briquetting of coal are illustrations of value given to the coal-tar and its products.

The thorough extraction of the ammonia and the impurities like the cyanides from the ammonical liquor and the production of commercially valuable products from them is, moreover, an accomplishment in the way of conservation. The utilization, too, of the gas carbon for the manufacture of electrode carbons, battery plates and for electric light carbons is another illustration of this work.

The second method of distilling coal is that for the production of coke for metallurgical purposes. This has been developed or more properly exploited to such a degree that, according to the "Mineral Resources of the United States," published by the U. S. Geological Survey, the production of coke for the year 1907 was 40,779,564 tons, of which, however, 35,171,665 tons were produced in beehive ovens. In these, as is well known, only the fixed carbon of the bituminous coal is saved and all volatile constituents including gas, tar and ammonia are absolutely wasted. On the other hand, 5,607,899 tons of coke were produced in by-product recovery ovens and the value of the by-products (gas, tar and ammonia) obtained therefrom amounted to $7,548,71. It is easy to reckon from this what the loss was on the 35,171,665 tons of coke made in beehive ovens. In fact, the article on "Coal," in Bulletin 394 before referred to, says with reference to this, "at the prices which prevailed in 1907, the value of the by-products wasted in beehive coke-ovens was a little over $53,000,000."

But the result achieved by the chemical engineer in the working of this by-product over represents more than merely saving certain by-products. The gas produced can be separated by the perfect control of the method into "poor gas" for fuel purposes and "rich gas" for illuminating purposes so that the highest economy or conservation of values is thereby attained.

Another industry in which the chemical engineer has worked first for the purpose of exploitation and later for conservation of material and values is the starch industry. Starch, as we all know, is one of the most widely distributed vegetable products and has always played a great part in the world's supply of food. In Europe, it is potato and wheat starch, in the United States it is corn starch, and in tropical countries it is rice, tapioca, sago, etc., that are the important varieties of this cereal food. Not only has the production of starch been developed, however, for food purposes, but enormous quantities of the starchy substances serve as the starting point in the fermentation industries. Then we have the use of both starch and its alteration product dextrine in the textile industries and the production of glucose as the product of the hydrolysis of starch, and the manufacture of nitro-starch and its utilization in the explosive industry. All of these industries have attained a high degree of perfection by the application to them of a chemical understanding of the nature of starch and its alteration products and the devising of processes by which the several reaction changes could be carried out with exactness and economy. What that means as applied to one single branch of the starch industry, those of us who had the opportunity on the occasion of our meeting last June to go through the works of the Corn Products Company at Edgewater, N. J., can appreciate. The glucose production of the United States in 1907 is stated to have been 830,000 tons, requiring 40,000,000 bushels of corn as raw material. But besides the production of the solid grape-sugar and the liquid glucose for a great variety of uses, the separation of the germ of the corn from the starchy portion has made possible the production on a large scale of corn-oil, a product adapted for a wide range of uses—from soap making to the manufacture of rubber substitute.

Turning now to Inorganic Chemistry for an illustration, we have a splendid example of the work of the chemical engineer in the direction of conservation in the case of the natural and artificial nitrate industry, to which latter the Germans have already given the expressive name of "air-salpeter." The great source of nitrate for forty or more years past has been the deposits on the west coast of South America furnishing the so-called Chili salpeter or sodium nitrate. This has been drawn upon increasingly until in 1908 the quantity shipped was 1,730,000 tons, valued at $87,500,000. But the Chilean deposits are far from being inexhaustible. It is estimated that if the annual consumption increases merely by 50,000 tons, and this is to be reasonably expected, from 30 to 40 years will see the practical exhaustion of this supply. So in 1899, Sir Wm. Crookes startled the industrial world by calling attention, in what the newspaper men would call a "scare-article," to the nitrogen problem and the necessity of maintaining a supply of nitrogenous plant food if the world's food requirements were to be met. Crookes pointed out the way of relief when he said "the fixation of atmospheric nitrogen is one of the greatest discoveries awaiting the ingenuity of chemists. It is certainly deeply important in its practical bearings on the future welfare and happiness of the civilized races of mankind." The chemical engineer has responded nobly to this demand for conservation of available nitrogenous material by the working out of practical methods for the manufacture of what we called "air-salpeter." It must not be supposed, however, that success was easily obtained.

Not only were the usual experimental difficulties to be overcome but, as Prof. Bernthsen has well shown in his address before the London International Congress of Applied Chemistry in May last, there are theoretical difficulties of the most serious kind standing in the road of ready fixation of atmospheric nitrogen and oxygen in the form of nitrogen oxides convertible into nitrates. So it happened that the first large enterprise of this kind, the process of the Atmospheric Products Company, established at Niagara Falls, had to be given up as commercially unavailable. This was followed by the Birkeland and Eyde process, started in Norway with the cheapest water power to be found, and this is still in successful operation. A few years later (in 1905) the Schönherr process was worked out by the Badische Anilin and Soda Fabrik, also using Norwegian water power and still later the Pauling process at Gelsenkirchen, near Innsbruck, in Tyrol. The Schönherr process seems to be the most successful.

It produces a 40 per cent nitric acid, calcium nitrate, or calcium or sodium nitrite, according as the absorption part of the process is modified. All of the processes mentioned require the cheapest electrical energy which can only be developed by cheap water power, and thus far best developed in Norway. Prof. Bernthsen states, however, that probably within a few years the annual output of calcium nitrate or "air-salpeter" will reach 100,000 tons. As this involves first the fixation of atmospheric nitrogen, and second the use of "white coal," as water power is sometimes fancifully called, it is a true lesson in conservation of natural resources, especially as it also enables us to replace a rapidly disappearing natural product.

Closely related to this recently developed inorganic industry of air-salpeter is the slightly older one of calcium carbide and

its offshoot the cyanamide or "nitrolime" industry. With the production of calcium carbide in the electric furnace in 1892 by Willson and the publication of Moissan's work on the electric furnace in 1894, sprang into existence a great industry, as the acetylene gas lighting made possible thereby had great advantages. Isolated lighting plants, acetylene lamps for automobiles and carriages, luminous buoys and signals, a new material for lampblack manufacture and other utilizations all were rapidly developed. The perfecting of the furnaces and the process for this manufacture of carbide enlisted the attention of electrical and chemical engineers and at the present time the world's production of calcium carbide is estimated to be about 200,000 metric tons per annum, largely in countries like Norway, Italy and Switzerland where cheap water power is available, as well as in the United States at Niagara Falls and Sault St. Marie. This production, however, for the time being outran the demand for carbide for acetylene lighting.

Relief from over-production by finding new outlets and utilizations is always to be preferred to closing of works already in operation, so chemical engineers have found new possibilities for calcium carbide. By far the most important of these is the production from calcium carbide of calcium cyanamide by the action of nitrogen gas, as worked out by Drs. Frank and Caro. We have here an exothermic reaction in which nitrogen is absorbed by the carbide with the production of calcium cyanamide and carbon. This takes place at a temperature of from 800° to 1000° C., much below that needed for the carbide manufacture. Not only can all the nitrogen of this cyanamide be converted into ammonia by decomposition with steam, but it is gradually decomposed by the chemical and bacteriological constituents of the soil into ammonia, which becomes fixed by the vegetable mould and is so held. The cyanamide is also convertible into calcium cyanide by melting with fluxes, into dicyandiamide for dye-color manufacture, into urea, guanidine and other hitherto relatively expensive organic compounds. The dicyandiamide is already used as a "deterrent" in smokeless powder manufacture, reducing the temperature of the explosion without diminishing explosive force, and the crude calcium cyanamide with certain fluxes under the name of "ferrodur" is employed for case-hardening of iron and steel.

The statement is made that the works for the manufacture of "nitrolime" now in operation or in course of construction have a capacity of 166,000 tons per year.

The illustrations of conservation, whether from the point of view of better utilization of materials, or the production of new and varied products or the recovery of what were waste products, could be greatly extended did time allow.

We might refer to the way in which sulphur recovery has been worked out in the alkali industry, or to the manganese recovery in connection with the chlorine production from manganese dioxide, so that "recovered manganese" is to-day a most valuable article of commerce. Or we might note the saving resulting from the manufacture of reclaimed rubber, but these and similar illustrations of the conservation theme as influenced by the work of the chemical engineer will have to be passed by for the present.

That there are numerous problems as yet unsolved of equal and possibly greater importance than some of these discussed will also be conceded by those possessing even a moderate acquaintance with chemical industries.

One of these problems, for instance, is the recovery of the valuable constituents of the waste liquor of the sulphite wood-pulp process. A German authority states that every liter of this waste liquor contains 120 gms. of organic material as against 10 to 15 gms. of mineral substance. Dr. A. Frank estimated that in 1904 there was wasted in this way in Germany 300 million kilograms of organic material, concerning which we know that it has value in several directions. In this country, the sulphite wood pulp process has also an extensive development and the same waste liquor is run off into our streams.

A somewhat different problem, but one of even greater importance, is the loss of valuable metals in smelter smoke and fumes. The American production of bismuth is not over 10,000 lbs. a year and considerable amounts of bismuth and bismuth compounds are imported every year. Yet it is estimated in the forthcoming report of the U. S. Geological Survey for 1908 that 880 lbs. of bismuth per day are being thrown off in the smoke of the great Washoe smelter at Anaconda, Mont., and with this also go copper, lead, zinc, arsenic and other mineral products. One way of saving much of this lost material is pointed out in noting that the replacing of smelting methods by electrolytic methods, possible in some cases, allows these metals to be recovered from the deposited slimes.

In conclusion let us emphasize the fact that our chemical industries need not merely development or exploitation, but if they are to continue to flourish as we are drawn more and more into international competition they must have the newest and best results of chemical research applied by the experienced chemical engineer. New and better materials must be sought, better processes evolved, economies effected at all possible stages, and waste products carefully looked after. The most successful chemical industries in the world, which show the highest scientific and technical development and which are steadily expanding and throwing out branches, are those of the great German chemical companies. Organizations like the Badische Anilin and Soda Fabrik, which have by their employment of both research chemists and chemical engineers effected industrial revolutions, one after the other, like the introduction of artificial indigo, the contact sulphuric-acid manufacture, and last of all the production of air-salpeter or artificial nitric acid and nitrates, are those which reap the rich commercial rewards and for the reason that they have done the work and earned them. The hope of a successful American Chemical Industry lies in the same direction.

The Electric Shaft Furnace at Domnarfvet, Sweden.

A full and interesting account of the experiments made with the electric shaft furnace at Domnarfvet, Sweden (see our Vol. VII, p. 419), and the conclusions drawn therefrom has been contributed by Mr. LARS YNGSTRÖM to the *Jern-Kontorets Annaler*, No. 9, 1909, from which the following abstract has been made.

The author first makes a heat analysis of the smelting process and determines the amount of coal and electric energy necessary to make a given grade of pig iron, which is assumed to have the following composition: Three per cent C, 1 Si, 96 Fe, and traces of Mn, P, S, Cu.

Further, it is assumed that the ore consists of Fe_3O_4 with 60 per cent iron plus burnt lime, and that the escaping gases contain 30 per cent CO_2 and have a temperature of 200° C.

In making the computations, the following heat values are assumed:

To reduce 1 kg Fe from Fe_3O_4 requires 1650 cal.
" 1 " Fe " Fe_2O_3 " 1800 "
" 1 " Si " SiO_2 " 7830 "

To oxidize 1 kg C to CO_2 generates 8080 cal.
" 1 " C to CO " 2470 "

1 kw-hour corresponds to 857 cal.

The smelting and overheating of 1 kg of pig iron requires 280 cal.

1 kg. of slag[1] requires 595 cal.

Specific heat for dry furnace gas, 0.245 cal. per kilograms.

It is also assumed that all raw material introduced into the furnace has an initial temperature of 0° C., that the ore is free of water and CO_2, and that the lime is burnt.

[1] This figure may appear somewhat high, but it was thought advisable to increase it by about 100 cal. over the value obtained by experiment in the laboratory; partly on account of the high temperature of the slag and partly on account of the formation of carbide in the slag when electricity is used.

Amount of C required per 1000 kg of pig iron.

Reaction formulas:

(a) $13\ Fe_3O_4 + 40\ C = 12\ CO_2 + 28\ CO + 39\ Fe$.

(b) $13\ SiO_2 + 20\ C = 6\ CO_2 + 14\ CO + 13\ Si$.

According to formula (a) 960 kg of Fe are obtained from 1325.71 kg Fe_3O_4 and requires for reduction 210.99 kg. C, forming 576.70 kg of gas ($CO_2 + CO$).

According to formula (b) 10 kg Si is obtained from 21.43 kg SiO_2, and requires for reduction 6.59 kg C., forming 18.02 kg of gas ($CO_2 + CO$).

The total amount of carbon required is:

For the reduction of 960 kg Fe	210.99 kg
For the reduction of 10 kg Si	6.59 "
For carbon in the pig iron	30.00 "
Total chemically pure carbon	247.58 kg

Amount of Slag.

With a mixture of ore and lime containing 60 per cent Fe there is required 1600 kg of this mixture for 960 kg Fe. This corresponds to 1325.71 kg Fe_3O_4.

Corresponding amount of slag	274.29 kg
SiO_2	21.43 "
Amount of slag per ton of pig iron	252.86 kg

Amount of Gas.

According to form. (a) there is developed $CO_2 + CO$ . . 576.70 kg

According to form. (b) there is developed $CO_2 + CO$ . . 18.02 kg

Total per ton of pig iron . . . . . 594.72 kg

Calorific Value of Fuel.

When 1 kg C combines with O, forming a gas which contains 30 per cent CO_2 by volume, there is generated $0.3 \times 8080 + 0.7 \times 2470 = 4153$ kg-cal.

Heat Balances.

Heat required:

Reduction of 960 kg Fe from Fe_3O_4 at 1650 cal. . . .	1,584,000 cal.
Reduction of 10 kg Si from SiO_2 at 7830 cal. . . .	78,300 "
Melting and overheating of 1000 kg of pig iron at 280 cal.	280,000 "
Melting and overheating of 252.86 kg of slag at 595 cal.	150,452 "
Heating of 594.72 kg $CO_2 + CO$ to 200° C. at 200×0.245 cal	29,142 "
Total	2,121,894 cal.

Heat developed:

Combustion of 217.58 kg C at 4153 cal.	903,610 cal.
Heat developed by electric current	1,218,284 "
Total	2,121,894 cal.

Therefore, the theoretical amount of electric energy required is

$$\frac{1,218,284}{857} = 1420 \text{ kw-hours.}$$

The curves in Figs. 1 and 2 are based upon calculations similar to the foregoing. The amount of carbon in coke and charcoal being assumed as follows:

Coke { C = 85 per cent; H_2O = 5 " "; Ash = 10 " "

Charcoal { C = 85 per cent; H_2O = 12 " "; Ash = 3 " "

Fig. 1 applies to iron reduced from Fe_3O_4, while Fig. 2 applies to that reduced from Fe_2O_3.

The experiments at Domnarfvet began in the spring of 1907, and with but few interruptions have continued up to the present time. The electrical equipment installed especially for these tests consists of a 3000-volt, 60-cycle, 900-hp synchronous motor direct connected to a three-phase 25-cycle generator which supplies energy to a transformer set up near the furnace. The generator permits of an adjustment of e.m.f. in small steps from 300 to 1200 volts. This wide range of voltage has made it possible to test furnaces under various conditions and of various designs with the most favorable voltage. The fields of the synchronous motor and generator are excited by a 20-kw, 220-volt, direct-current generator mounted on the same shaft. The set is started by a 500-volt, three-phase induction motor also mounted on the same shaft. The machines are all equipped with instruments which permit the calculation of the energy and power being used.

From the switchboard the electric energy is carried by copper bars laid in conduits under the floor direct to the three-phase air blast transformers. These transformers have a combined capacity of 1500 kilovolt-amperes with a transformation ratio of 14 to 1—that is, by means of voltage control at the generator, the secondary voltage of the transformer can be changed in small steps from 20 volts to 80 volts.

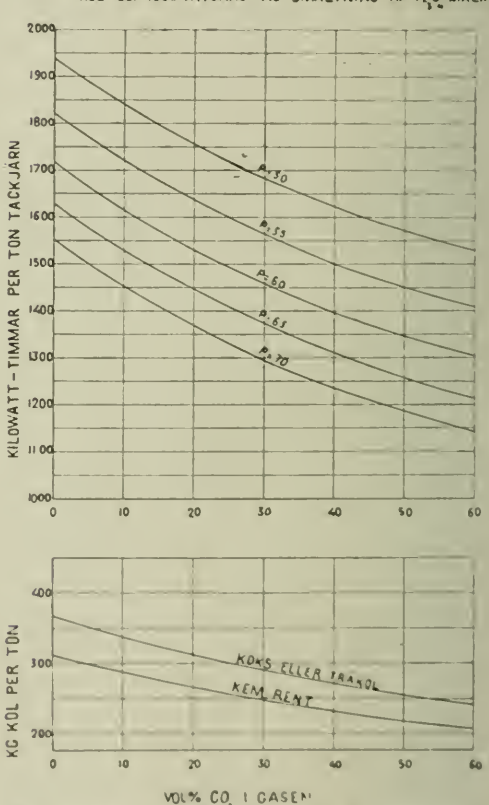


FIG. 1.—ENERGY AND CARBON REQUIRED FOR SMELTING IRON FROM Fe_2O_3.

(The abscissas in both diagrams are per cents. by volume of CO_2. The ordinates of the upper diagram are kilowatt hours per ton of pig iron. The five curves refer to 50, 55, 60, 65, 70% Fe in the ore. The ordinates in the lower diagram are kg. per ton. The upper curve refers to coke or charcoal, the lower to chemically pure carbon.)

For controlling the operation of the furnace an instrument board is mounted nearby which contains one three-phase precision wattmeter for unbalanced phases; three ammeters, one for each phase, and one voltmeter with switch for connecting to various phases.

The ammeter and wattmeter are connected through series transformers to the leads. The voltmeter and wattmeter potential coil are direct connected to the leads. Right under the instruments are placed the hand wheels for controlling the position of the electrodes in the furnace.

The first experiments were performed with an electric furnace constructed in accordance with certain patents owned by Messrs. Grönwall, Lindblad and Stalhane, who together formed a company Elektrometall in Ludvika. These gentlemen, together with the Grangesbergs Company, were also interested in the continued experiments.

The construction of the first furnace is shown in Fig. 3. It

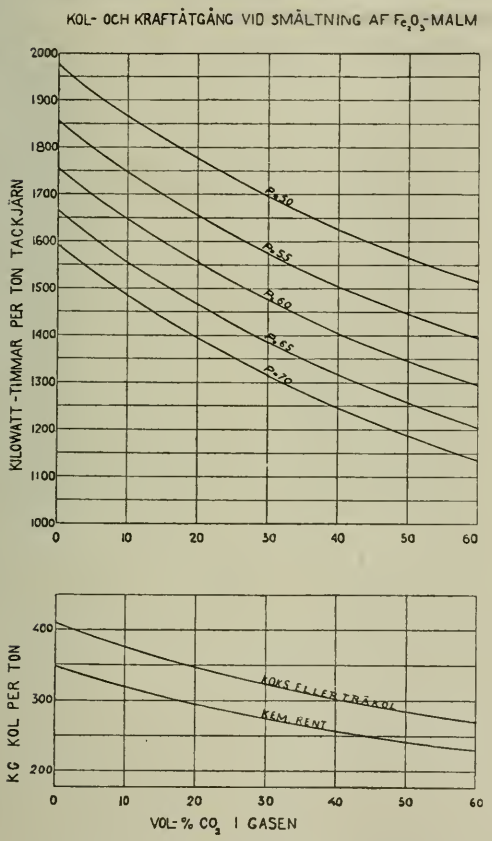


FIG. 2.—ENERGY AND CARBON REQUIRED FOR SMELTING IRON FROM Fe_2O_3.

(The abscissas in both diagrams are per cents. by volume of CO_2. The ordinates of the upper diagram are kilowatt hours per ton of pig iron. The five curves refer to 50, 55, 60, 65, 70% Fe in the ore. The ordinates in the lower diagram are kg per ton. The upper curve refers to coke or charcoal, the lower to chemically pure carbon.)

is a shaft furnace with the hearth of tamped quartz. In the bottom of the hearth there are large grooves. The one in the middle leads to the tap hole, while the other two on the sides lead to wells which are filled with iron. The bottoms of these wells are made of copper plates covered with pressed graphite blocks, each of these forming the electrodes of the furnace.

The furnace is started with blast as in the ordinary shaft furnace, and when a sufficient amount of iron has collected in the hearth, the blast is turned off and the electric energy turned on. It was intended that the electric current entering the furnace through the iron on the one side, passing through the furnace on the other side would, partly by the overheating of the molten iron and partly by the energy lost by the resistance in the path, liberate enough heat to melt the ore.

For short periods of time this furnace could operate, but it was found impossible to obtain a hearth bottom which could stand the high temperatures. Therefore, this furnace was rebuilt in the form shown in Fig. 4. The same principle is used

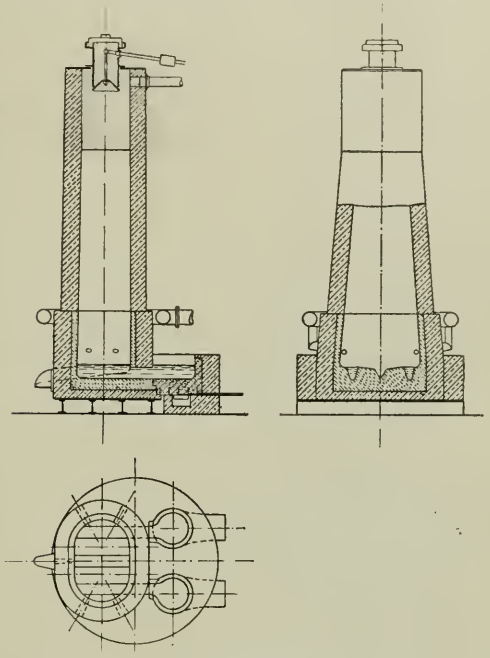


FIG. 3.—FIRST DESIGN OF ELECTRIC SHAFT FURNACE. (TWO BOTTOM ELECTRODES.)

as in the first furnace, but the lower part is more solidly constructed. The electric energy enters and leaves at opposite sides of the furnace. The most important changes consisted in using magnesite. This material gave very much better results than were obtained in the first furnace. However, it had the disadvantage of being a relatively good conductor of electricity at very high temperature.

These first experiments clearly indicated that the solution of the problem could not be accomplished with furnaces where the current was introduced into the molten iron at the bottom of the hearth. Therefore, the next furnace was built along different lines.

The first one of this type, shown in Fig. 5, was a small shaft furnace with one graphite electrode in the bottom, and two diagonally opposite electrodes in the walls of the furnace at a certain height above the bottom. During the experiments the upper electrodes became considerably burnt and had to be exchanged for ordinary adjustable carbon electrodes fitted with water-cooled jackets The connections were so made that the current could be conducted either across the shaft from one electrode to the opposite one, or through the bottom electrode. This furnace proved much more durable than the former ones, but had the disadvantage that the walls of the shaft were badly damaged around the electrodes, due to the very large amounts of energy there liberated.

If it could be so arranged that the electrodes were surrounded by ore, the heat which in this furnace was lost in heating up the walls, could be turned to good account, and the walls made

to last a great deal longer. In accordance with these ideas a furnace was constructed as shown in Fig. 6, and experiments with this one showed that the problem was very near its successful solution and led to the construction of the large shaft furnace which forms the subject of this article. The construction of this furnace is shown in Fig. 7, which in appearance is very similar to an ordinary blast furnace in which the tuyeres

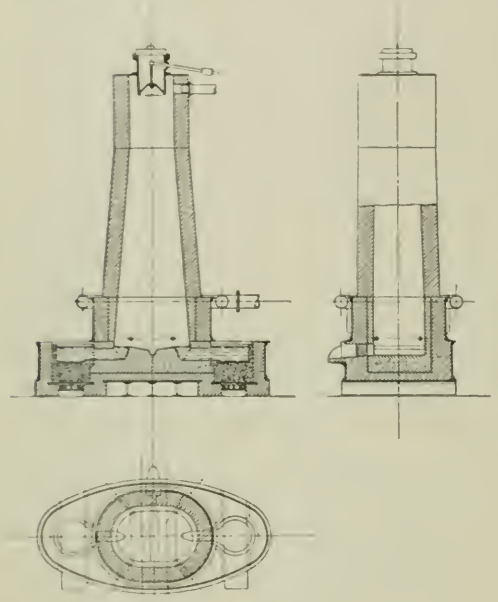


FIG. 4.—SECOND DESIGN OF ELECTRIC SHAFT FURNACE. (TWO BOTTOM ELECTRODES.)

are replaced by electrodes. This furnace was described by Dr Eugene Haanel in a paper before the Electrochemical Society, May 6, 1909 (this journal, Vol. VII, p. 251, June, 1909).

The melting chamber is in the form of a crucible about 2,250 mm in diameter and 1500 mm high and lined with magnesite. As the charge descends through the opening in the top of the melting chamber, free space is left between the charge and the upper walls. This point is of great importance and has much to do with the success of this type of furnace. Experiments have shown that the angle assumed by the descending charge is about 50° or 55° with the vertical.

This free space around the roof of the chamber is utilized for cooling the walls and electrodes by drawing comparatively cool gas from the tunnel head and forcing it through tuyeres into the space at the top of the crucible, as shown in the illustration. This method of cooling is extremely efficient, since the heat absorbed by the gases is given off to the charge as the gases ascend to the top of the shaft, and, therefore, no heat is lost except by radiation from the tuyeres. By this method of circulation, of gas a better distribution of heat throughout the charge is obtained than is usual with other types of electric furnaces.

The electrodes are introduced through the roof of the melting chamber passing through the free space into the charge, which is made up of two square carbons 33 cm on a side, having a total cross-section of 2178 sq. cm. The electrode holder consists of a strong steel frame which supports wedges by means of which the copper plates conducting the electricity from the cables are pressed against the faces of the electrode. The position of the electrodes is controlled by the steel cable which passes by means of pulleys to a winch situated at the instrument board opposite the furnace. The external part of the electrodes is protected with an armored asbestos covering to prevent oxidation in the air, and the stuffing boxes are water-cooled, special devices, not shown in the illustration, being provided to prevent the gas within the melting chamber from leaking out around the electrodes.

The shaft, which is 5.2 meters high and 1.525 meters internal diameter at the largest section, rests upon an iron framework supported by six cast-iron columns. This construction permits repairs to be made on the melting chamber without in any way disturbing the shaft itself.

This furnace was put into operation the seventh of last May, and with the exception of a few slight interruptions was run continuously to last July, and could have continued for an indefinite period had it not been necessary for other reasons to shut it down.

The furnace is started as an ordinary blast furnace with the electrodes removed, then several charges of charcoal and coke are put in, the electrodes are inserted and the shaft filled with ore and lime, together with charcoal and coke. The amount of charcoal used is reduced continually and at the end of the first week the charges consist of 100 kg ore, 3 kg lime (burnt),

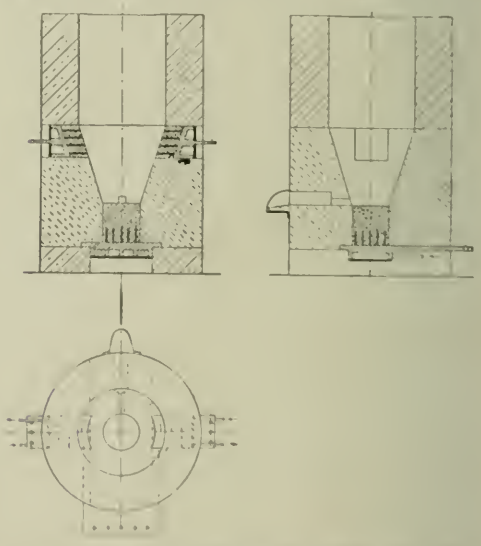


FIG. 5.—THIRD DESIGN OF ELECTRIC SHAFT FURNACE. (ONE BOTTOM ELECTRODE AND TWO ELECTRODES IN WALLS.)

24 kg coke, and from that time on 100 kg of ore, 4 kg of lime and from 22 to 24 kg coke. The greatest number of charges per day when operating with coke was 62. The shaft held 72 charges, and, therefore, the number of times it was filled per day was 0.86.

When operating with charcoal the number of charges per day was 80, and since the shaft then held only 41, the number of times the shaft was filled per day was 1.95. When using charcoal alone the amount of carbon per 100 kg of ore varied from 21 to 28 kg. The charcoal was measured by volume, 1 hektoliter being taken as 16 kg, because of its relatively high moisture content.

At the beginning, when the furnace was still cold in the bottom, the current existed principally under the arch. The path of the current followed close under the arch, and, therefore, the latter was considerably heated and its surface partially melted. As soon as the gas circulation was put into operation

its cooling effect was very effective, the temperature of the arch sunk and the heat descended toward the bottom of the crucible. As the lower part became heated, the resistance was lowered, and, therefore, the current gradually worked down toward the bottom of the furnace. After the furnace was in normal operation, the gas circulation could be discontinued for several days without any tendency for the current to rise toward the roof of the crucible. Charcoal in the charge seems to assist in holding the heat near the bottom.

In several cases it happened that the quantities of iron produced did not correspond to the percentage in the ore. For example, in one week from 48 tons of ore only 17 tons of iron, which corresponds to 27 tons of ore, was obtained; while in the following week from 22.6 tons of ore, 23 tons of iron were obtained. This can be explained only by the fact that more or less of the reducing iron clung to the walls and bottom of the furnace in the first case, and was drawn off in the second.

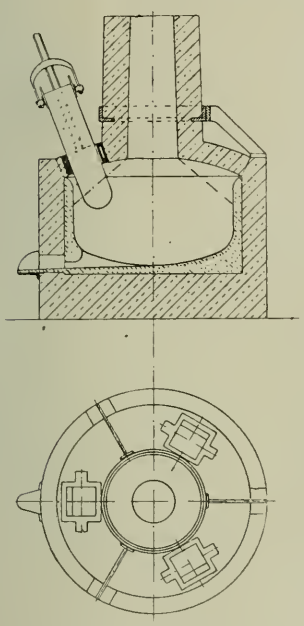


FIG. 6.—ELECTRIC SHAFT FURNACE. (THREE ELECTRODES IN WALLS.)

The furnace was sometimes cold on account of there being too much carbon in the charge. However, this was quickly overcome by increasing the amount of ore. Moreover, such an occurrence can be easily prevented by proper construction of the furnace walls. The slag indicates plainly an excess of carbon in the charge, in which case calcium carbide is formed and an odor of acetylene gas is noticeable.

In certain cases the furnace showed a tendency to hang, but after the amount of charge along the walls had reached 80 per cent of the total this difficulty disappeared. Experiments proved that an electric furnace can be charged with ore dust if all the dust is introduced along the walls and the height of the charging head is so chosen that the upper surface comes somewhat under the gas-collecting cylinders so that the dust cannot be drawn over into the gas tuyeres. The briquetting of ore dust is, however, not necessary with electric furnaces.

A few slight mishaps during the operation of the furnace had a certain influence on the results obtained. For example, near the end of the first week the water jacket of one of the electrodes sprang a leak so that it was necessary to operate the furnace for a time with only two electrodes. Again, on account of failure of one of the electrical machines, the transformer at two different times was connected directly to the high-tension line and operated at 60 cycles instead of 25.

While in operation no surges of current or sudden peaks were noted. Even when tapping the furnace the measuring instruments indicated no fluctuation, and, therefore, the electrodes required very little attention, being regulated about once a day, and in one case they ran five consecutive days without any adjustment. The largest current carried by the electrodes was about 9000 amp per phase. The power factor at 25 cycles was from 0.8 and 0.9, while at 60 cycles it was about 0.7.

The amount of energy which can be delivered to the furnace depends largely upon the internal resistance of the mass; the larger the resistance, the greater the amount of energy which can be developed without increasing the current to excessive values. With 25 cycles, 80 volts was sufficient to produce the proper current, and with 60 cycles, 65 volts was necessary. The voltage impressed upon the furnace depends principally upon the fuel used and its relation to the ore. Table 1 shows this relation.

TABLE 1.

Charge—	Volts across electrodes	Current per electrode in amperes
Coke in excess	34	9,600
" not in excess	36	8,800
Charcoal in deficiency	60	6,300
" right	54	7,600
" in excess	48	7,600
Coke and charcoal in excess	35	9,200
" " " right	48	7,600

In operation the furnace proved to be very steady and easy to care for, and it was tapped about every six hours.

Seven different kinds of charges were used in the tests. Since the principal purpose of the experiments was to develop

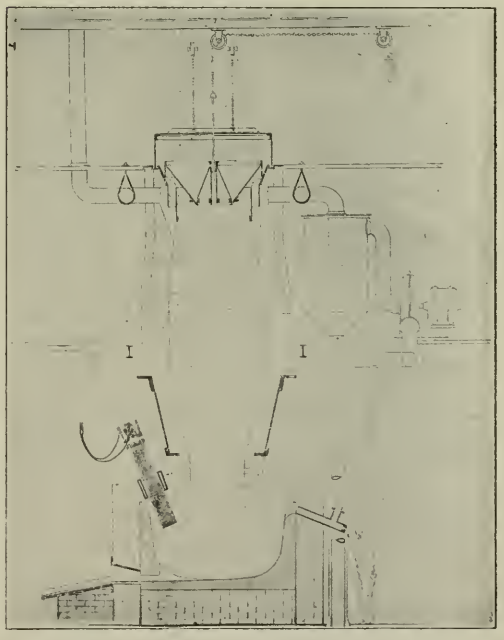


FIG. 7.—FINAL DESIGN OF ELECTRIC SHAFT FURNACE.

a practical furnace construction, the charge at first consisted of a rich, easily reduced hematite ore which had a considerable phosphorus content. As a result, however, later on it was impossible to produce a first-class pig iron from first-class ore, because the iron was always adulterated by phosphorus from the ore which remained on the walls of the furnace.

Burnt Lime.		Coke.		Charcoal.	
Fe_2O_3	0.5%	C	85%	C	65-80%
MgO	2.0%	H_2O	4%	H_2O	12-20%
CaO	96.5%	O + N	1%	Ash	3%
Al_2O_3	0.2%	Ash	10%		
SiO_2	0.2%	(0.5-0.7% S)			

The composition of charge mixtures was as follows:

Mixture No. 1.

826 charges of
100 kg Grängesbergs Skarnings ore
4-5 kg lime (burnt).
10-26 kg coke.
10-0 charcoal.

Mixture No. 2.

575 charges of
10 kg of Langgrufve ore (briquettes)
11-18 kg of lime (burnt).
0-21 kg of coke.
20-0 kg of charcoal.

upon the amount of this carbon which is oxidized to CO_2. In short, the amount of carbon used under normal conditions is always the same as would be theoretically required. The results of experiments demonstrate the truth of this statement. The carbon required can be expressed mathematically in terms of the CO_2 in the furnace gases. If M denotes volume per cent CO_2 in the gases, and k the percentage of carbon in iron, the carbon required in kilograms per metric ton of pig iron, if iron occurs in Fe_2O_3, is:

$$C = \frac{321.43\,(100 - k)}{100 + M} + 10\,k.$$

If Fe occurs in Fe_3O_4, we have

$$C = \frac{285.7\,(100 - k)}{100 + M} + 10\,k.$$

Table 2.—Ore Analyses.

Ores from—	Fe_2O_3	Fe_3O_4	FeO	MnO	MgO	CaO	Al_2O_3	TiO_2	SiO_2	P_2O_5	H_2O +CO_2	Fe	P	S	Mixture No.
							%								
Grängesberg:															
Skärnings ore	13.75	71.65	—	0.12	0.74	6.80	0.35	—	3.02	4.19	—	60.11	1.83	trace	1 and 3
Langgrufvan:															
Briquettes	5.89	76.36	—	0.24	2.66	3.28	—	—	11.52	0.009	—	57.70	0.004	trace	2
Magnetically separated dust	80.89	—	—	0.13	2.80	2.96	2.14	—	11.30	0.009	—	58.57	0.004	trace	6
Johannisberg	58.58	—	6.78	5.58	5.61	4.98	0.91	—	5.56	0.007	11.72	47.69	0.003	0.13	6
Skottgrufvan															
Vask ore	66.37	—	2.16	0.33	3.19	12.18	2.46	—	4.50	0.037	8.44	49.73	0.016	0.00	6
Tuollavaara	92.46	1.99	—	trace	0.03	1.28	trace	0.90	1.80	0.007	—	68.31	0.003	0.00	4 and 5
Taberg	74.48	—	—	0.42	4.96	4.90	—	—	15.30	0.014	—	53.93	0.006	0.00	5
Finnmossen	79.64	—	—	0.29	3.78	3.30	4.31	—	8.55	0.023	—	57.66	0.010	0.00	5
Nordmark	68.58	—	—	0.60	5.50	6.27	1.95	—	16.90	0.023	—	40.61	0.010	0.00	5
Brazil:															
Hematite	—	91.72	5.78	trace	0.25	0.20	trace	—	0.80	0.126	1.0	68.8	0.055	0.03	7

Mixture No. 3.

959 charges of
100 kg Grangesbergs Skarnings ore.
2-4 kg of lime (burnt).
8-13 kg of coke.
13-7 kg of charcoal.

Mixture No. 4.

599 charges of
100 kg Tuollavaara ore
4 kg of lime (burnt)
7-11 kg of coke
17-11 kg of charcoal.

Mixture No. 5

623 charges of
20 kg Tuollavaara ore
80 kg equal parts of Finnmossen, Tabergs and Nordmarks ore.
5 kg of lime (burnt)
4-0 kg of coke.
18-26 kg charcoal

Mixture No. 6

No. of Charges	Langgrufve Ore	Skottgrufve Ore.	Johannisbergs-grufve Ore	Lime	Charcoal
92 parts kg	50	20	30	6	21
13 " "	50	20	30	[illegible]	21
168 " "	35	15	20	[illegible]	16
160 " "	45	10	15	2	16
202 " "	55	10	10	2	16
81 " "	60	10	5	3	16
96 " "	75	—	—	4	16

Mixture No. 7.

[illegible] charges of
64 [illegible] kg Hematite ore
2-5 kg Lime (burnt)
16 kg charcoal

[illegible] air and enter the furnace, providing it is well built and properly run, the amount of fuel for different ores depends entirely upon the amount of carbon in the fuel and also

These formulas hold for chemically pure carbon and gas which is made up exclusively of CO_2 and CO.

It is evident that the circulation of the furnace gases has a definite influence on the amount of CO_2 in the gases. The experiments showed that the CO_2 content increases with the gas circulation, and that the temperature of the gases is very considerably increased. For instance, when operating with mixture No. 3, the CO_2 content of the gas was increased from 14.5 with no gas circulation to 39 per cent with constant circulation, and the corresponding temperatures were 75 deg. C. in the first case and 220 in the second. The experiments also indicated the existence of a certain circulation velocity which was best in each particular case, and that an increase over this velocity increased the temperature of the gases accompanied by a decrease in CO_2 content which was quickly noticeable.

What velocity to use in each case must be determined by experiment, and will depend to a certain extent upon the volume of the shaft. No exact measurement was made of the flow of gas used in these tests, but it may be estimated as about four or five times the rate at which the gases were generated in the furnace.

The high hydrogen content of the gas is noteworthy. This is partially due to the hydrogen in the charcoal, but to a greater extent to the moisture in the fuel, because when the moist gases are blown into the lower part of the furnace where the temperature is very high, hydrogen is developed.

A question which assumes great importance when judging the economical results which can be obtained with electric furnaces is that of the life of electrodes. With mixtures Nos. 1, 2 and 7 the consumption of the electrodes, on account of certain disadvantageous conditions, was abnormal; therefore, in the results given in Table 3 these tests have been omitted.

The electrodes were of Swedish make, and consisted of retort carbon with a permissible current density of 4 amp per sq cm. The experiments indicate that the figures for the burning of the carbon can be reduced by about 30 per cent by using a higher quality of electrode.

Still more certain is the fact that the electrode consumption can be greatly reduced by suitable design. For instance, the electrodes used in the experiments were only about half consumed when they had to be discarded on account of their insufficient length. Therefore, if some construction be devised which would permit the continuous feeding of the electrode a great saving would be accomplished.

Of great importance is the electrical efficiency of the furnace. By electrical efficiency is understood the ratio between the electrical energy used and the amount of energy required to reduce the ore. The electric current in the furnace produces the following effects: (1) It raises the mass to the proper temperature. (2) Melting and overheating of pig iron and slab. (3) Driving off of H_2O and CO_2. (4) Heating of the gases. The sum of these quantities of energy, less the heat which is developed by reduction of carbon by oxygen in the ore, to a gas of greater or less CO_2 content, is just the amount of energy which theoretically must be supplied from the electric circuit, and which is denoted as the theoretical electric energy. The losses of electric energy are as follows: (1) Heat in the cooling water around the electrodes. (2) Heat in leads and contacts. (3) Radiation from the walls of the furnace.

In tabulating the results obtained with the various mixtures, Nos. 1, 2 and 7 have been omitted, because during the first two runs the furnace was not yet in normal operating condition, while the last run was too short to give consistent and reliable results.

TABLE 3.

Mixture No.	Electrode burning. Kg. per ton	Electrode consumption. (metric) pig iron.
3	1.4	11.1
4	7.0	12.7
5	8.1	14.3
6	6.9	19.4
3—6	5.2	13.8

The losses varied between 230 and 270 kw, and were higher near the end of the experiments than in the beginning. These losses may be divided approximately as follows:

Cooling water	118 to 125 kw
Loads in contacts	40 kw
Radiation	110 to 180 kw
Total losses	239 to 275 kw

By building furnaces of higher capacity, for instance of 2500 hp, and by paying particular attention to the factors which have a bearing on the heat losses, it is not impossible that a furnace can be built with an electric efficiency of about 80 per cent—that is, one which surely could produce more than three metric tons of pig iron every per horsepower year, provided a rather rich ore is used.

It is difficult to estimate from the experiments which have been made what the maintenance cost of a furnace would be. The present furnace, however, has been in continuous operation for 85 days, and successfully met all requirements which indicate a durable furnace.

The furnace's weakest point is the arched roof of the crucible, but by cooling with circulating gas this can be prevented from burning out. The bottom of the crucible and the walls of the shaft did not show any appreciable ill effects from the heat more than the ordinary shaft furnace. It has happened that the masonry back of the electrodes became red hot, but by suitable cooling devices, utilizing circulating gas, this can easily be overcome. Moreover, if the masonry should burn out at these points, it is easy to repair them without interfering with the operation of the furnace. Therefore, by suitable design and proper material in the walls, the durability of the furnace can be rendered thoroughly satisfactory.

With electric furnaces it appears that the process of producing iron of different compositions is under more exact control than in other processes, since it consists mostly in regulation of voltage, which will produce corresponding changes in temperature. It has been possible to produce iron with less than 1 per cent carbon, but in this case the slag content had a noticeable amount of iron. As a rule, the carbon content varied between 2 and 3 per cent.

With an electric furnace it is very easy to increase the silicon content in the pig iron. Analysis has shown 3.61 silicon, but it is also possible to reduce the silicon content as low as from 0.3 to 0.5 per cent.

It can be counted on that all the phosphorus in the charge will enter into the composition of the iron, but it should be noted here that in electric furnaces only one-third as much fuel is used as in the ordinary shaft furnace, and that in the production of first-class charcoal pig iron it is the charcoal which supplies the phosphorus. Therefore pig iron produced in an electric furnace will have a lower phosphorus content than that produced in an ordinary blast furnace.

A special advantage of the electric furnace is the possibility of driving off sulphur. Using a mixture of about 5 per cent sulphur, an iron with less than 0.005 per cent sulphur was produced. The requirements for driving off of sulphur are as in electric steel furnaces, a basic slag and a high temperature. The calcium carbide formed by electric current also plays an important part in the driving off of the sulphur.

Judging from the tests made at Domnarfvet, it appears that the production of iron from iron ore in electric furnaces is successfully accomplished, both technically and economically speaking, but in order to obtain a more complete knowledge of the subject, these experiments should be continued on a larger scale. If electric operation is to be worth while, however, sufficient electrical energy at low price must be available. Summing up the advantages of the electric shaft furnace over the ordinary blast furnace, they are as follows:

1. Lower installation cost, since blowing engines and heating apparatus are not required.
2. Saving of about two-thirds of the fuel required in an ordinary blast furnace.
3. Ore dust can be used without briquetting.
4. The generated gases have a high heat value when free from nitrogen.
5. The possibility of producing in the furnace itself low-carbon iron.
6. Less attendance, since the transportation and charging of two-thirds of the fuel used in ordinary shaft furnace is done away with.

Evidently it would be advantageous to combine electric shaft furnace for iron ore reduction with an electric steel furnace, which would refine the melted iron from the electric shaft furnace.

We are obliged to Mr. Yngström for a copy of his paper from which this translation was prepared, and to the editor of *Jern-Kontorets Annaler* for placing the cuts of the illustrations at our disposal.

TABLE 4.

Mixture No.	Carbon required (chem. pure).	Energy required calculated from amount of carbon, kw-hr per metric ton.	Actual energy used, kw-hr per metric ton.	Electric efficiency.
3	252	1470	3114	0.47
4	254	1438	2473	0.58
5	284	1741	3245	0.54
6	294	1870	3334	0.56

Rapid Method for the Determination of Chromium in Chrome and Chrome-Nickel Steel.

BY N. M. RANDALL.

Dissolve one gm. steel in 25 cc nitro-sulphuric acid. When in solution remove from the heat and add about 15 or 20 cc cold water. Rotate the liquid in the beaker and drop in at once about one gm. sodium bismuthate and continue to whirl for a few seconds.

Replace on the light and boil hard, the permanganate formed

by the sodium bismuthate is rapidly decomposed and there remains a clear violet liquid (manganic metaphosphate) which on further boiling converts the remainder of the chromium to CrO_3.

Decompose excess of manganic metaphosphate with 1/2 cc or more if necesary of dilute HCl; boil one minute, cool somewhat, dilute to 200 cc with cold water, add slight excess ferrous ammonium sulphate, and titrate with permanganate.

The end reaction is very sharp and leaves no doubt in the operator's mind.

The results are accurate and an analysis can be completed in 10 minutes.

This method has not yet been tested in presence of W and Mo.

Acid for dissolving steel:

300 cc nitric acid, 1.42 sp. gr.
300 cc sulphuric acid, 1 pt. acid, 3 pts water.
300 cc water.
100 cc phosphoric acid, 85 per cent
1½ grams manganese sulphate.

Sparrows Point, Md.

Uniformity of Blast Furnace Operation.

By Jos. W. Richards.

One of the great essentials for the attainment of high efficiency of furnace operation is the careful striving of the engineer for uniformity of working conditions. In practice it is the irregularities which fetch the superintendent out of bed at midnight, keep him on duty 24 or 48 hours at a stretch, and unduly sprinkle his hair with silver. What engineer has not experienced the sense of restful satisfaction which comes from seeing a furnace at last operating regularly, and has not felt at the same moment the conviction that his entire future problem was the preserving of those conditions by uniformity in everything pertaining to the furnace and its operation?

The history of blast furnace practice, when intelligently understood and properly appreciated, is mostly a record of efforts to attain greater uniformity of operation. The directions which these efforts have taken are in regard to the quantity, dryness and temperature of the blast; the composition, size of lump and relative proportions of the fuel, flux and ore; the times of charging, and the distribution of these within the furnace; the running out of pig iron and slag; the quality and temperature of iron and slag; the preserving constant of the interior shape of the furnace. We will discuss each of these in detail.

Composition of ore, flux and fuel.—No furnace can run successfully or uniformly unless the materials charged are of nearly uniform composition, both individually as ore, flux and fuel, and collectively as charge. To this end managers prefer to have large stocks of supply of uniform materials. An iron ore mine which yields millions of tons of ore a year of nearly uniform quality is of great relative value to the blast furnace plant which can be economically connected. Our Lake Superior ores are just such from the huge Alabama deposits are inferior and numberless rich ore "Minette" of Luxemburg is inferior and have yet all these are rightly valuable deposits because of their uniformity and reliable uniformity. To find more deposits of uniform composition is to seek as for a jewel of great price.

Similar considerations govern the supply of flux and fuel. Large quantities of materials of uniform quality, an abundant supply of coke of constant composition are worth the best efforts of the metallurgist and are comfort to the manager even when procured.

Taking the above conditions, when several varieties of ores, fluxes or fuels in varying proportions have to be worked, the foremost desirable for the greatest and successful working upon the chemical laboratory. Each variety of ore must be analyzed and the several varieties thus stored in certain proportions to obtain a uniform ore mixture. It is safe to say in general terms that this is not any laboratory necessary preliminary to large deposits of uniform material are few and far between.

Given satisfactory and uniform quality of materials brought to the plant, or proper ore mixture, flux mixture or mixed fuel at hand, they should be preserved from rain and snow in order that they enter the furnace substantially dry. The charging one day of dry materials and the next of wet materials soaked by heavy rain will necessarily cause irregularities in the furnace working; use the charges either dry, moist or wet, but whichever it may be, uniformly so, and another cause of irregularity is scotched.

Size of Lump.—All of us have wondered, on first looking at the charges being dropped into a blast furnace, how the furnace could digest them in such huge lumps. And the query is a reasonable one, the blast furnace has a foot, belly and throat, but it has no teeth or anything corresponding to masticating apparatus, and it only too frequently suffers from indigestion. We should help it by "Fletcherizing" the charges. The old-fashioned manager who says it does not make any difference, and throws in lumps of ore, flux and fuel as large as a football, simply does not know that uniformity of lump, in fairly moderate size, is a great advantage to the furnace. One furnace in the United States, upon the advent of a new manager with the above idea in his head, increased its output 25 per cent, accompanied by much greater regularity of working, by merely sizing the ore, flux and fuel down to that uniform lump size for each which by experimenting was found to give the best efficiency.

Composition of the Charge.—Given fixed or uniform qualities of ore, flux and fuel, the proportioning of these so as to have the proper relations between them is in the hands of the chemist. He knows the analysis of each constituent, and calculates the charge upon the two requirements of (1) a given composition of slag to be produced and (2) a given quantity of inert material or burden (ore plus flux), melted down per unit of fuel used. The most important factor in the regularity of furnace working is the one under consideration. If this is faultily or improperly attended to, nothing can save the furnace from working with continued irregularity.

In making these calculations the chemist relies primarily on the accuracy of his analyses and upon experience as to what is required. By experience we mean that he must know from numerous tabulations of analyses of proper slags, within what limits of composition he must bring his slag in order to have it work properly; from similar tabulations he must know the maximum quantity of ore plus flux which unit weight of his fuel can properly take care of in the furnace. He thus draws on his experience, or the recorded and reliable experience of others, as the working basis or assumptions on which to base his calculations. All this must be accurately and reliably worked out; if not, uniformity of furnace working is impossible from the start.

Intervals of Charging.—Three meals a day is a good regime for others, but the blast furnace would soon be "dead" on it. Most furnaces are charged at intervals of five minutes to half an hour, neither of these is the ideal of proper feeding. Uniform driving requires uniform charging. Intermittent charging causes a varying height of charge column, a varying back pressure of blast in the furnace, a varying temperature of the issuing gases, and discontinuously varying layers of material descending the furnace shaft. The ideal charging method is undoubtedly continuous charging, without interruption. This will insure continuity in place of the above mechanical irregularities. It has not yet been adopted for any blast furnace, as far as the writer is aware, but the constant feed of mechanical stokers, and particularly of certain modern gas producers, has certainly laid the foundation for the solution of this problem. The ideally regular working furnace of the future must necessarily have constant and continuous charging. Modern furnaces are approximating this by reducing to the smallest possible time the intervals between charges.

Distribution of the Charges.—An enormous amount of thought and effort has been given to the subject of securing uniformity in this respect. The bell top and mechanical distrib-

utors have done much to overcome the difficulties and to substitute mechanical regularity for human fallibility and carelessness. Every blast furnace manager will admit that the devices so far employed are not yet perfect; there is large room for their improvement. Coke will gravitate towards the middle and ore towards the sides, or *vice versa*, with attendant irregularity in the descent of the charges and working of the furnace. The mechanical engineer can here greatly help the metallurgist to solve a difficult and an important problem affecting uniformity of furnace operation.

Running Out of Iron and Slag.—Present practice is essentially discontinuous in this respect, with consequent irregularity in the working of the furnace. Pig iron is tapped two, three or four times a day, slag is usually tapped every one or two hours. It was formerly the practice to shut off the blast while tapping, but the irregularity caused by this was so great that it has been successfully abandoned. A few furnaces run off slag continuously. This solves half the problem, or advances half-way towards ideal regularity.

The continuous running out of slag and matte is a *fait accompli* in copper and lead blast furnaces, and has really been a most important improvement over discontinuous tapping, with the consequent irregularity of working of the furnace. Attempts to apply these principles to the iron blast furnace have succeeded so far only with regard to the slag. It is highly desirable, from the standpoint of uniform working, that both iron and slag should flow continuously from the furnace; whether through siphon taps or large settlers, etc., is a matter of detail. If this were accomplished, the sudden rushes of hundreds of tons of molten material can be dispensed with, casting machines of limited size could take care of the pig iron, small ladles handle the slag, and, best of all, the conditions in the crucible of the furnace, as far as concerns the presence of melted slag and iron, be kept perfectly uniform.

Interior Shape of the Furnace.—In the "days of old" this was a cause of secular irregularity which often resulted in the "blowing out" of the furnace. It has been nearly completely overcome by improving the lines of the furnace, making the walls thin, and using water cooling. Such long-period irregularity, however, is rather different from the short-time variations which we have been considering.

Temperature of the Blast.—When blast furnaces were blown by cold air the variations in its *temperature* as it entered the furnace from, say, 0° C. to 40° C. (32° Fahr. to 100° Fahr.), made little difference to the running of the fuurnace; it certainly made much less difference than the difference in the quantity pumped by the engines, resulting from the difference in temperature of the outside air. But that is another consideration, and we can say of the temperature of the blast, considering its *temperature only*, that a difference of 40° C. would not make more than 40° C. difference in the furnace temperature, and therefore introduce a rather small irregularity. When hot blast was first used, the iron pipe stoves, operating continuously, furnished blast of fairly constant temperature, with unimportant variations. When, however, regenerative fire-brick stoves came into use, it was soon evident that the discontinuous principle of operation caused great variations in the temperature of the blast, which would be highly heated when first turned onto a hot stove, and have its temperature fall several hundred degrees by the time the stove was turned off blast. This difficulty was met by increasing the number of stoves from two to three, four, and even five, and thus correspondingly reducing the time that any one stove received blast. The uniformity of high blast temperature thus attained has been satisfactory, but not entirely so. "Equalizers" were next invented, to stand near the furnace to receive the hot blast from the stoves and to smooth down the "peaks" and raise the "valleys" in its temperature. They are gigantic thermal flywheels. Fine results have been thus achieved, and splendid uniformity of blast temperature has correspondingly increased the uniformity of the furnace working.

Moisture of the Blast.—Nature varies the atmospheric conditions so that one day the air may be very dry and the next very moist; close observation shows large variations at times even from hour to hour. But increase in moisture cools off the furnace and renders the working irregular. The manager could meet these conditions by varying the amount of fuel in the charge, if he knew 24 hours in advance how much moisture he had to prepare for, since it takes about that time for a fresh charge to descend to the foot of the furnace. That being impossible, he tries to meet the unfavorable conditions by increasing the temperature of the blast. That keeps him "on the jump"; it is meeting one irregularity by practising another, and is never more than moderately successful.

Mr. Gayley first practically solved this problem by drying the blast, reducing its moisture at all times to a small, substantially uniform amount. He achieved success, and his improvement is notable in two ways, viz., it reduces the absolute amount of moisture, thus effecting material saving in fuel, and it leaves it uniform in amount, thus abolishing one of the most annoying and serious causes of furnace irregularity. To have the moisture *uniform* is a great advantage; to have it *uniformly low* is the splendid combination which works the wonders which Mr. Gayley has achieved.

Quality of the Blast.—Everyone skilled in blast-furnace art knows that the more air is pumped into a furnace the faster it runs. Shut the air off, and it stops working altogether. Regularity of working, at uniform rate, is therefore absolutely dependent upon the furnace receiving a uniform weight of oxygen per minute.

This depends absolutely on two factors, the speed of the blowing engine and the temperature of the air entering its cylinders. Make these uniform, and uniform rate of running of the furnace results.

Mr. Gayley has attacked this problem also. In his United States patent 935,628, of September 28, 1909, he describes appliances for furnishing the blowing engine with air of substantially uniform temperature. Where the engine is distant from his drying apparatus (which also cools the air) he passes it to the engine through pipes or conduits with heat insulating sides, protecting it from outside sources of variation of temperature. so that the temperature furnished to the engine shall be constant.

Mr. Gayley avers that he is the discoverer of the great importance of the substantially uniform temperature of the air as it enters the blowing engine. It is now in order for some one to "discover" that it is of equally great importance for the *speed* of said blowing engine to be kept substantially uniform—of which equally important fact there is no hint or mention, suggestion or suspicion in the aforesaid patent, specification or claims.

It seems to the writer that variations in the temperature of the air as it comes to the blowing engine will always be present, that absolute uniformity in this respect cannot be practically attained, and that a much more practical solution would be the following: Construct a mechanical thermo regulator controlled by a thermometer, which will increase the speed of the engine 1 per cent for every 3° C. rise in temperature of the incoming air (1 per cent for every 5° Fahr. rise), and the problem of supplying the furnace with a constant weight of air is solved with exactness.

The Preparation of Silicon in the Laboratory.

BY SAMUEL A. TUCKER.

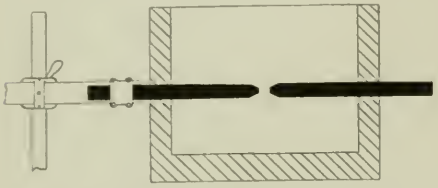
The reduction of silica by carbon in the electric furnace is now carried on quite extensively on a large scale, but its preparation in the laboratory has presented certain difficulties. The method here described can be worked very successfully and affords an interesting application of the electric furnace.

For the charge 12 kilos of a mixture of sea sand and gas coke in the proportion of 77 parts of sand to 25 of coke are used, and placed in a rectangular furnace of firebrick giving a working space of about 10 in. x 9 in., by 8 in. high, which

should be cemented as tight as possible to prevent the admission of air. At a height of 3 in. from the bottom at each end openings are provided for the two horizontal graphite electrodes, 2 in. diameter. These electrodes are adjustable so that at the start the two ends are in contact, but as the operation proceeds they can be separated about 1½ in. to 2 in.

Alternating current was used in the experiments here described and the capacity available was about 50 kw, with means for regulation of the voltage and an impedance coil to balance the arc at the start. After a minute or so the impedance coil was cut out of the circuit and the arc maintained at the terminals of the alternator.

The charge being well mixed and ground so as to pass, say, a 10-mesh screen, is placed in the furnace to a depth of 3½ in. The arc can then be started by withdrawing the electrodes and more of the charge added, carefully at first so as not to ex-



LABORATORY FURNACE FOR PRODUCTION OF SILICON.

tinguish the arc, and then the rest of it so that the furnace is completely filled. The electrodes can then be given a little separation so that a longer arc is obtained and the process carried on in this way to the end of the run.

The first run was as follows:

Time, Min.	Amperes	Volts.
Start— 3	300	54
13	400	65
28	580	30
38	650	31
46	500	36
53	500	48
63	500	48
64	500	54
71	360	58
81	300	..

The gases find a vent and come off at considerable pressure and produce a great deal of vapor, so that good ventilation of the room in which these experiments are conducted is advisable.

After the furnace has cooled down it will be found that a crater has formed about the arc and that approximately one-quarter of the charge has been used up. The silicon is found on the bottom and in the above run the yield was 200 grams, having all the characteristics of the commercial metallic silicon. The next run was made with the same furnace and charge, but the energy was greater, as the following figures show:

Time, Min.	Amperes	Volts
Start— 1	220	57
15	550	30
60	590	54
63	660	55
68	670	58
70	680	61
74	660	69
79	700	69
80	740	71
81	760	72

The yield in this run was 400 grams, but the heat was somewhat too great for the size of the furnace, as the brick work was melted at places and ran down with the silicon at one the specimen obtained was not quite so clear as in the first case. For the size of furnace used about 20 kw-hours would be about right.

Electrochemical Laboratory,
Columbia University.

Boiler Feed Waters with Special Reference to Treatment.

By E. G Bashore.

Water, whether it falls to earth as rain or snow, is never pure. Rain water contains dissolved or absorbed gases—carbon dioxide (CO_2), oxygen (O), ammonia (NH_3) and during thunder showers, nitric acid. Precipitated moisture either in the form of ground water or surface water accomplishes a marked errosive action on the earth's crust, both through mechanical weathering and chemical action. The result of the former type is shown by matter in suspension, while the latter manifests itself through the dissolved impurities, and usually both processes occur simultaneously and mutually aid each other. By means of seeping water, easily soluble rocks, salts and minerals are dissolved as rock salt ($NaCl$), potassium chloride (KCl), magnesium chloride ($MgCl_2$), potassium carbonate (K_2CO_3), sodium sulphate (Na_2SO_4), potassium sulphate (K_2SO_4), sodium carbonate (Na_2CO_3) and gypsum ($CaSO_4$).

Many rocks and minerals which are sparingly soluble in pure water are put into solution in large quantities through the action of water containing carbonic acid (CO_2). Prominent in this class are the carbonates, calcite ($CaCO_3$), dolomite ($CaMg(CO_3)2$), magnesite ($MgCO_3$) and spathic iron ore ($FeCO_3$). Silicous rocks, such as hornblende (containing silica hydrate, with aluminium oxide, magnesium oxide, calcium oxide, soda, and usually some iron) may be decomposed by water containing carbon dioxide and certain component parts held in solution as bicarbonates of potassium, sodium, calcium, magnesium, etc. Furthermore, water may be contaminated with free acid. Such pollution may be due to organic vegetable acids from swamps or bogs, acid discharge from industrial plants or often drainage from coal or ore mines, sulphuric acid being prevalent where the ores are sulphides.

The importance then that the nature of a boiler-feed water bears to boiler operation may be made obvious by considering the quantity of water evaporated with its consequent residual products. For instance, in a 100-hp boiler using water containing only 7 grains of solids per gallon, over 300 lb. will be the quantity of solid matter for one month, while many spring waters would give over 2000 lb of residual matter, which must either be blown out or deposited.

As a result of the impurities that a feed water may carry, the following conditions are conventionally recognized in boiler practice, namely, scale formation, corrosion, foaming and priming.

Many reagents have been suggested for the treatment of feed waters, prominent among which are lime, soda ash, caustic soda, barium hydrate, barium chloride, tri sodium phosphate, petroleum, tannate of soda and tannin extract. Oak, hemlock, sumach, logwood, etc., are used for scale on account of tannic acid being extracted; also vinegar, fruits, molasses, distillery refuse, etc., have been used for the prevention or removal of scale, their action depending upon the presence or formation of acetic acid. Organic decoctions of the type are to be avoided, for often they are injurious to the iron and cause corrosion and pitting. No absolute rule can be established for the treatment of all feed waters, nor can a "cure all" be improvised, as each feed water from a different source will demand a treatment peculiar to itself.

The functions of the chemical treatment of a boiler feed water is to minimize the existing evil. However it must be remembered that every chemical reaction calls for a by-product, and while possible, through chemical treatment to

correct one unfavorable condition, it is also possible to induce another. But with intelligent operation, and through cognizance of the principles involved, a great saving in coal and labor in cleaning the boiler can often be accomplished in industrial plants through the application of a properly advised treatment.

The processes for conducting the treatment can be outlined under three heads:

(1) The addition of some substance in the water in the boiler;

(2) Preliminary treatment with heat in some special apparatus;

(3) Preliminary chemical treatment.

In considering the first subhead (1), emphasis should be made of the fact that a feed water should always be purified before it enters the boiler, for the pertinent function of a boiler is to generate steam and not to serve as a still in which to conduct chemical reactions. However, cases arise where it might seem advisable to use some "solvent" in the boiler for the purpose of converting the scale into a form in which it will be easily dislodged. Such "solvents" (usually soda ash, sodium hydroxide or tri-sodium phosphate) should be used with a thorough understanding of their reactions. Too great concentration should be guarded against on account of priming; the danger of severe foaming if animal or vegetable oil is present in the water on account of saponification should be borne in mind; and great care should be exercised that the dislodged scale does not cause overhauling and burning of the steel through accumulation.

Under the subhead (1) also comes the use of metallic zinc in the boiler. With waters which have a corrosive tendency on account of the presence of magnesium chloride, it has been found that metallic zinc suspended in the boiler counteracts the corrosive action. Where corrosion of iron is suspected through the agency of electrolysis, zinc is also a specific counteractant. With the use of zinc it is to be noted that the metal should be suspended at a point away from the direct heat of the furnace, and the utmost care should be taken that the zinc has perfect contact with the shell of the boiler. The use of such substances as clay, fats, starchy matters, wood extracts, vinegar, etc., in the boiler is certainly not to be recommended, and is to be avoided.

Under the second category feed-water heaters are properly classified. Aside from the economy gained by the boiler generating more steam through the use of flue gases or exhaust steam, which otherwise would go to waste, an important point is the reduction of the temporary hardness of the water. This is brought about through the liberation of the carbon dioxide at 212° Fahr., which holds the bicarbonates of calcium and magnesium in solution.

Many types of feed-water heaters are on the market, but it is hardly within the compass of this paper to describe them. However, for their successful operation it is necessary that sufficient exhaust steam be used to keep the temperature of the water to 212° Fahr if possible, and they furthermore must be kept clean.

When there is oil present in the condensed water a separator or filter should always be used.

The third sub-division, namely, purification by preliminary chemical treatment, leads to the process which will be given most consideration, for, as previously pointed out, the boiler is not a proper apparatus in which to correct the deleterious effect of bad feed water. This method is usually termed "the water softening process" and is conducted with the use of reagents in settling tanks. The simplest form of apparatus would be two tanks, one from which the treated and settled water would be drawn off for use, while the addition of reagents, agitation and settling would be going on in the other tank.

A number of reagents can be used to accomplish the precipitation of the scale-forming constitutents, but on account of the efficiency under most conditions and low cost, the lime, soda ash treatment will be considered. This process is conducted in settling tanks and is either intermittent or continuous. In the intermittent process fixed quantities of water are treated in the tanks with weighed amounts of each reagent required for the softening, while in the continuous system the requisite quantity of chemicals are introduced in direct proportion to the flow of the water. In order to insure the highest efficiency from this process, the reagents (lime and soda ash) should be added in the proper proportion and thoroughly mixed with the water, the reaction should be accelerated if possible and should be complete. The apparatus should be designed so as to cause rapid sedimentation and perfect clarification. With proper care, particularly incrustaceous waters can be softened to the extent of containing from three to five grains of incrustaceous matter per gallon.

Scale formation is due to the presence of carbonates, sulphates, sediment and mud, and soluble salts in the feed water and its deposition on the inside of boilers not only lessens the available water space but acts as a non-conductor of heat. (It is estimated that 1/16 in. of scale causes a loss of 13 per cent of fuel; ¼ in., 38 per cent, and ½ in., 60 per cent).

To calculate the amount of material that will be deposited in 1000 gal. of water per day, take the number of grains per gallon of each as shown by the analysis and divide by 7; the quotient will be pounds deposited in 1000 gal. of water per day. The scale-forming carbonates in water are those of calcium and magnesium. Both are very sparingly soluble in pure water, but the presence of carbonic acid (CO_2) in the water renders them soluble as bicarbonates; however, carbonic acid is insoluble in boiling water, and is liberated with a consequent precipitation of calcium and magnesium carbonates which tend to deposit as a soft scale. Each grain of calcium or magnesium bicarbonate reported in the analysis as carbonates, requires the number of grains divided by 5.5 expressed in pounds of lime for 1000 gal. of water per day. For example, 5.5 grains of total carbonates requires 1 lb. of lime for 1000 gal. of water per day. The precipitation of carbonates is best conducted in external settling tanks, and particularly so if the carbonates exceed 10 grains per gallon.

The sulphates which are scale forming are those of calcium and magnesium. Magnesium sulphate alone does not form a hard scale, but in the presence of calcium sulphate it forms a very stony scale hard to remove. Calcium sulphate alone forms a hard, tenacious scale. The presence of these sulphates having been determined, soda ash should be used to prevent the formation of the resultant hard scale. However, it is to be noted that the reaction forms sodium sulphate, which, if too concentrated, will produce priming. This condition will be dealt with under the proper head. The quantity of soda ash required may be calculated by dividing the number of grains per gallon of magnesium sulphate by 8 and the number of grains of calcium sulphate by 8.75, and the quotient represents pounds of soda ash per 1000 gal. of water per day. If carbonates and sulphates exceed 15 grains per gallon, external treatment is advisable.

Sediment and mud also cause scale by depositing and baking on the fire sheet of a tubular boiler, the lower tubes in a water tube boiler and the crown sheet in a marine and locomotive boiler.

Soluble salts can also be incorporated in the scale if the concentration is too great. Excess beyond 300 grains per gallon should be avoided.

Corrosion or pitting, which is by far the most dangerous condition met with in boiler practice, may be caused by free acids or substances which develop acidity.

Free sulphuric acid often exists in water and should be neutralized or precipitated. Divide the number of grains of free sulphuric acid by 6.5 and the quotient equals the number of pounds of soda ash required to neutralize 1000 gal. of

water per day. This same rule applies to hydrogen sulphide when present. Under certain conditions it is advisable to use barium chloride or hydroxide and precipitate the barium sulphate externally.

If organic acids exist in the water they have a very corrosive action and can best be corrected by external treatment with alum and soda ash. Divide the organic acidity as reported in grains per gallon in the analysis by 6 to get pounds of soda ash and by 8 to obtain pounds of alum required per 1000 gal. of water per day.

Hydrochloric acid is sometimes found in feed waters and can be corrected by the use of soda ash. The number of grains of free acid divided by 9 equals the pounds of soda ash required for 1000 gal. of water per day.

Magnesium chloride is said to be unstable at high temperatures and in the presence of heated iron and water will split up to form magnesium oxide and hydrochloric acid. The number of grains of magnesium chloride as shown by the analysis, divided by 4.8, equals the number of pounds of soda ash required for 1000 gal. of water per day.

Sodium chloride and silica are said to develop free hydrochloric acid at high temperatures through the formation of sodium silicate. To counteract this effect an alkali such as soda ash must be used. Take the number of grains of available silica present that will act on the sodium chloride and divide by 9.6 and the quotient equals the pounds of soda ash required.

Foaming and priming may be considered under the same head, for both tend to cause wet steam. Priming is due to high surface tension, while foaming is due to matter in suspension.

Sodium sulphate is found whenever soda ash is used to precipitate calcium or magnesium sulphate and is also found in many southern waters. Priming is the result of too great concentration. While the limit of concentration is determined by the type of boiler, the following figures are a close approximation of the truth. Old type two-flue boilers, 1000 grains per gallon, steam at 50 lb. pressure. Horizontal return tubular boiler, 500 to 600 grains per gallon, 100 lb. pressure. Modern water tube boilers, 250 to 400 grains per gallon, steam at 125 lb. pressure. Locomotive boilers, 100 to 200 grains per gallon, steam at 200 lb. pressure.

Sodium chloride also causes priming, but not to such a pronounced degree as sodium sulphate. Usually twice as much sodium chloride can be allowed to concentrate as sodium sulphate without producing priming.

Foaming can be caused through mud and suspended matter and the remedy is either blowing down or filtration.

Sewage and algæ also cause foaming and create a scum on top of the water. This condition may be taken care of either by the use of a surface blow-off or settling filtration.

The subject as here outlined is at best only a brief resume of conditions as met with in boiler practice with reference to the treatment of feed waters, and many cases arise where it is advisable to seek the advice of an experienced chemist or consulting engineer who makes a specialty of water softening.

Bethlehem, N. J.

Decomposition Points of Some Sulphates in an Air Current.

By Walter S. Landis, M. S.

It has been rather disappointing to look in the many new treatises on metallurgy and find such statements as "ferrous sulphate decomposes at a low red heat" or other equally vague statement. This decomposition point of the sulphates is of very great importance in all roasting operations, and very often their occurrence is being forced down our throats (because we would not take it in any other way), and we are expected to plan to utilize a part of our plant as the furnaces themselves, more exact data is wanted on such decomposition points.

The purpose of this work was to determine a few of these decomposition points of the sulphates. The work has already been done without doubt, but the results have never appeared where they are readily accessible, and have certainly not been copied in our more recent books. This determination was not made with the great accuracy that the physical chemist might desire, but rather more to simulate the actual conditions existing in a roasting furnace, that is, in a current of air such as would pass over the materials in a roasting furnace.

The method employed was to insert a porcelain boat, carrying the sulphate in a porcelain combustion tube, into a combustion furnace. Lying on the surface of the sulphate in the boat was the hot junction of a Le Chatelier thermo-couple, previously carefully standardized, the cold junction being in melting ice. Connected to the one end of the combustion tube was a three-mouthed bottle containing a solution of caustic soda carefully neutralized with sulphuric acid, using methyl orange as an indicator. The connecting tube ended in a glass tube dipping well below the surface of the liquid in the three-mouthed bottle. Through the central mouth passed the stem of a separatory funnel, in the bulb of which was placed a dilute solution of caustic soda. Finally the third mouth of the bottle was connected with an aspirator for producing the necessary air current through the apparatus. As decomposition took place the SO_2 and SO_3 passed along with the air current into the liquid in the bottle and affected the indicator. The introduction of caustic soda from the separatory funnel allowed of this acid being neutralized and served to keep the apparatus in continuous working order. Some idea, only qualitative, could be gained as to the rate of decomposition by the amount of caustic soda added to keep the indicating solution neutral.

The results obtained on the sulphates studied were as follows:

$FeSO_4$: The first traces of decomposition showed themselves at 550° C. The increase was but slight up to 580° C and then became more rapid until 600° C. was reached when there was a sudden increase. On heating for two hours at this temperature all decomposition seemed to have stopped as only very slight traces appeared on further heating to 650° C. No further traces were noted between this last temperature and 960° C., the highest temperature attainable in the test.

$CuSO_4$: The first traces of acid appeared at 400° C. Only slight amounts were noted as the temperature was slowly increased to 600° C., when a very rapid evolution took place. A further trace was obtained at 700° C., and then no more up to 900° C., the highest temperature attained in this investigation.

$ZnSO_4$. The first traces of decomposition appeared at 730° C., more appearing at each increase of temperature up to 760° C., when the decomposition was very rapid. All the acid seemed to have been removed at this temperature for no more was found on raising the temperature to 980° C.

In all cases the experiments were repeated several times and gave almost identical temperatures for each stage at each repetition, the points seemingly very markedly characteristic of each salt.

Data of this nature needs no further comment as to its value to the roaster, showing the minimum temperature at which the furnace may be run for complete decomposition and the limits which need not be exceeded for producing the same result.

Metallurgical Laboratory,
Lehigh University.

A Primer on Explosives is to be issued by the United States Geological Survey. According to the scope indicated in advance sheets, this primer will be a need in aiding coal miners and all who handle explosives.

The Magnetic and Electrical Properties of the Iron-Nickel Alloys.

BY CHARLES F. BURGESS AND JAMES ASTON.

In iron and nickel we have the two most magnetic elements. The fact that a mixture of certain percentages of these two metals results in a non-magnetic alloy makes an investigation of the magnetic qualities of an entire series of these alloys of interest.

There is much literature embodying the results of the research on the irreversibility of certain properties of the nickel-iron alloys and most of the published data of magnetic investigations is concerned with the temperatures of this appearance and disappearance of the magnetic susceptibility for different percentages of nickel.

In the investigation of Barrett, Brown and Hadfield (Jour. Inst. Elec. Engrs., Vol. XXXI, page 674) there is included complete hysteresis loops for a number of nickel-iron alloys in which the carbon runs from 0.13 to 0.23 per cent and the manganese from 0.65 to 1.08 per cent. Also, the magnetization curves of a series was determined by Carpenter, Hadfield and Longmuir (7th Report of the Alloys Research Committee, Inst. Mech. Engrs., Vol. 2, 1905, page 857). In this series the carbon was that of a medium steel (0.41 to 0.52 per cent) and the manganese content varied from 0.79 to 1.03 per cent. These steels, except for the increase of carbon, were almost identical with those of Hadfield, mentioned above; in fact, the work was intended as supplementary to this previous research.

This present paper deals with an investigation of a complete series of iron-nickel alloys in which the endeavor has been to keep down the impurities to the lowest possible point in order

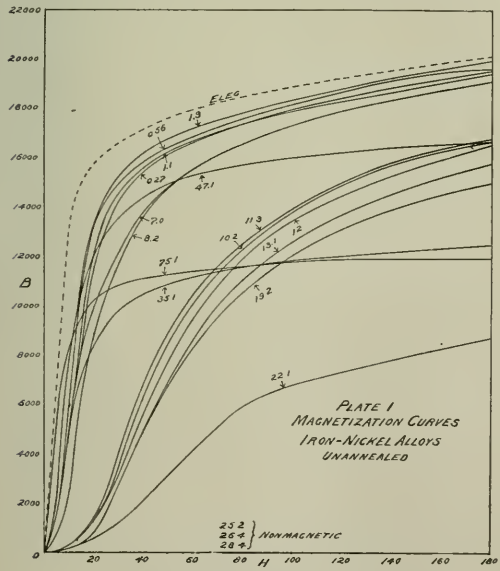


FIG. 1.—MAGNETIZATION CURVES. IRON-NICKEL ALLOYS UNANNEALED.

that the effects observed may be ascribed entirely to the alloying element. The work is part of an extended investigation of electrolytic iron and its alloys carried out during the past several years at the chemical engineering laboratories of the University of Wisconsin, under a grant of the Carnegie Institution of Washington.

The iron used is of a high degree of purity obtained by a process of double electrodeposition. The electrolytic material has a purity represented by an iron content of 99.97 per cent, the total content of sulphur, phosphorus, manganese and silicon being thus 0.03 per cent. The alloys were prepared by melting this iron with the desired quantity of nickel, also electrolytic material of a high purity, in magnesia crucibles supported by a graphite jacket. The crucibles were covered with a magnesia and a graphite lid and buried in an electric resistor furnace in which the melting was done. The resulting ingots

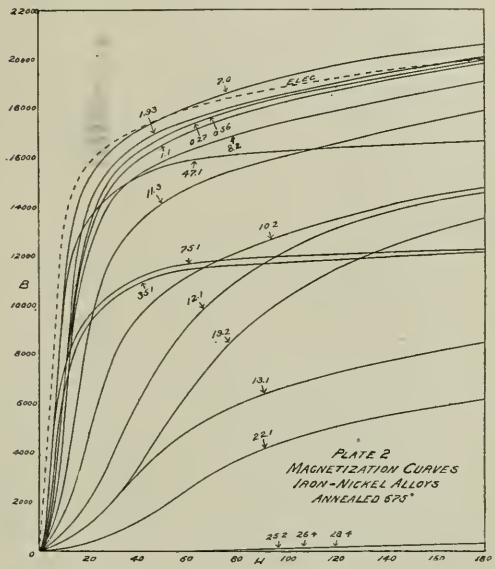


FIG. 2.—MAGNETIZATION CURVES. IRON-NICKEL ALLOYS ANNEALED 675°.

of about 500 grams (1 lb.) in weight, were heated in a forge fire and drawn down under a steam hammer into rods about ⅝ in. in diameter and 18 in. long.

Experience has shown that, in spite of every precaution, there is a slight carbon absorption, unquestionably due to the reaction of the iron upon the carbon monoxide atmosphere of the furnace. The amount, however, is small, and may be assumed in most cases to be well under 0.10 per cent. The other impurities, such as silicon, sulphur, phosphorus and manganese, are entirely negligible. In the present discussion the materials may be considered as binary alloys of iron and nickel and the effects to be due to these two elements alone.

Analysis.

The following series was subjected to magnetic tests.

Bar.	% Nickel— Added.	Analysis.	% Carbon.
144C	0.25	0.27	
144D	0.50	0.56	
144E	1.0	1.07	
144F	2.0	1.93	
144J	7.0	7.05	
157A	8.0	8.17	
157D	10.0	10.20	
157E	11.0	11.29	
157H	12.0	12.07	
144M	13.0	13.11	0.088
144P	19.0	19.21	
166G	22.0	22.11	
154S	25.0	25.20	
166I	26.0	26.40	
166C	28.0	28.42	
166L	35.0	35.09	
166O	50.0	47.08	
166Q	75.0	75.06	

As will be seen, iron and nickel alloy readily and without segregation, since the agreement is strikingly close between the added percentage and that found upon analysis. This evidence is more conclusive when it is remembered that the analyses

were made from turnings during the machining of test bars and after the forging of the ingot.

From the rods forged from the ingots, bars were reserved for tensile strength and one was taken for the magnetic tests. The latter was finished to a diameter of 1 cm with a length of about 20 cm. No sample of pure nickel was tested, since it was impossible to forge the ingot into a bar. It was possible

This instrument was selected because of its adaptability to the work on the several hundred bars on hand and in prospect. It has given good satisfaction in its present form, both as regards rapidity and consistency. The results are not based upon an absolute measurement and the instrument must be calibrated against a standard determined by ballistic test. In our tests, magnetization curves were taken by the method of reversals

Table I. Data of Magnetic Tests.

Bar	% Nickel	H = 50 Unann.	H = 50 Ann. 675°	H = 50 Ann. 1,000°	H = 50 Quench 900°	H = 20 Unann.	H = 20 Ann. 675°	H = 20 Ann. 1,000°	H = 20 Quench 900°	H = 10 Unann.	H = 10 Ann. 675°	H = 10 Ann. 1,000°	H = 10 Quench 900°	H = 100 Unann.	H = 100 Ann. 675°	H = 100 Ann. 1,000°	H = 100 Quench 900°
144C	0.27	2,600	6,000	9,100	10,450	11,400	13,200	13,650	13,900	16,150	16,850	16,800	16,750	17,900	18,400	18,300	18,100
144D	0.56	5,900	5,100	11,950	12,000	13,250	13,250	14,800	15,000	16,400	16,750	17,000	18,050	18,200	18,500	18,400	18,400
144E	1.07	3,500	5,100	11,500	7,600	12,050	12,800	14,650	11,200	16,250	16,500	17,000	15,600	17,800	18,250	18,400	18,400
144F	1.93	4,850	6,600	10,600	9,700	13,250	13,950	14,800	14,200	16,750	17,050	17,000	16,950	18,300	18,550	18,400	18,400
144J	7.05	4,850	10,000	9,500	9,300	9,950	15,050	13,800	13,250	14,750	17,550	17,000	16,750	17,250	19,250	18,750	18,500
157A	8.17	2,600	5,100	8,000	8,350	8,750	12,050	12,150	13,000	14,800	15,800	16,250	16,750	17,250	17,350	18,300	18,500
157D	10.20	50	1,450	5,300	7,950	900	4,650	10,150	12,350	8,550	10,500	15,650	16,450	13,800	12,850	18,350	18,600
157E	11.29	150	2,800	4,100	5,900	1,300	8,800	7,650	10,100	8,900	14,150	12,400	15,450	13,950	16,000	15,400	18,050
157H	12.07	350	750	2,700	4,600	1,300	2,000	5,900	10,100	7,700	7,750	10,700	15,250	13,450	12 200	13,850	17,700
144M	13.11	100	450	700	2,450	800	1,150	1,850	8,150	6,850	4,100	6,800	14,600	12,600	6,650	11,550	16,650
144P	19.21	400	400	650	2,200	1,150	1,050	1,700	7,000	6,750	5,000	7,050	12,950	12,050	10,350	12,050	17,200
166G	22.11	100	100	400	300	500	350	1,100	800	3,100	1,800	4,800	4,150	6,800	4,500	8,500	8,050
154S	25.20	0	0	50	50	0	0	150	100	0	0	500	350	0	100	1,050	850
166J	26.40	0	0	100	0	0	0	200	50	0	0	650	350	0	0	1,200	850
166C	28.42	0	0	0	0	0	0	0	0	0	0	0	0	0	100	100	100
166L	35.09	4,850	6,400	8,800	6,500	8,350	9,250	10,750	8,400	10,850	11,300	12,000	10,650	11,850	11,700	12,300	11,800
166O	47.08	7,850	10,600	13,650	10,300	12,100	13,450	15,150	12,400	14,950	15,550	16,250	14,800	16,000	16,200	16,500	16,000
166Q	75.06	8,300	7,200	8,450	8,700	10,200	9,550	10,650	11,200	11,250	11,450	11,850	11,800	11,800	11,950	12,300	12,100

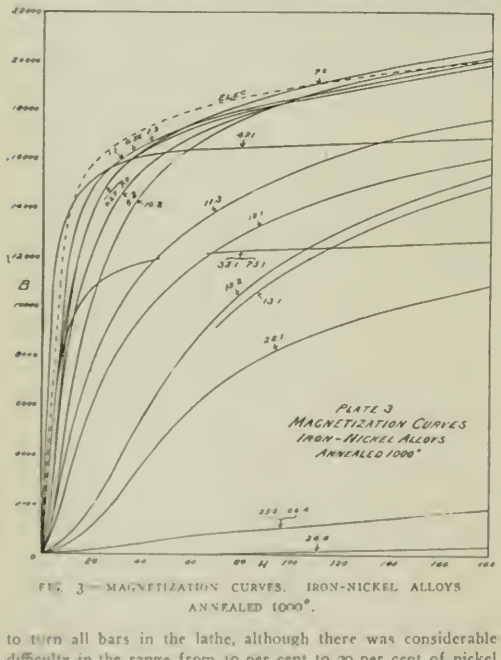


FIG. 3—MAGNETIZATION CURVES. IRON-NICKEL ALLOYS ANNEALED 1000°.

to turn all bars in the lathe, although there was considerable difficulty in the range from 10 per cent to 20 per cent of nickel because of the hardness.

Magnetic Tests.

The magnetic tests were made upon an Esterline permeameter. This instrument is essentially a small generator, the armature of which is driven at a predetermined constant speed. The voltage generated is proportional to the magnetic flux through the armature, since all factors are constant except this flux. The total number of lines of force is equal to that in the bar under test, since this bar forms part of the magnetic circuit. The necessary rheostats and switches are provided for regulation and reversal of the magnetizing current, for compensation, etc. But one recording instrument is used—a Weston millivoltmeter, calibrated to read directly, without calculation, the flux density B and the magnetizing force H in C.G.S. units.

and hysteresis loops by the step-by-step process. Complete magnetization curves and readings of the retentivity and the coercive force were taken under four different heat treatments, first, as forged; second, after heating to 675 deg. C and slowly cooling; third, after heating to 1000 deg. and slowly cooling, and fourth, after heating to 900 deg. and quenching in water.

The results of the magnetic tests are given in Tables I and II and in the curves of Plates 1, 2, 3 and 4. For purposes of comparison, there is given in each plate a dotted curve, plotted from the data of a test on a bar of electrolytic iron in the unannealed state, and which is of very high magnetic quality. The effect of the increase in the percentage of nickel is very apparent. In the unannealed state the alloys fall into several groups. Bars 144C, 144D, 144E and 144F, with 0.27, 0.56, 1.07 and 1.93 per cent of nickel, are very close together, and but little inferior to the standard. Increase of the nickel to 7.05 and 8.17 in bars 144J and 157A, has resulted in a falling off in quality. This is still greater for 157D (10.20 per cent), 157E (11.29 per cent), 157A (12.07 per cent). 144M (13.11 per cent) and 144P (19.21 per cent), which have a low permeability, and the curves of which take positions in the order of the nickel content. The quality of bar 166G, with 22.11 per cent, has become very poor, and finally bars 154S (25.20 per cent), 166I (26.40 per cent) and 166C (28.42 per cent), have become non-magnetic. Further increase of nickel content in 166L, 166O and 166Q, with 35.09, 49.08 and 75.06, has restored the magnetic quality very markedly, the maximum values being reached with 166O (47.08 Ni). Below the saturation bend of the curves (at about H = 40) these three bars have a very high permeability, but beyond this point the permeability becomes practically unity, with the flux density increasing but little more than the increase in the value of the magnetizing force. This latter feature is even more marked after the subsequent annealings at 675 deg. and 1000 deg.

Annealing at 675 deg. has resulted in general in an improvement of magnetic quality throughout the entire range. This is most marked for 144J, with 7.05 per cent Ni, and 157E, with 11.29 per cent Ni. On the contrary, there has been a deterioration in bars 144M (13.11 per cent), 144P (19.21 per cent) and 166G (22.11 per cent); this is very pronounced in the case of the first of these. A very slight amount of magnetism has been induced in the three non-magnetic bars 154S, 166I and 166C.

The second annealing at 1000 deg. has for those bars with smaller nickel content resulted in an increase of permeability for the lower magnetizing forces, with but little change in the upper ranges. Very pronounced changes have taken place in

bar 157D, with 10.20 per cent Ni, which is better throughout, as are bars 157H (12.0 per cent), 144M (13.11 per cent) 144P (19.21 per cent) and 166G (22.11 per cent). The permeability of bar 166O, with 47.08 Ni, has become very high at the lower values of H; in fact, it is about the highest of any of the materials tested in this work. Bars 154S (25.20 per cent) and 166O (26.40 per cent) have become appreciably magnetic.

Quenching at 900 deg. has not resulted in much change, except for the alloys with 11.29 per cent, 12.07 per cent, 13.11 per cent and 19.21 per cent of nickel (157E, 157H, 144M and 144P) where the permeability has greatly increased.

Referring to Table II, the retentivities tend to decrease as we increase the nickel to the compositions of the non-magnetic alloys; and the values of coercive force increase. Beyond this composition the retentivities are again evident, and the coercive forces are small. This is in line with the deductions drawn from the curves and data. In general the coercive forces become smaller with each succeeding heat treatment. For the smaller nickel percentages of 0.27 and 0.56, in bars 144C and 144D, the values 4.3 and 4.2 are about those of electrolytic iron;

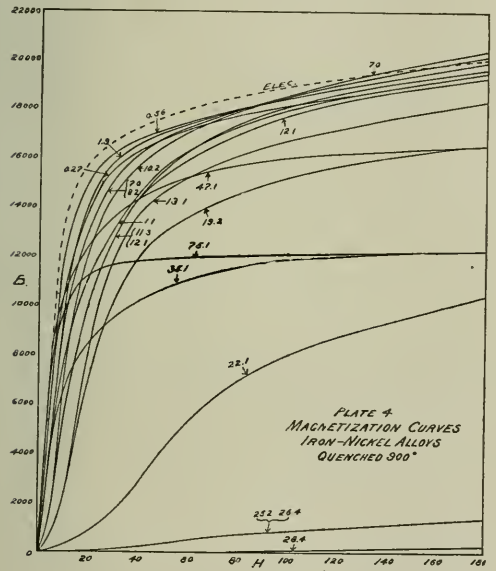


FIG. 4.—MAGNETIZATION CURVES. IRON-NICKEL ALLOYS QUENCHED 900°.

and the values of coercive force for the three bars with the maximum content of 35.08, 47.08 and 75.06 per cent are markedly low.

The alloys with medium percentages of nickel (10 per cent to 20 per cent) seem to be very susceptible to heat treatment, and are at their best after quenching. This might be explained from the fact that nickel has a tendency to lower the temperature of transition in the iron from the hard γ to the soft α form. Consequently, while slow cooling would result in γ state, with its hardness and low permeability, the quenching, with cooling at lower temperatures, might have a tendency toward transition to the α form, with a resultant softness and better magnetic quality.

The strange feature of this nickel-iron series is that with two highly magnetic materials we can obtain all degrees of magnetization, even to a non-magnetic range. Equally striking is the fact (not here investigated) that these alloys in the non-magnetic range at normal temperatures become strongly magnetic at lower temperatures which are dependent upon the composition. They remain magnetic upon a rise of temperature of several hundred degrees, when they again become non-magnetic. Many attempts have been made to explain this anomaly, the reasons being based largely upon allotropy in the two elements.

The behavior is such that one might almost suppose that in the molecular arrangement of the alloys the iron and nickel

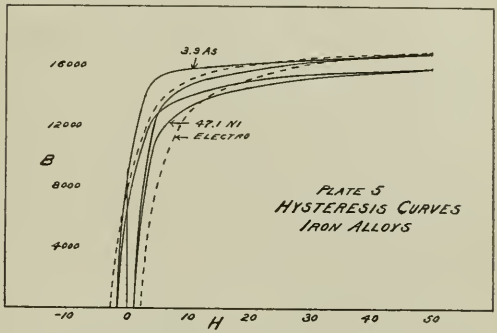


FIG. 5.—HYSTERESIS CURVES. IRON-ALLOYS.

become so oriented as to be of opposing polarity; this opposition becoming more pronounced with increase of the nickel content until finally a balance is reached with the resulting non-magnetic alloy. A further increase of nickel upsets this balance in favor of the nickel with its polarity preponderating.

In the range of commercial usage with 2 to 4 per cent of nickel the magnetic properties of the alloys have not fallen off to such an extent as to obviate against the use of the nickel in case the resulting mechanical qualities required are of sufficient importance to warrant its selection.

Hysteresis.

In connection with the magnetic tests, hysteresis curves were taken on such alloys as showed high permeability, especially such as had low magnetizing forces, and in consequence seemed particularly adapted to transformer working.

In the tests the cycle was carried to a maximum magnetizing force of $H = 50$. Typical bars with low hysteresis losses are noted in the following table; all were annealed at 1000 deg C.

Bar.	Per Cent Composition.	Relative Area of Loops.
117A	Electroforged	1.00
NSW	Swedish anode	1.08
55D	0.067 Al.	1.10
113D	3.862 As.	0.58
123C	1.033 Co.	1.20
166O	47.080 Ni.	0.53
96H	4.655 Si.	0.76
121J	2.059 Sn.	1.01

The results of the measurements upon bar 166O (47.08 per cent Ni) are plotted in Plate 5, together with those on electrolytic iron bar 117A, which was used as the standard of comparison, and 113D with 3.86 arsenic, which latter material showed the highest magnetic quality of any sample tested. The half-loops only are given.

In the above table is noted the relative area of the loops compared to the standard 117A as unity. These figures are indicative of the relative hysteresis losses per cycle of the materials in question. In our tests, therefore, the 50 per cent nickel alloy had the lowest loss, but 53 per cent of the standard and exceeded even the arsenic bar where the loss was 58 per cent of the standard. However, the great cost of this large quantity of nickel would eliminate its extensive use in general practice; also it is in reality not so good a material as some of the other alloys which have but sightly higher loss per cycle and which reach values of flux density considerably higher than the nickel bar for the same magnetizing forces.

Electrical Resistance.

As a study concomitant with that of the magnetic properties

the electrical resistance of the bars of the nickel-iron series was determined. The measurements were taken by the fall of potential method by recording the drop across a measured

Table II. Coercive Force and Retentivity.

Bar	% Nickel	H (Max.) = 200 Unann.	Ann. 675°	Ann. 1,000°	Quench 900°	H (Max.) = 200 Unann.	Ann. 675°	Ann. 1,000°	Quench 900°
		Coercive force.				Retentivity			
144A	0.27	13.4	9.0	6.0	4.3	12,400	12,500	10,600	10,900
144D	0.56	11.9	9.1	4.0	4.2	11,100	12,300	10,100	10,700
144E	1.07	11.4	9.1	4.9	6.0	12,600	12,800	11,300	8,900
144F	1.93	10.2	9.0	5.0	6.0	12,800	14,400	11,300	11,700
144J	7.95	11.0	7.5	5.8	5.5	9,100	14,200	10,100	10,700
157A	8.17	12.5	10.0	6.3	6.0	8,000	12,500	9,600	9,500
157D	10.20	14.0	18.0	8.5	7.2	8,700	9,500	8,600	10,300
157E	11.29	32.5	13.5	9.2	8.8	9,200	11,800	7,000	9,500
157H	12.07	35.0	35.5	10.7	9.5	8,800	9,800	5,900	9,500
144M	13.11	34.8	32.5	31.5	14.9	8,500	4,500	7,700	11,900
144P	19.21	38.5	40.5	34.2	15.5	8,400	8,300	8,500	9,300
166G	22.11	44.5	51.8	37.0	42.7	5,000	3,900	6,300	6,100
144S	25.20	Non.	200+	52.3	64.5	Non.	200	1,000	1,000
161I	26.40	Non.	200+	45.5	66.3	Non.	200	1,100	900
166C	28.42	Non.	200+	200+	200+	Non.	2,000	400	400
166L	35.09	7.9	6.5	1.9	2.7	4,800	6,200	4,000	4,500
166O	47.38	6.7	5.5	1.6	1.9	7,700	9,000	8,700	5,200
166Q	75.06	3.1	3.5	1.9	1.5	6,000	4,600	3,600	4,200

length of the bar when a large current of about 50 amp was flowing. In the following table the results are given in

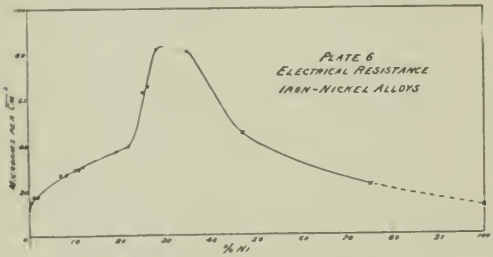


FIG. 6.—ELECTRICAL RESISTANCE. IRON-NICKEL ALLOYS.

microhms per cubic centimeter. There is also noted the relative resistance in terms of the same standard bar of the electrolytic iron which was used in the magnetic tests. (See also Fig. 6.)

Electrical Resistance.

Bar	% Ni	Microhms per cu. cm.	Relative resistance.
157A	0	12.1	1.00
144C	0.27	13.1	1.08
144D	0.56	15.4	1.27
144E	1.07	16.9	1.40
144F	1.93	16.4	1.36
144J	7.95	26.9	2.22
157A	8.17	26.7	2.20
157D	10.20	28.6	2.36
157E	11.29	29.4	2.43
157H	12.07	30.3	2.50
144M	13.11	34.8	2.87
144P	19.21	36.2	2.99
166G	22.11	38.7	3.20
144S	25.20	63.2	5.22
161I	26.40	65.5	5.41
166C	28.42	82.0	6.77
166L	35.09	81.1	6.70
166O	47.38	41.7	3.69
166Q	75.06	22.1	1.83
	100.00	12.4	1.02

There is a gradual increase of resistance with the increase of nickel up to a percentage of 22.11, where there is a sudden rise in the curve at temperatures corresponding to those of the austenitic alloys. However, the curve continues to rise, reaching a maximum in the range between 28.42 per cent and 35.09 per cent. At these two points the resistances are respectively 82.0 and 81.1 microhms per cu. cm. or 6.77 and 6.70 times the resistance of the standard. Beyond this range the two determinations made indicate a fall of resistance with increase of nickel in a curve very similar to that on the other side of the maximum range. The final point of the curve at a nickel content of 100 per cent is taken from a standard handbook. Maybe not the maximum point in this curve of resistance but at 35 per cent of nickel, corresponding to the compound Fe_2Ni? Measurements of other kinds, such as dilatation, indicate a change of properties at this composition and tend to support the above view.

Chemical Engineering Laboratories, University of Wis

Current Densities and Energy Losses in Electrodes.

By Jakob Forssell, Ph.D.

At the meeting of the American Electrochemical Society in Philadelphia, May, 1907, a paper on "carbons for electrometallurgy"[1] was read by F. A. J. FitzGerald and the writer. It was our intention at that time to continue our investigation of this subject in order to obtain a full knowledge of the influence and the relations of the different properties of carbon electrodes with regard to energy losses and current densities in electrodes. The paper by Mr. FitzGerald on "the heat conductivity of carbon"[2] read at the meeting of the society in New York, October, 1907, was another contribution to the series of researches contemplated.

In order to obtain a basis for further work along these lines the writer of this article attempted at that time to carry out a theoretical analysis of the different factors influencing the energy losses in electric furnace electrodes. The results obtained from this analysis seemed to offer a very promising guide for the experimental work. However, for various reasons this work was interrupted. The general conclusions derived from the theoretical treatment of the subject as given below were demonstrated and discussed in a paper on "some principles regarding the construction of electric furnaces" read by the writer on Oct. 8, 1908, before the Chemical and Metallurgical Section of the Swedish Association of Technical Societies in Stockholm.

Now that the work on this subject has been taken up again by Mr. C. A. Hansen[3], it seems to the writer that a publication of his calculations may be of interest, since the results obtained by Mr. Hansen appear as logical consequences of these theoretical calculations, thus tending to confirm their correctness. The value of this analysis of the problem regarding energy losses and current densities in electrodes, however, extends far beyond an interpretation of this special case. It gives a general understanding of the relations of the different factors influencing the working of furnace electrodes under different conditions and therefore may serve as a useful aid in the construction of electric furnaces and in suggesting further valuable and necessary experiments regarding electrodes.

In dealing with the problems of energy losses in furnace electrodes it will be necessary to consider the conditions in furnaces of different types. We will, therefore, first take up the question with regard to such furnaces where the electrode enters the furnace through an insulating medium or wall, as shown in Fig. 1. This is the case, for instance, in a number of resistance furnaces.

The general arrangement assumed is evident from the cut. The energy waste consists of two main parts, i. e., heat generated in the electrode by the electric current and heat conducted away from the charge into the electrode. It is disposed of by conduction to the cable connection and by the cooling influence of the atmosphere surrounding the protruding part of the electrode. Further, some heat will also be conducted away by the wall surrounding the electrode.

In order to carry out an exact analysis of the energy losses in the terminal, we would have to know, not only all its dimensions and working conditions, but also the mathematical expressions for the electric resistivities of the materials of the electrode and of the surrounding wall and for the thermal conductivities of said materials as functions of the temperature. Even if we possessed such a knowledge the deduction in this way of the energy losses incurred would lead to a very complicated expression, unhandy for practical use and undoubtedly requiring empirical modifications in its practical applications. It, therefore, seems more rational to carry out our first analysis while assuming certain simplified conditions and to introduce afterward such corrections as are required.

[1] Trans. of Am. Electrochem. Soc., Vol. 11, p. 117.
[2] Trans. of Am. Electrochem. Soc., Vol. 12, p. 161.
[3] Trans. of Am. Electrochem. Soc., Vol. 15, p. 270.

The assumptions made are:

1°. No exchange of heat takes place between the electrode and the surrounding wall.

2°. The electric current is confined to the electrode, no leakage into the surrounding wall taking place.

3°. The electric resistivity and the heat conductivity of the electrode are constant and independent of the variations of temperature.

4°. A uniform temperature prevails in each cross-section of the electrode.

We will use the following notation:

A = area of cross-section of terminal in sq. cm.
I = current in amperes.
a = length of terminal inside of the furnace in cm. (from E to B in Fig. 1).
l = length of terminal outside the furnace in cm. (from B to C).
d = current density in amperes per sq. cm..
t_1 = temperature of terminal in a section through E.
t_2 = temperature of terminal in a section through B.
T = temperature drop between E and B $(= t_1 - t_2)$.
Q = heat conducted from charge through the section at E into the electrode (in watts).

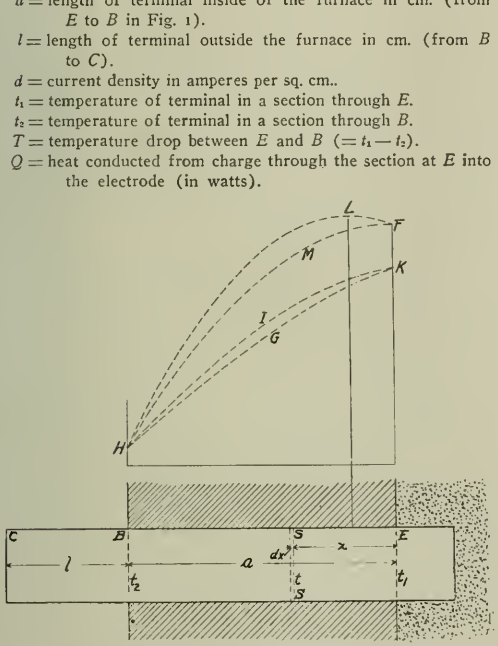


FIG. 1.—HEAT FLOW AND TEMPERATURE DROP IN FURNACE ELECTRODES.

W_1 = power in watts lost by heat conduction from the charge and on account of the electrical resistance of the terminal between E and B.
W_2 = power in watts lost on account of the electrical resistance of the outside part (from B to C) of the terminal.
h = heat conductivity of terminal.
r = electrical resistivity of terminal (per cm. cube).
k = heat equivalent of one watt (= 0.24 cal.).

The variables t and x signify the temperature t in a certain cross-section of the electrode at a distance x from the inside of the wall (E).

We will regard the quantity of heat passing through an imaginary, very thin slice of the terminal at the cross-section $S - S$. The thickness of this slice is dx and the temperature difference between its sides dt. The heat passing through it consists of the heat lost from the charge $(= kQ)$ and the heat produced by the electric current from S to E $(= krxI^2/A)$. Since the temperature difference between the sides of the slice, its thickness and area are known, the heat passing must also be equal $-hAdt/dx$. We obtain consequently the following differential equation:

$$-hAdt/dx = kQ + krxI^2/A \qquad (1)$$

By integration and from the facts that

$$t = t_2 \text{ for } x = a$$

and

$$t = t_1 \text{ for } x = o$$

and further

$$t_1 - t_2 = T$$

we obtain:

$$hA\,(t_1 - t) = (hTA/a - \tfrac{1}{2}\,kraI^2/A)\,x + \tfrac{1}{2}\,krI^2x^2/A \qquad (2)$$

$$Q = hTA/(ka) - \tfrac{1}{2}\,raI^2/A \qquad (3)$$

Equation (2) gives the temperature t in any section at a distance x from E. It represents a parabola with vertical axis.

The temperature drop per cm. (dt/dx) in any such section we obtain by derivation of equation (2):

$$-hAdt/dx = hTA/a - krI^2(a/2 - x)/A \qquad (4)$$

The left-hand term expresses the amount of heat passing through a given section at the distance x from E. The total loss of energy in watts in the part of the terminal inside the wall (E to B) by conduction from the charge and from the heat generated by the current we find by inserting a for x. We thus obtain:

$$W_1 = hTA/(ka) + \tfrac{1}{2}\,raI^2/A \qquad (5)$$

If no electric current was flowing through the electrode, the energy lost by conduction of heat from the charge into the electrode would amount to $hTA/(ka)$ watts, but on account of the current heating the terminal, the energy lost by conduction from the charge into the electrode is decreased by an amount equal to one-half of the heat generated by the current in the terminal. Thus we find that the heat generated by the current serves a useful purpose in keeping down the loss from the charge.

We will now consider the influence of the assumptions made in these deductions. According to our first assumption no exchange of heat should take place between the electrode and the surrounding wall. This condition will usually not be fully satisfied in practical working. However, as this subject will later be discussed at some length, it may at present be sufficient to say that the corrections to be made on this account will, as a rule, not affect our results very materially.

Regarding our second assumption that the current should be confined to the electrode, no leakage through the wall taking place, it probably in most cases answers the conditions in practical working so nearly that no corrections on this account are needed. One point should be raised, however, in this connection concerning the value assigned to the length a of the electrode. The electrode frequently enters the charge for some distance, the current in the meantime branching off into the charge. The current density will therefore gradually change in this part and the conditions assumed in our calculations will not be fulfilled. Therefore, the inside length of the electrode a should strictly be counted only to the point where the current begins to branch off into the charge. For practical purposes, however, it will be better to include also about one-half of the projecting end into the length a of the electrode.

The third assumption regarding the constancy of the conductivities of the electrode for heat and electricity is not fulfilled in practical operation, since, as well known, both increase to some extent with increasing temperature. By using mean values between the temperature limits the error which is committed will, however, be considerably reduced.

In this connection it is of interest to discuss somewhat in detail the distribution of temperature in the electrode. As already stated the heat passing through each section of the electrode will consist of the heat conducted away from the charge plus the heat generated by the current due to the resistance of the electrode between this section and the inside end of the electrode. As the distance from the inside end to the section under consideration increases, the total amount of heat passing through the section will, therefore, also increase. In order to render possible the passage of this increasing quan-

ity of heat, the temperature drop per centimeter must increase correspondingly. If the conductivities for heat and electricity of the electrode are constant, the temperature curve thus will assume the form of a parabola as shown (equation 2).

However, when these conductivities increase with increasing temperature, as is actually the case, the conditions are somewhat changed. We will still assume that no exchange of heat takes place between the electrode and the surrounding medium. If the working conditions are the same as in our first case (with constant conductivities) and the same amount of heat is conducted away from the charge, a smaller temperature drop per centimeter will now be needed at the inside end of the electrode to dispose of this heat. The amount of heat generated by the current per unit length of the electrode in this end will also be smaller than in the first case discussed, on account of the higher electric conductivity, and the increase in the temperature drop per centimeter, when gradually moving our point of observation from the inside end of the electrode, will consequently be correspondingly smaller, although growing also on account of decreasing heat conductivity. At a certain section of the electrode its conductivities for heat and electricity will correspond to the mean values assumed in the first case which we discussed.[1]

However, the amount of heat passing through this section is smaller than in the first case on account of the lower resistance of the inside part of the electrodes, and consequently the temperature drop in this section will still be smaller than in the first case. When passing further outwards the amount of heat generated by the current per centimeter of the electrode will now be larger than in the first case, due to the higher electric resistivity, while the lower heat conductivity also contributes to a more rapid drop of the temperature curve. Since the total electric resistance is the same in both cases (on account of our adoption of the mean value[2] as resistivity of the electrode in the first case) the same total amount of heat would be lost through the section at the outside end of the electrode in both cases.

In the second case the temperature drop per centimeter in this section, as well as in the part of the electrode near its outside end, will consequently be greater than in the first case. However, it is impossible to foresee, without an exact knowledge of the values of the conductivities at different temperatures, whether the temperature curves of the electrode would indicate the same temperature of the electrode in both cases at the outside of the wall. However, by defining the mean value of the heat conductivity in the first case in a special way with regard to calculations concerning electrodes, the temperatures of the electrode at the outside end will be identical in both cases. Curves *K-G-H* and *K I H* in Fig. 1 will give a general idea about the run of the two curves discussed.

The determination of the mean value of the heat conductivity as usually made by measuring the amount of heat conducted through an object while a constant temperature difference is maintained between its ends, will give values not suitable for our purposes.

When determining the electric resistivity of electrodes for practical purposes we measure the voltage drop and the current strength in working electrodes. The value thus obtained is not, however, the mean resistivity for the material of the electrode between the temperature limits occurring. If the latter was to be measured strictly the determination should be carried out with an electrode exposed only to a constant temperature difference, but not to a simultaneous heating by a current, using, for instance, a current too weak to influence the temperature distribution in the electrode. However, a value thus obtained would not be of practical usefulness for our purpose.

The same considerations apply when measuring heat conductivities of electrodes. In order to get values which may be directly applied in practical calculations, it will be preferable to carry out the measurements under practical conditions.

It was originally the intention of the writer to determine such values from experimental data by the aid of equation (5). For instance, an electrode (Fig. 2) may be surrounded by a highly refractory and well-insulating powder, such as magnesia. The waterjacket of the contact serves as calorimeter and the flow of water is regulated so that the temperature of the upper end of the electrode is the same as the temperature of the insulating packing at the same depth. The temperature of the electrode is also increased by means of a thermo-couple as far down as the couple will permit (1400° C.).

Under these circumstances the conditions assumed in equation (5) will hold with sufficient accuracy, and all measurements necessary for calculating from equation (5) the mean heat conductivity of the electrode may easily be made. By using such methods (properties of carbon) for determining the very high temperature of the hot end, as described by Fitz-Gerald (Trans. of Am. Electrochem. Soc., Vol. 6) and by Hansen (ib. Vol. 15) the total temperature drop in the electrode may, of course, be extended to much higher ranges.

From the nature of these methods for determining the mean

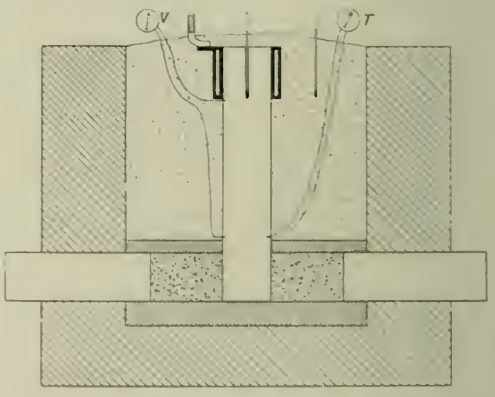


FIG. 2.—ARRANGEMENT OF TEST

conductivities for heat and electricity, it follows that by using such values no considerable error will be introduced on account of our third assumption regarding the constancy of these conductivities.

Our fourth assumption that a uniform temperature prevails in each cross section of the electrode will in most cases have only a small bearing on our deductions. At both ends of the electrode the isothermic surfaces will, of course, be somewhat disfigured, but for the greatest part of the electrode they will at least have a uniform shape and probably, as a rule, come close to being flat surfaces with some curving at their peripheries.

From this discussion of the assumptions involved in our deductions and the definitions made we have found that the energy loss, expressed by equation (5), will in certain cases probably correspond closely to actual conditions. The most serious objection to its correctness depends upon the assumption that no heat should be conducted away from the electrode to the surrounding wall. When this is the case the loss will be greater than indicated by the equation.

However, we may claim that the largest possible loss of energy from the charge into the electrode would occur if the heat produced by the electric current did not cause any decrease in the amount of heat which would be lost by conduction from the charge into the electrode in case no current was flowing.

[1] We assume that the graphic [illegible] of the curves in these conditions is identical of the [illegible] are the same.

[2] The [illegible] of the mean value is supposed to have been made by measuring voltage drop and current strength in working electrodes.

Such a case may be imagined if the surrounding wall would cause a sufficiently strong cooling of the electrode. The total loss would then amount to:

$$W_1 = hTA/ka + raI^2/A \qquad (6)$$

As evident, the only difference between equations (5) and (6) consists in the fact that in the first-mentioned case the term raI^2/a is multiplied with the factor $\frac{1}{2}$. We will now substitute for this factor a coefficient c, the value of which may, according to the working conditions of the electrodes, vary between the limits $\frac{1}{2}$ to 1, and consequently write the equation thus:

$$W_1 = hTA/ka + craI^2/A \qquad (7)$$

where h and r express mean values, as defined, with regard to our temperature range.

This equation, therefore, represents a general solution of the problem of energy losses of electric furnace electrodes independently of any assumptions regarding heat insulation of the electrode.

Regarding the value of c we have so far only stated that it is between $\frac{1}{2}$ and 1. Its exact value in each special case will depend upon a number of considerations.

The first condition for a transfer of heat from the electrodes to the surrounding wall is that there should exist a temperature difference between them at their places of contact. In this conection we must remember that the wall is also heated by conduction from the charge.

At the inside of the furnace wall its temperature will, therefore, be about the same as that of the electrodes, no exchange of heat taking place between them at this point. On its outside the furnace wall will assume a certain temperature. Here the temperature of the electrodes will depend upon the cooling effect of the contact plates, the holder, and, possibly, the water-cooling applied. If water-cooling is used, this temperature may easily be the same as that of the wall, or close to it. In such a case there will be only a very small exchange of heat between the terminal and the wall, depending upon the difference in their temperature distribution and c will therefore have a value of practically $\frac{1}{2}$.

However, if the temperature at the outside of the terminal is considerably higher than that of the wall we must figure with a certain transfer of heat from the terminal to the wall. Another circumstance, in conjunction with the higher heat conductivity of the terminal, will offset to some extent this transfer, namely, the fact that the heat passing from the terminal to the wall will have to travel a longer path than that going directly through the terminal. Besides, there is always a contact-resistance against the transfer of heat from the terminal to the wall.

The amount of heat conducted away from the electrode to the wall will also depend upon the relation between the circumference of the electrode and its cross-section and the relation between the heat conductivities of the material of the electrode and of the surrounding wall. It is obvious that this heat quantity will be larger the larger the circumference of the electrode is as compared with its cross-section. From this fact we may conclude that in case of electrodes with uniform cross-sections, working with the same current densities and otherwise under the same conditions, the one with a larger cross-section will cause a relatively smaller energy loss and consequently give a smaller value for c.

It is also evident that the shape of the cross-section has in the same way an influence upon the energy loss incurred. Of electrodes with the same cross-sections, working under the same conditions, a round electrode will cause the smallest loss. Of rectangular electrodes a square electrode will give the most favorable results in this respect. The more the cross-section of an electrode is extended in one direction the greater will be the loss by conduction to the wall. As an extreme case we may state with great probability that if h is the heat conductivity of the electrode and h_w the heat conductivity of the wall, the value of c, derived from the following equation:

$$c = \frac{1}{2}\left(1 + \frac{h_w}{h}\right) \qquad (8)$$

will represent an upper limit of the possible value of c.

If, therefore, the relation between these conductivities is 1 to 5, as probably in the case with amorphous carbon electrodes, made from petroleum coke, when laid in brick walls, c would probably have a highest possible value of 0.60. If graphitized electrodes are used under the same conditions, the relation of the conductivities would be at least 1 to 40 and c consequently have a highest possible value of 0.51.

If the circumference of the electrode is p (A being its cross-section), the formula given for c will apply when p/A has a high value, i. e., in case of very small electrodes. On the other hand, if p/A has a small value, as is the case with large electrodes, especially of a compact cross-section, c will assume a value tending to approach $\frac{1}{2}$. For commercial furnaces the value of c will therefore probably be located closer to $\frac{1}{2}$ than to the value found from equation (8).

Although in this way c is shown to be a function of two variables, i. e., the cross-section and circumference of the electrode, the possible variation of c appears to be comparatively small. Instead of attempting to introduce into our calculations an expression for c (for instance, $c = \frac{1}{2}[1 + h_w p/(hp + hfA)]$ where f is a constant) so as to take into account the considerations presented, thus complicating our deductions and conclusions in a high degree, we will therefore attribute a constant value to c, depending upon the type of the furnace and, within certain limits, upon the size of the electrode.

In order to give equation (7) a form more suitable for our deductions we substitute for A the expression: $A = I/d$ and thus obtain:

$$W_1 = I\,(hT/(kad) + crad) \qquad (9)$$

It should be noted that this energy loss is expressed in watts. As the current I is one factor in this expression it follows that the other factor represents the *total voltage loss* in the inside part of the electrode, due to the combined loss of energy by heat conduction from the charge and on account of the resistance of the electrodes.

As the total loss of energy W_1 consists of the sum of the loss on account of the resistance of the electrode $radI$ and the loss by heat conduction from the charge Q (i. e., the heat conducted from the charge into the electrode) we may get an expression for the last mentioned loss by means of equation (9):

$$Q = I\,[hT/(kad) - (1 - c)\,rad] \qquad (10)$$

Evidently, by using a certain current density, the conduction of heat Q from the charge into the electrode may be entirely avoided. This is the case if:

$$d = \frac{1}{a}\sqrt{\frac{hT}{kr\,(1-c)}} \qquad (11)$$

The electrode loss, when no heat is conducted away from the charge, amounts to:

$$W_1 = I\sqrt{hrT/k\,(1-c)} \qquad (12)$$

As the electrode loss in this case is caused only by its electric resistance, we may also express this loss in the following way:

$$W_1 = radI \qquad (13)$$

This expression may, of course, also be deduced from a combination of equations (11) and (12).

From equation (9) we also conclude that the total electrode loss will have the smallest possible value for a certain value of the current density. This value is:

$$d = \frac{1}{a}\sqrt{\frac{Th}{ckr}} \qquad (14)$$

The smallest possible loss, therefore, amounts to:

$$W_1 = 2I\sqrt{Tchr/k} \qquad (15)$$

We have considered so far only electrodes surrounded by an insulating wall. However, it is well known that electrodes may

enter the furnace also in other ways. Thus they may be suspended vertically in the furnace and the charge fed down around them, as in the case, for instance, in carbide or ferrosilicon furnaces. In other cases, for instance, in the steel furnaces of Heroult and Girod, the electrodes enter the furnaces through openings in an arched roof and do not come into direct contact with the charge.

In the first-mentioned case our formula may usually be applied. The surrounding charge will, as a rule, conduct more or less electric current. Its resistance, however, will be considerably higher than that of the electrode and, besides, the contact resistance between the terminal and the charge will prevent to a great extent a spreading of the current into the upper layers of the charge. The electric conductivity of the charge will, therefore, as a rule, not constitute any serious objection to the use of our formula.

Regarding the question of heat losses from the terminal to the surrounding charge we must consider the fact that any heat possibly given off from the terminal to the charge will preheat the charge and, therefore, cannot be charged as a loss.[1] At the same time it is of greatest importance that no heat should be conducted away by the terminal from the zone of highest temperature at its lowest end. This will depend upon the temperature distribution in the electrode.

At the lowest end of the electrode, where reduction and melting of the charge takes place, highly heated gases are formed which pass upwards through the charge around the terminal. These gases are therefore cooled and their heat is absorbed by the charge. In the lower part of the charge, near the end of the terminal, heat is also supplied to a great extent by conduction from the electrically heated zone. The temperature drop per centimeter (counted upwards) will therefore be small in this part. Higher up, where the heat is mostly carried by the gases, the drop will be faster.

It will be remembered from our discussion that in the first case considered the temperature in the electrode varied according to a parabola and that when no heat was lost from the charge by conduction through the electrode, this parabola had its center at the inside end of the electrode. This distribution of the temperature in the terminal, therefore, agrees in its general features with the temperature distribution in the charge as described.

There seems, therefore, to exist no principal objection against the application of our deductions to electrodes in furnaces in which the charge is fed down around the electrodes, due consideration being given in each case to the circumstances which influence the value of c. The drop in temperature T in our calculations represents in this case the temperature difference between the hottest end of the electrodes and the part of the electrodes at the top of the charge and a the distance between these points.

In the third class of furnaces, with electrodes freely suspended, the conditions are much more complicated. Our equations, however, undoubtedly may be applied also in this case. An account of the probable conditions prevailing in these electrodes, however, would involve so many uncertain assumptions that a detailed discussion of this subject may preferably be postponed until experimental data are available regarding the questions involved.

We have so far considered only the losses of the part of the electrodes inside the furnace. The protruding end of the electrodes will, of course, cause an additional certain loss due to its resistance. This loss may be expressed in watts by the following formula:

$$W_1 = rId^2l \qquad (16)$$

The electric resistivity r in this case has a higher value than obtains in the inside part of the electrode due to the lower temperature of the outside part.

As the outside end of the electrode should, from every consideration, be made as short as possible, we will not further enter upon a discussion of equation (16), but limit our deductions to the inside part of the electrode.

We may draw a number of conclusions of practical value from the equations derived. First, however, a general remark deserves attention. We have found that the heat conductivity of the electrode is of exactly the same importance to the working of the electrode as its electric resistivity. The electric resistivity has until recently been practically the only property which has been taken into consideration, both by furnace men and by manufacturers, especially when favorable as a "talking point." With electrodes of different materials, the loss is proportional to the square root of the product of the thermal conductivity h, and the electric resistivity r, other things being equal. If in case of two electrodes, A and B, made from different materials, the electric resistivity of A is only one-fourth of that of B, but the heat conductivity of A is eight times higher than that of B, then the smallest possible loss of energy in A would be 41 per cent higher than the smallest possible loss in B in spite of the lower electric resistivity of A. From the point of view of energy losses incurred, a low electric resistivity is therefore no inducement to the use of an electrode if its heat conductivity is very high.

In discussing the energy losses of electrodes the greatest interest, of course, centers around the equations expressing the most favorable conditions attainable with regard to these losses. Equations (11) to (15) refer to these conditions. It is obvious from these equations that when c possesses the value $\frac{1}{2}$, as probably is often the case, at least approximately the values of the current density d, according to equations (11) and (14), will be identical. In such a case, therefore, the same current density will give the smallest possible electrode loss and, at the same time, prevent any loss of heat from the charge by conduction through the electrode.

If c has a higher value than $\frac{1}{2}$, the electrode loss, equation (14), will be a minimum for a current density with which a certain amount of heat flows from the charge into the electrode. In order to avoid this loss it would, therefore, be necessary to raise the current density to a higher value, according to equation (11). If we assume that c has a value as high as 0.75 this raise of the current density would amount to 73 per cent, while the corresponding increase in the total loss of energy in the electrode would only amount to 15 per cent. If c, however, possesses a more normal value, such as 0.6, the corresponding percentages only amount to 22 per cent and 2 per cent. These figures for the increase of the energy losses, when compared with other losses in the furnace, are so insignificant and the advantages gained from these small sacrifices of energy so important that we may state it as a general rule that if other considerations do not interfere, the lowest current density, as given by equation (11), should be used which will cause no loss of heat by conduction from the charge into the electrode.

This loss will also be prevented if a still higher current density is used, but the total loss of energy will, in such a case, be further increased, as may be seen from equation (9). This case is interesting because it corresponds to certain results obtained by Mr. Hansen. He found in an experiment that the total loss of energy of his electrode was less than corresponded to the Joulean heat of the electrode as figured from its resistance.

It is obvious that the necessary requisite for avoiding any loss of heat by conduction from the charge is that in some section of the electrode there should be no temperature drop (per centimeter). For instance, in case of the electrode in Fig. 1, the tangent of the temperature curve at some point should be horizontal. To accomplish this it is only necessary to use a current density which will produce this turning point of the temperature curve right at the inside end of the electrode; the temperature is then at this end the same as prevails in the furnace (Fig. 1, curve $F_1M_1I_1$). This current

[1] As we use equation for W_1 include also the heat, conducted away by the surrounding medium, these expressions will therefore give somewhat too high values (cf. p. 24) in case of furnaces of this type.

density has been defined by our equation (11). If a higher current density is used, the top point (L) of the temperature curve F-L-H will fall within the wall and the heat generated from electrical energy will flow both ways from L, part of it being lost in the ordinary way, the other part being supplied to the furnace. This consequence of our deductions has been realized in the case of Mr. Hansen's experiment. Although Mr. Hansen was able in this way to get a smaller loss than corresponding to the resistance of his electrode, he nevertheless had a larger loss than necessary. With a somewhat lower current density, it would have been possible to avoid any heat conduction from the charge into the electrode with a smaller energy loss in the electrode.

There is another side to this question. If such a high current density is used that an overheating of the electrode occurs within the wall, the temperature attained will often ruin the wall, with consequent disastrous results to the furnace, certainly a sad experience of many electric furnace workers. On the other hand, if the same condition prevails in a furnace where the charge is fed down around a vertical electrode, the overheating of the electrode may cause a clogging of the charge around it and a wasteful leakage of the current. If in such furnaces two electrodes of different polarity are suspended side by side, the undue conductivity thus imparted to the charge between the electrodes will sometimes cause a short circuit and the current will not pass through the molten bath below the electrodes, as intended.

If we have electrodes with a thermal conductivity h and electric resistivity r, and want to use them in a furnace with a total temperature drop in the electrode T, and a current I, we may assume for practical purposes that c will have a certain fixed value independently of the length a we may give the electrode. Equations (11) and (12) may then be written as follows:

$$d = C_1 \sqrt{T/a} \qquad (17)$$

$$W_1 = C_2 I \sqrt{T} \qquad (18)$$

where C_1 and C_2 are two constants.

The product $C_2 \sqrt{T}$ in equation (18) represents the total voltage loss in the electrode when no heat is lost by conduction from the charge, the current density required for this purpose being given by equation (17). This current density is inversely proportional to the length of the electrode, while the voltage loss thus incurred is independent of the length of the electrode, remaining a constant for a certain total temperature drop in the electrode. A practical application of this rule is that if a certain current density has proved to give the most satisfactory results obtainable in a furnace, we may in this way compute the best value of the current density for another furnace of the same kind but with different length of the electrode. Because this principle has not been recognized there has certainly resulted much distress and surprise on the part of electric furnace men, the troubles being mostly due to the overheating of electrodes, already discussed.

With a certain total temperature drop in the electrode it is, therefore, feasible to incur the smallest possible loss of energy in the electrode whatever the length of the electrode may be, if only the current density is accordingly adjusted. In order to reduce the cost of electrodes, therefore, it would seem rational to use very short electrodes with a correspondingly high current density and to prevent an excessive heating of the outside end of the electrode by water-cooling. Although this method undoubtedly may be used with advantage to a larger extent than practiced so far, its application in most cases is limited by practical considerations.

In furnaces with fixed electrodes, surrounded by a wall, the method may sometimes prove useful if the water-cooled metallic holder is allowed to enter the wall for some distance. However, the risk of burning off the electrode, with all its dangerous complications and the difficulty in making good contact on the inside between the electrode and the charge or perhaps the permanent resistor will limit our endeavors in this direction. Besides, as the cost of the permanent electrodes in this class of furnaces is a comparatively small item, this possibility of using very small electrodes has no great bearing on electric furnace construction.

On the other hand, in many cases, especially with furnaces where the charge is fed down around the electrode, the best current density is dictated not only by electrode considerations, but also by the conditions at the place of transfer of the current from the electrode to the charge, where, as a rule, most of the heat in the furnace is generated. The electrode—its dimensions as well as its quality—should therefore be adjusted largely according to these conditions.

If a current density too high for the purpose of the process is used, the heat generated here may be too excessive to be usefully disposed of. Parts of the charge may be needlessly volatilized, products, already formed, may be decomposed, and undesirable reactions may take place. Besides, in such a case the electrode itself will be attacked and consumed to a much greater extent than necessary. It is probably a fact that from considerations to the first cost of electrodes a higher current density is used in many cases than consistent with the lowest consumption of electrodes per ton of output.

If we have furnaces of the same kind, but with different temperature drops T in the electrodes, the factors C_1 and C_2 assumed as constants in equations (17) and (18) will no longer remain so. The mean conductivity for heat h and the electric resistivity r will assume different values in the different cases. However, if there is no considerable difference in the total temperature drops we may assume that the relative increase in the mean thermal conductivity will be approximately the same as the relative decrease in resistivity and that, therefore, the agreement of their product will remain practically a constant. Under the same assumption there cannot exist any considerable difference between the values of c. Even if its different values varied as much as, for instance, from 0.55 to 0.60 the increase in the value of $\sqrt{1/(1-c)}$ would only amount to 6 per cent. We therefore find that the value of C_2 in equation (18) may be regarded as a constant for practical purposes when there are no great variations in the total temperature drop. However, the value of C_1, containing as a factor the square root of the quotient of h and r, will be influenced by a change in the total temperature drop to a degree which we cannot determine in advance. The only rule we can give at present for the most suitable current density in case of an increase of the total temperature drop in an electrode of a certain length a is therefore, that this current density should be higher than the one obtained by multiplying the best current density at a lower temperature drop by the square root of the quotient of the larger and the smaller temperature drops.

For the reasons given we may consider that equation (18) will hold with sufficient accuracy also in case of different temperature drops in the electrode within certain limits. As a rule, the inside temperature of a furnace used for a certain purpose is determined by the process worked. A comparison of the losses in the electrodes in case of different processes is therefore not of the same interest as an analysis of the possibilities offered to reduce the loss in a certain case. We have already at length discussed this subject, assuming a constant temperature drop. However, different temperature drops may occur in electrodes, having a definite temperature at their inside ends, on account of different temperatures being maintained at their outside end. Equation (18) may be safely applied to such cases. We may derive some important conclusions regarding the effect of water-cooling the outside end of the electrode upon the loss incurred.

The outside end of an electrode, exposed to the air, must not assume a temperature causing a burning of the electrode. If the protruding end would become red hot without water-cooling, its application is imperative. If the current density used corresponds to the value defined by our equation (11) or

(17), water-cooling in such a case cannot be regarded as causing an additional loss. It would be possible by giving the electrode a greater length and using a lower current density according to equation (17) to produce the same temperature at the outside end without water-cooling, but the loss, incurred in the electrode, would nevertheless be the same in both cases, as shown in our previous deductions.

A temperature drop reaching down to the burning temperature of carbon is unavoidable in any case, and we should, therefore, only charge the loss incurred on account of the additional drop in temperature against the water-cooling applied. This additional drop will, at the most, amount to about 400° C. If the inside temperature of a furnace is 2000° C., and the temperature drops in the electrodes are 1950° C. with water-cooling and 1550° C. without water cooling, then the additional loss, according to the equation (18), on account of the water-cooling, would amount to 12 per cent of the loss without such cooling. If the voltage loss, due to the electrode, is 1 volt without cooling, then the additional loss, caused by its application in this case, would only reach 0.12 volt. Besides, the higher the inside temperature of the furnace is the smaller will be the relative increase of the loss due to the water-cooling.

We therefore find that water-cooling, when applied to electrodes working with proper current densities, does not cause any significant losses. Those incurred are more than compensated for by the benefits.

It may be appropriate in this connection to discuss somewhat the practical advantage of water-cooling, as this subject is closely related to the conditions for the best working of electrodes. The purpose of the water-cooling is not limited to the purpose of preventing the burning of the electrode. Water-cooling is also of the greatest importance for the maintaining of a good contact between the cable connection and the electrode. As well known, such a contact is usually accomplished by firmly pressing metallic contact plates against the electrode. When possible (compare the article by FitzGerald in Vol. 3, 1905, page 14, of this journal) this contact is often improved by placing a layer of graphite between the plates and the electrode. In many cases, however, the application of such a layer is impracticable and the contact resistance produced may therefore be unduly high on account of the combined influence of a high specific contact resistance and of the fact that the plates and the electrode actually touch each other only at a few spots.

Without water-cooling a very strong heating of the contact place may follow, resulting frequently in oxidation of the plates. This increases the contact resistance still more with the consequences of a burning of the electrode and arcing contact. Water-cooling will largely prevent these troubles and therefore serve to keep the electrodes in running order. However, if the contacts have not been carefully made, water-cooling will conceal the bad condition of the contacts and the waste of energy on their account will be, as a rule, attributed to the electrode alone, if recognized at all. This serious cause of energy losses deserves more attention than is usually bestowed upon it.

As an example of the losses sometimes occurring on account of bad contacts, an occurrence within the writer's experience may be cited. The connection to a permanent electrode carrying 1800 amp was made by firmly clamping against two large copper plates with interposed layers of graphite ("decomposed carborundum"), the current density in the contact surfaces being only 11.4 amp per square inch. When the connection was first made, a voltmeter, permitting readings of 1/200 volt, only indicated a voltage drop of about 1.07 volt in the connection. After three runs of the furnace the electrode became excessively hot and on renewed reading of the voltage drop it was found to have risen to 1.47 volt. The voltage drop in the electrode itself amounted to 1.16 volt. This enormous energy loss in the contact was found to be due to a tarnishing of the copper plates from fumes emerging from the furnace. This instance may serve as a caution against similar occurrences of wasteful losses.

A short outline of a simple way to calculate the dimensions of electrodes, based upon the above formulas, may now be of interest.

In a given case the inside temperature of the furnace may be regarded as known. Although this is one of the fundamental quantities for any calculation of this kind, its knowledge is far from satisfactory in most cases on account of our limited knowledge of the temperatures required for the reactions or processes contemplated. Sometimes we may find from preliminary experiments some guide for estimating the temperature required. An examination of the electrode may give important clews, as shown by Mr. Hansen. However, the results obtained in this way must be interpreted with great caution if the experimenter has not an exact knowledge of the quality and composition of the electrode used. Otherwise an extensive investigation of the same kind as carried out by Mr. Hansen will be necessary.

However, we may find other indications of the temperature attained. The carborundum frequently formed on the tips of the electrodes may render such assistance. From the measurements of Tucker and Lampen (compare the synopsis in this journal, Vol. 4, 1906, page 363) we may conclude that the temperature in a given case has been between 1615° C. and 1950° C. if green amorphous carborundum is found on the electrode, but no crystalline carborundum. If crystalline carborundum has been formed, but no crystals of "decomposed carborundum" (graphite) are found, the temperature has been above 1950° C. and probably not exceeded 2220° C., but the graphitizing of the electrode tip does not necessarily indicate a temperature above 2220° C. Some other valuable indications may also be obtained from data given in the original paper of Tucker and Lampen (*Jour. Amer. Chem. Soc.*, July, 1906). It is to be hoped that an extended knowledge of the "chemistry of high temperatures" as well as of the physics of those temperatures will soon give us an exact basis for work of this kind.

Generally electric furnace workers have a tendency to exaggerate the temperatures prevailing in their furnaces. It is a common claim that the charge has been heated to the temperature of the arc, citing 3500° C. to 3700° C. as this temperature, because an arc was drawn in the furnace. The experimenter forgets that said temperature prevails when the arc is extended between electrodes of pure carbon with carbon vapor alone as conductor and not in a furnace usually filled with metallic vapors of comparatively high pressures. Further, the constituents of the charge and the products formed, as a rule, have a high vapor pressure at these temperatures, the sum of which may reach atmospheric pressure, preventing an increase of the temperature on acount of the subliming or boiling which then ensues.

Magnesia, one of our very best refractories, melts at 1920° C. (Goodwin and Maily, this journal, Vol. 4, page 217) and boils at a temperature shortly above its melting point according to Moissan, which statement is confirmed by observations of the author. In most extreme smelting processes we will probably not be far from the truth in assuming that the temperature will be somewhere between 1800° C. and 2200° C. When heating solid charges, such as certain carbides or carbon, the temperature may, of course, run up considerably higher.

When the inside temperature has been determined or estimated in one way or the other, the outside temperature must be decided upon. If water-cooling is used no appreciable error will be committed in placing this temperature at 100° C. Otherwise it may be about 100° C. or 200° C. higher.

The length of the electrode is the next consideration. This dimension will depend upon the working conditions of the furnace, and no general rule can be made for its proportioning except that the length is limited by the technical difficulties in manufacturing (preventing at present the manufacture of amorphous carbon electrodes of greater length than 7 ft.)

When the temperature drop T and the length a are thus fixed, the current density d will be the important object of our attention.

As already stated, the best current density is determined not only by the electrode, but also to a great extent by the purpose of the process worked. With regard to the last mentioned consideration we will usually have some latitude for fixing the current density, and we may therefore first investigate what density is most suitable on account of electrodes of different materials available. In order to do so we have to know the various physical constants of the electrodes and the electrode constant c of the furnace.

With a knowledge of the progressive and scientific spirit of the concerns manufacturing electrodes in the United States, we may confidently hope that they will carry out such investigations as will place to the disposal of their customers the information wanted regarding their electrodes.

For furnaces with an electrode constant c, which has the value $\frac{1}{2}$, we may calculate from equation (12) the voltage drop $\sqrt{hrT/k\ (1-c)}$, which corresponds to the temperature drops T when no heat is conducted away from the charge. Experiments should then be carried out with the same kind of furnace and using the calculated current densities to check whether the energy actually lost corresponds satisfactorily to the quantity calculated from equation (13). If this is the case a very nice check has been obtained for the correctness of the determinations made.

The next step in the investigation required is to determine the energy losses in the electrodes under working conditions of furnaces in order to find the value of c under different conditions.

Such experiments will surely offer certain difficulties, which, however, may be overcome. When the value of c is known in a given case, we may find the smallest voltage drop required in the electrode in order to avoid heat conduction from the charge by multiplying the voltage drop for $c = \frac{1}{2}$ with the correction factor.

Having access to the results of these determinations, the designer may draw a curve showing the smallest voltage drops in the electrode at different total temperature drops, when no heat is conducted away from the charge, assuming the value $\frac{1}{2}$ for c equation (12).

In a given case in practice, characterized by a certain total temperature drop and a special value of c, he will take from the curve the voltage drop, corresponding to the temperature drop in question, and multiply this voltage drop by the correction factor $\sqrt{\frac{1}{2}\ (1-c)}$. He will find in this way the voltage drop which should prevail in his electrode, regardless of its length, in order to avoid heat-conduction from the charge into the electrode.

By resorting now further to equation (13), he finds that this voltage drop should equal rad, and since the mean electric resistivity r of the electrode at this total temperature drop is supposed to be known and the length a has been decided upon, the most suitable current density d, with regard to the electrode, is found by a simple division.

It is possible, and even probable, that this current density does not correspond to the value, desirable on account of the process to be carried out. In such a case, of course, the original assumed working conditions of the electrode may be adjusted in order to satisfy both considerations, or some other compromise may be effected. It is also perhaps possible to select electrodes of another and more suitable material in such a case.

In one way or the other way a set of conditions may thus be worked out from calculations supported by practical experiments, which will give ultimately the best economical efficiency of the furnace electrodes with regard to the electrode losses and costs and to the process worked.

HÖGANÄS, SWEDEN, September, 1909.

Cyanidation of Manganese-Silver Ores.

Some of the metallurgical difficulties which have been encountered in the cyanidation of manganese-silver ores in Mexico are brought out in two communications by Mr. ROBERT LINTON (of Los Angeles, Cal.) and Mr. E. M. HAMILTON (the well-known metallurgist of the Butters interests) to the *Journal of the Chemical, Metallurgical and Mining Society of South Africa.* They are published in the issues of March and August, 1909, respectively.

The problem with this special class ore is to unlock the silver in the manganiferous ore so as to make it soluble in potassium cyanide solutions. The insoluble silver is contained principally in the highly oxidized surface ore between the croppings and the 200 level, while ore from the bottom of the mine usually yields 90 to 95 per cent of the silver without any trouble.

Mr. ROBERT LINTON found a solution of the problem in preliminary chloridizing the ore. By treating with potassium chlorate and hydrochloric acid, or roasting with salt and sulphur, and following this treatment by weak KCy solutions in the usual manner, a uniformly high rate of extraction was obtained.

For operation on a practical scale Mr. Linton proposes to apply the common laboratory method of preparing chlorine, using salt, manganese dioxide and sulphuric acid according to the reaction:

$$2NaCl + MnO_2 + 3H_2SO_4 = MnSO_4 + 2HNaSO_4 + 2H_2O + Cl_2.$$

The salt was mixed with the ore, crushed through 100-mesh screen. The sulphuric acid was added in weak solution (5% H_2SO_4) and allowed to stand with occasional agitation for 24 hours. The solution was then filtered off, the residues washed and treated with 0.2 per cent KCy solution. The silver extraction varied from 60 per cent to 94 per cent. The method has the advantage of low cost of materials; commercial sulphuric acid and salt are cheap, and the manganese dioxide is already present in the ore. So far the method has been applied by Mr. Linton on a laboratory scale only.

Mr. E. M. HAMILTON, in his quite extended experiments with manganese-silver ore, has reached somewhat different conclusions. He confirms the difficulty of dissolving the silver. He tried cyanidation with all strengths of cyanide, even up to 5 per cent, on samples of the ore ground to pass 200 screen, and could only extract from 5 per cent to 15 per cent of the silver. He tried the addition of all kinds of reagents, such as lead salts, oxidizers, chlorides, oxygenation with air, hot solutions, etc., without effect.

Since the difficult ore is from the highly oxidized rock, while the ore from the bottom of the mine offered no difficulties, Mr. Hamilton concluded to treat the difficult ore first with a reducing agent. He accordingly subjected the finely ground ore to treatment with solutions of various reducing agents, among which were hydrogen sulphide, sodium sulphide, sodium hydrosulphide, and roasting with sulphur in a closed vessel. The charges were agitated for about 24 hours, and then well washed with water and cyanided by agitation in bottles.

The sodium sulphide had no effect whatever, but both the ammonium and sodium hydro-sulphides and the hydrogen sulphide increased the subsequent silver extraction to about 73 per cent against about 5 per cent by straight cyaniding. The cyanide consumption was also increased from about ½ lb. per ton in straight cyaniding to 12 lb. per ton for the samples treated by hydro-sulphide and to 27 lb. for that treated by hydrogen sulphide. The resultant cyanide solutions carried large quantities of thiocyanate.

The samples roasted with sulphur have about the same yield as those treated with hydro-sulphide, and about the same cyanide consumption.

To try to reduce this heavy loss of cyanide, the ore, after reduction treatment, was given various treatments before cyanidization, among which were the application of chlorine water and various chlorides and of oxidizers, such as potassium permanganate and hydrogen peroxide, and also the blowing of air through the

pulp; they were then well washed before cyanidation. In the case of intermediate oxidation tests some cyanide was saved in the subsequent treatment, but the extraction of silver was considerably less than when the ore was cyanided immediately after the reduction treatment. The intermediate chlorine treatment made no difference in the cyanide consumption, but increased the silver extraction by 3 per cent or 4 per cent.

Owing to the excessive cyanide consumption and the low value of the ore ($8.00 to $10.00 per ton) this method of treatment by reducing agents could not be entertained, although if the cyanide loss had been normal the method would have been well worth considering. The reason why this treatment increased the silver extraction was rather mysterious, and attempts were then made to ascertain the mode of occurrence of the silver.

By grinding the ore to pass 200 screen and boiling for an hour in strong nitric acid, then washing well with distilled water and assaying the residue, the latter was still found to contain 75 per cent of the original silver content. This residual silver insoluble in nitric acid could not have been in the metallic or sulphuric forms, nor was it mechanically inaccessible in the ordinary sense, because a sample of the pure clay slime from the ore was even more refractory to nitric acid than the general sample. It might have been in haloid form, but in that event it should have been soluble in cyanide or sodium thiosulphate, which reagents had practically no effect on it.

About this stage of the inquiry Mr. Hamilton obtained the assistance of Mr. G. H. Clevenger, who made many experiments, of which the following is a short summary: (1) Bromocyanide, results rather inferior to those by straight treatment. (2) Addition of zinc dust as a reducing agent, no result. (3) Addition of metallic sulphides with a view to breaking up any hypothetical haloid compounds, no result. (4) Addition of strong alkali with and preliminary to cyanide treatment, no result. (5) Preliminary treatment by pan amalgamation, no result. (6) Preliminary treatment with chlorine and bromine prior to cyaniding, no result. (7) Preliminary roast before cyaniding, result—not a trace of silver soluble in cyanide. (8) Preliminary chloridizing roast, result—fair silver extraction (75%). (9) Concentration of the ore ground to 200 mesh, result—tailing assayed rather higher than concentrate. (10) Oil concentration, no result. (11) Acid flotation, no result. (12) Preliminary treatment with hydrosulphides, results confirmed the tests made by Mr. Hamilton as described above. (13) Preliminary treatment with hydrochloric acid, result—94 per cent extraction of the silver.

From the foregoing it appeared that the silver is in very intimate association with some other mineral, probably with the manganese, possibly occurring as a definite manganese-silver compound.

As to how the silver may be made amenable to cyanide solution, however, Mr. Hamilton concludes that chloridizing is not the only, and probably not even the best, method. He is led to think that instead of trying to attack the silver, which is comparatively small in quantity, the aim should be to attack the predominant partner in the combination—that is, the manganese.

In the above detailed tests a chloridizing roast yielded only 75 per cent of the silver on subsequent cyanidation, while the preliminary treatment with hydrochloric acid allowed an extraction of 94 per cent. Of course, in this latter instance it may be assumed that the result was due to nascent chlorine formed by the manganese dioxide; it might, however, be equally well assumed to be due to the dissolving action on the manganese, exposing the contained silver to the action of the subsequent solvent, and this view would seem to be strengthened by the fact that the chloridizing roast gives results so much inferior to those of hydrochloric acid.

But if this latter view be correct, how are we to account for the results due to a preliminary reduction with hydro-sulphide? The only suggestion Mr. Hamilton has to offer is that the manganese, being converted into sulphide, became sufficiently soluble in the cyanide solution to expose most of the silver to the action of the solvent, forming thiocyanate and mangano-cyanide of potassium.

Working on the hypothesis of the necessity of finding some solvent that would break up the combination and free the silver, he tried a preliminary treatment with many reagents. (1) Nitric acid had been found useless in the early stages of the investigation, probably because it did not attack the manganese dioxide. (2) Hydrochloric acid, while very effectual, appeared to be prohibitive in cost. (3) Sulphuric acid was without effect. (4) The hydro-sulphide treatment, while being effectual, was prohibitive on account of the loss of cyanide, though he found it possible to regenerate KCN from KCNS by electrolysis with caustic soda, and it might have been possible to recover some of the cyanide in this way. (5) Finally he tried 5 per cent solution of sulphurous acid. After washing and cyaniding the residue he obtained an extraction of 84 per cent of the silver with a cyanide consumption of only 4 lb. per ton of ore. This method seemed by far the most feasible of any, and if the ore had assayed a few dollars higher than it did, the method would certainly have been tested on a working scale. The amount of sulphur required would in this instance have been about 100 lb. per ton of ore, which was more expense than the grade of rock would have warranted.

Thus Mr. Hamilton concludes from the foregoing results that there are several methods of treatment besides chloridising which were found to give good cyanidizing extractions on the silver in this manganiferous ore: (a) Preliminary roasting with sulphur in absence of air. (b) Preliminary treatment with alkaline sulphides. (c) Preliminary treatment with hydrochloric acid (which may be considered to fall under the chloridizing class). (d) Preliminary treatment with sulphurous acid.

Of all these the last seems to Mr. Hamilton the most practicable and least expensive, and with an ore assaying a little higher than that which was experimented on, would seem to offer a workable solution of the difficulty in the cyanidation of these manganese-silver ores. It may be remarked that the gold in such ores usually yields a uniformly high extraction of about 90 per cent under almost any circumstances, regardless of the behaviour of the silver.

Why Is Electro-Steel Sometimes Unhomogeneous?

By Assar Grönwall.

Several steel manufacturers in England, Germany, and the United States have found that electrically produced steel is at times unhomogeneous. In one and the same ingot the one end may be of high quality, while the other end is quite inferior. This complaint has been made too often to accept the usual explanation that in the beginning of electric steel manufacture part of the steel which was placed on the market was not yet up to perfection, and that ever since the conservative crucible-steel manufacturers, unwilling to abandon their crucible furnaces, emphasized on all occasions the lack of homogeneity of electric steel.

The main part of the high-class steel hitherto produced in the electric furnace has been made from cold raw materials, viz., pig iron and scrap. The furnaces used have principally been of the following three types: First, single phase electrode furnace with two adjustable electrodes (Heroult). Second, single phase electrode furnace with one adjustable electrode and one bottom electrode (Girod). Third, induction furnace.

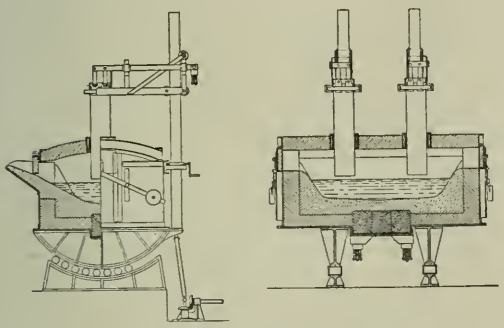
An investigation of the principles of operation of the different systems will give an explanation why lack of homogeneity is sometimes found in the steel. As to the Heroult furnace, after the main part of the charge has been molten the current passes from one electrode through an arc to the slag, then through the slag and the upper part of the metal, and through another arc back to the second electrode. The main part of the heat is produced in the two arcs, a smaller part in the slag, and only a very small part in the metal. Thus the metal is heated

through conduction from the slag, and may therefore not be uniformly heated; the surface is always hotter than the metal near the bottom, as the hotter metal is lighter than the cooler.

Therefore it may happen that if, for instance, a large piece of scrap has been thrown in the furnace with or after the other charge, or if such a piece has come to be lodged in a corner of the smelting chamber, this piece may be barely molten when the charge seems to be ready for tapping, to judge by the look of the slag or from the tests. If, then, the metal is tapped, the hottest and purest steel will run out at first, and the cooler unrefined metal at last.

In steel furnaces with one adjustable electrode and a bottom contact, just the same can happen; however, this type is in this respect more advantageous, since some part of the heat here is developed in the bottom.

In the induction furnace the same conditions may also exist. In it the electrical energy is converted into heat in the bath itself. If the lining is properly made, the cross-section of the bath is the same all over the annular hearth, and the development of heat is uniformly distributed. But if, for instance, after some time the lining in some place is eaten away and thus the cross-section enlarged, less heat is developed just at that



SWEDISH ELECTRIC STEEL FURNACE.

place where rather more heat is wanted, since more metal is accumulated there. This unequal temperature at different parts of the bath may, of course, lead to non-homogeneity of the steel.

A new type of electric steel furnace recently patented and placed on the market by the Swedish Elektrometall Company seems to fulfill better the conditions for getting a homogeneous steel. This furnace is shown in the adjoining illustration. The electrical energy is supplied in form of two-phase currents, one of the terminals of each of the two phases being connected to the two adjustable electrodes, while the two other terminals are both connected to the bottom contact. The two-phase currents produce a rotating magnetic field in the bath, which causes a circulation of the metal sufficient to get a homogeneous product and a uniform temperature in all parts of the bath. Since the circulation takes place here in a vertical plane, the metal near the bottom is continuously brought up to the surface and the hot slag.

In all the above types of furnace it is important, when producing high-class steel, that the time for treating one charge is sufficiently long. A prolonged treatment also makes sure that the metal is completely freed from gases and slag. This is of great importance for the quality of the steel, although it is as yet impossible to prove the presence of gases by analysis.

It is, of course, necessary to tap all the steel into a ladle before tapping into the moulds in order to get the metal thoroughly mixed. The author has seen at one steel works that the steel was tapped from the furnace (an induction furnace) into a small ladle, taking only a quarter of the charge, so as to run metal into the moulds, placed on a rotating table, simultaneously with the tapping from the furnace. Since the metal could not possibly be well mixed, this procedure is not recommendable.

Finally, it is to be added that the production of high-class steel always requires the greatest accuracy as well as long experience, and the production of electric high-class steel is by no means an exception from this rule. The attendance of an open-hearth furnace is never left to a person with less than one year's experience, and it is not to be expected that it is otherwise when dealing with an electric furnace. The electric furnace has been in practical use for too short a time. There are to-day not yet many men thoroughly familiar with the running of an electric steel furnace for high-class steel. But after some years' experience the electric furnace will displace the crucible furnaces, which are too uneconomical.

Calculations for Dryer Design*

BY DR. WILLIAM M. GROSVENOR.

The following methods were developed as a matter of commercial necessity. They were needed to design a great number of drying installations adapted to meet widely different requirements, operating under a system the advantages of which have hitherto been largely overlooked, i. e., the repeated circulation of the same air over condensing coils to remove moisture, over heating pipes to increase the "moisture capacity," over the material being dried and again over the condenser.

Surprising steam economies are thus accomplished, in some cases amounting to more than 80 per cent. Insurance risks are reduced. So-called "moist drying" becomes a matter of course, great capacity is permitted, and complete independence of climatic conditions makes accurate design and perfect regulation possible.

The engineering of drying is to-day one of the most neglected and by no means one of the most simple branches of applied science. There are thousands of installations where the air travels 10 feet in idleness to one of useful occupation, where it is taken from the streets and the gutters and blown loaded with bacteria over fermentive products (many of them food stuffs), where it is received ice-cold and heated up 50° or 100° before it begins to do its profitable work, or is drawn in nearly saturated at 85° to dry a product that cannot safely be heated above 95°, where, worst of all, it is expected to reverse the performance of the widow's oil jug, and to keep on drinking up moisture indefinitely without renewal or dehydration. Taking advantage of the variable weather conditions, which the seller of machinery had no object in controlling, the machinery was made big enough—the bigger the better.

In developing simple, rapid, and at the same time reasonably accurate methods of design, we were met at once with the fundamental difficulty that there is no simple equation expressing the relation between the tension of aqueous vapor and the amount of water, either by volume or weight, saturating air under given conditions.

The next difficulty was that the base or divisor (cubic foot) of all humidity calculations has hitherto been most unfortunately selected, as it varies with every change of pressure, temperature, or humidity, needlessly complicating all calculations. Lastly, that "percentage humidity," more properly called "degree of humidity," is not really comparable percentage at all. Ten per cent of humidity means that 1 cu. ft. of mixture contains one-tenth the amount of water which a cubic foot of another mixture containing less air would carry if saturated under the same conditions of temperature and pressure.

While such a standard of humidity may be satisfactory for meteorological records, practical design in a field thus obstructed was a dangerous thing to turn over to subordinates, and it seemed advisable to begin by getting rid of stumpage and doing some grading. Finally there was marked disagreement in published tables of the essential constants.

The last question was first taken up. The original sources were examined and an entirely new set of calculations made

*An abstract of a paper presented at the Pittsburg meeting of the American Institute of Chemical Engineers. The complete paper is published in Vol. I. of the Transactions of the American Institute of Chemical Engineers.

from vapor tensions to weight of water per cubic foot, using the following formulæ and constants:

Density of air + .04% CO_2 = $\frac{1.293052}{1 + .00367 \times \text{Temp. C.}}$ in kg per cu. m.

Density of water vapor = 62186 × density air.

Ratio of density at partial pressure to density at 760 mm = 1/760 × partial pressure in mm.

Wt. in kg of 1 cu. m of residual air = 1/760 × part. press. × density of air (in kg per cu. m).

Wt. in kg of 1 cu. m of water vapor at part. pressure = 1/760 × 0.62186 part. pressure × density of air.

Specific heat water vapor = .475

Specific heat of air = .2373.

TABLE I. HUMIDITY TABLE.

Temp F	Vapor Tension in Millimeters	Lbs. Water Vapor per Pound Air	Humid Heat in B T U	Humid Volume in Cu Ft	Density in Lbs. per Cu Ft @ 760 Millimeters		Specific Volume in Cu. Ft. per Lb of—	
					Dry Air	Sat'd Mix	Dry Air	Sat'd Mix.
1	2	3	4	5	6	7	8	9
32	4 569	003761	2391	12 462	080726	080556	12 388	12 414
33	4 746	003917	2392	12 491	080559	080377	12 413	12 442
34	4 949	004076	2392	12 521	080393	080198	12 439	12 469
35	5 152	004238	2393	12 549	080233	080025	12 464	12 496
36	5 354	004415	2394	12 578	080069	079852	12 489	12 523
37	5 572	004593	2395	12 606	079910	079680	12 514	12 549
38	5 795	004778	2396	12 636	079748	079511	12 539	12 576
39	6 023	004968	2397	12 666	079585	079343	12 565	12 604
40	6 264	005168	2398	12 695	079420	079181	12 590	12 629
41	6 503	005373	2399	12 724	079272	079012	12 615	12 655
42	6 764	005589	2400	12 754	079113	078849	12 640	12 682
43	7 026	005805	2401	12 784	078954	078677	12 666	12 710
44	7 301	006030	2401	12 813	078797	078514	12 692	12 736
45	7 582	006270	2403	12 843	078641	078348	12 718	12 763
46	7 874	006513	2404	12 872	078485	078178	12 741	12 791
47	8 177	006763	2405	12 904	078329	078011	12 766	12 819
48	8 486	007020	2407	12 935	078172	077844	12 792	12 846
49	8 811	007292	2408	12 966	078019	077678	12 817	12 874
50	9 140	007567	2409	12 999	077867	077511	12 842	12 901
51	9 488	007860	2410	13 030	077714	077344	12 868	12 927
52	9 841	008157	2411	13 062	077562	077178	12 893	12 957
53	10 210	008468	2413	13 095	077409	077016	12 919	12 984
54	10 580	008765	2415	13 127	077258	076851	12 944	13 012
55	10 980	009116	2416	13 159	077109	076685	12 968	13 041
56	11 386	009458	2418	13 192	076926	076522	12 993	13 068
57	11 801	009864	2420	13 224	076810	076363	13 019	13 096
58	12 235	010165	2421	13 258	076661	076198	13 044	13 124
59	12 674	010546	2423	13 292	076512	076032	13 070	13 152
60	13 138	010939	2425	13 326	076363	075865	13 095	13 180
61	13 608	011338	2427	13 360	076215	075698	13 121	13 210
62	14 100	011755	2429	13 395	076069	075529	13 146	13 240
63	14 603	012172	2431	13 431	075924	075362	13 171	13 270
64	15 122	012625	2433	13 467	075780	075198	13 196	13 298
65	15.640	013081	2435	13 501	075635	075036	13 222	13 325
66	16 209	013547	2438	13 535	075493	074882	13 246	13 354
67	16 783	014043	2440	13 572	075347	074718	13 272	13 384
68	17 363	014539	2442	13 609	075201	074552	13 298	13 413
69	17 975	015065	2443	13 645	075062	074382	13 323	13 441
70	18 595	015597	2447	13 683	074921	074219	13 348	13 471
71	19 242	016154	2450	13 721	074778	074054	13 373	13 502
72	19 903	016724	2452	13 759	074636	073890	13 398	13 532
73	20 585	017312	2455	13 797	074496	073735	13 424	13 562
74	21 249	017922	2458	13 837	074363	073570	13 449	13 593
75	22 000	018545	2461	13 876	074218	073405	13 474	13 624
76	22 759	019198	2464	13 916	074079	073239	13 499	13 654
77	23 537	019867	2467	13 956	073940	073072	13 524	13 685
78	24 315	020554	2470	13 997	073801	072909	13 549	13 715
79	25 124	021260	2474	14 039	073664	072749	13 575	13 746
80	25 965	021998	2478	14 081	073531	072585	13 600	13 777
81	26 724	022752	2481	14 123	073395	072419	13 625	13 809
82	27 711	023532	2485	14 167	073258	072248	13 650	13 841
83	28 625	024319	2488	14 211	073123	072080	13 675	13 873
84	29 557	025124	2492	14 255	072987	071913	13 701	13 906
85	30 529	026026	2497	14 301	072852	071744	13 726	13 938
86	31 519	026898	2501	14 347	072716	071574	13 752	13 971
87	32 518	027818	2505	14 399	072584	071404	13 776	14 004
88	33 582	028750	2510	14 442	072443	071234	13 801	14 038
89	34 667	029724	2514	14 490	072321	071063	13 827	14 072
90	35 774	030718	2519	14 539	072189	070894	13 852	14 106
91	36 913	031743	2524	14 588	072058	070724	13 787	14 139
92	38 082	032810	2529	14 638	071926	070556	13 903	14 173
93	39 283	033896	2534	14 688	071794	070390	13 929	14 206
94	40 528	035029	2539	14 740	071664	070223	13 954	14 241
95	41 794	036174	2545	14 793	071535	070051	13 979	14 275
96	43 122	037388	2550	14 846	071407	069878	14 004	14 310
97	44 434	038603	2556	14 900	071279	069704	14 030	14 346
98	45 824	039817	2562	14 956	071149	069530	14 055	14 382
99	47 208	041078	2569	15 013	071021	069356	14 080	14 419
100	48 670	042357	2575	15 071	070894	069170	14 106	14 455
101	50 171	043655	2581	15 131	070767	068984	14 131	14 492
102	51 691	044982	2588	15 192	070641	068822	14 157	14 530
103	53 271	046376	2595	15 253	070514	068645	14 181	14 568
104	54 861	047801	2603	15 317	070387	068464	14 207	14 606
105	56 514	049273	2611	15 379	070264	068288	14 232	14 643
106	58 199	050782	2618	15 440	070141	068110	14 257	14 682
107	59 938	052351	2626	15 503	070018	067926	14 282	14 722
108	61 741	053996	2634	15 572	069894	067744	14 307	14 761
109	63 578	055634	2641	15 641	069751	067564	14 333	14 803
110	65 479	057327	2651	15 711	069647	067383	14 358	14 840
111	67 424	059092	2660	15 781	069525	067192	14 384	14 883
112	69 419	060956	2671	15 854	069403	067002	14 409	14 924
113	71 459	062842	2677	15 931	069281	066821	14 434	14 965
114	73 551	064811	2686	16 004	069159	066647	14 460	15 005
115	75 699	066873	2695	16 084	069037	066464	14 485	15 048
116	77 895	068941	2704	16 164	068916	066281	14 510	15 084
117	80 146	071113	2713	16 245	068794	066111	14 535	15 127
118	82 451	073514	2722	16 329	068681	065871	14 561	15 179
119	84 821	077121	2741	16 411	068551	065756	14 585	15 225
120	87 257	078902	2755	16 496	068431	065477	14 611	15 272
121	89 457	081055	2767	16 583	068315	065315	14 635	15 319

Temp. F	Vapor Tension in Millimeters	Lbs. Water Vapor Per pound Air	Humid Heat in B. T. U.	Humid Volume in Cu. Ft.	Density in Lbs. per Cu Ft @ 760 Millimeters		Specific Volume in Cu.Ft. per Lb of—	
					Dry Air	Sat'd Mix.	Dry Air	Sat'd Mix.
1	2	3	4	5	6	7	8	9
122	91 978	085622	.2780	16 679	068207	.065082	14 661	15.365
123	94 581	088398	2793	16 777	068090	064884	14 687	15 412
124	97 207	091208	2806	16 870	067973	.064689	14.711	15 460
125	99 924	094147	2820	16 968	067857	.064480	14 736	15 509
126	102 688	09716	2835	17 069	067741	064277	14 762	15 558
127	105 518	10027	2849	17 172	067625	064075	14 788	15.606
128	108 428	10350	2864	17 278	067510	063870	14 813	15 652
129	111 382	10680	2880	17 386	067394	063663	14 838	15 708
130	114 437	11022	2896	17 499	067280	063440	14 863	15 761
131	117 516	11368	2913	17 612	067166	.063234	14 888	15 814
132	120 724	11742	2931	17 730	067052	.063022	14 914	15 867
133	123 960	12121	2949	17 851	.066938	062809	14 939	15 921
134	127 304	12514	2968	17 976	.066826	062591	14 964	15.977
135	130 702	12917	2987	18 103	.066713	062374	14 989	16.032
136	134 182	13335	3006	18 235	066601	062158	15 015	16 088
137	137 749	13768	3026	18 370	.066489	061939	15 040	16.145
138	141 372	14122	3047	18 508	.066377	061717	15.065	16 204
139	145.114	13678	3070	18 653	.066267	061487	15 090	16 264
140	148.885	15150	3093	18.800	.066156	061255	15.116	16.325
141	152 807	15652	3117	18 952	066045	061022	15.141	16 387
142	156.762	16161	3141	19 108	065935	060790	15.166	16 450
143	160 841	16696	3166	19 270	065825	060554	15.192	16.512
144	164 986	17245	3192	19 447	065712	060318	15.217	16 576
145	169.227	17816	3219	19 609	.065601	.060082	15 242	16 643
146	173.569	18409	3247	19 787	065498	060842	15 267	16 710
147	177.976	19017	3276	19.971	065390	.060599	15 293	16 780
148	182 522	19659	3307	20 161	065283	060353	15 318	16 846
149	187 103	20310	3338	20 354	.065176	059106	15 343	16 919
150	191.860	21005	3371	20.559	065068	058865	15 368	16 993
151	196 654	21710	3404	20.767	064961	058618	15 394	17 068
152	201 595	22455	3440	20.987	064856	058349	15 419	17 141
153	206 610	23221	3476	21.211	.064750	058092	15 444	17 215
154	211 759	24019	3514	21.445	.064644	057833	15 469	17 291
155	216 983	24854	3553	21.687	.064539	057570	15 494	17 370
156	222 305	25713	3594	21.936	.064435	057305	15 520	17 450
157	227 786	26623	3637	22.201	064328	057036	15 545	17 533
158	233 308	27546	3681	22 466	064224	056767	15 570	17 616
159	239 034	28542	3728	22 754	064120	056497	15 596	17 701
160	244 803	29553	3776	23.045	064016	056218	15 621	17 788
161	250 742	30627	3827	23.382	063913	055938	15 646	17 877
162	256 767	31737	3880	23 670	063810	055656	15 671	17 968
163	262 923	32900	3936	24 001	063707	055372	15 697	18 060
164	269 213	34123	3994	24.349	063605	055084	17 722	18 154
165	275 592	35395	4054	24 708	063502	054795	15 748	18 250
166	282 155	36713	4118	25 090	063400	054507	15 773	18 349
167	288 764	38106	4183	25 478	063298	054203	15 798	18 449
168	295 609	39602	4254	25 900	063198	053901	15 823	18 545
169	302 504	41125	4327	26 300	.063098	.053598	15 848	18 654
170	309 593	42756	4105	26 790	.062997	053292	15 874	18 761
171	316 781	44472	4487	27 267	062899	052983	15 899	18 875
172	324 120	46257	4570	27 768	062799	052676	15 924	18 986
173	331 612	48162	4666	28 301	062700	052367	15 949	19 101
174	339 207	50140	4754	28 855	.062599	052032	15 975	19 218
175	347 015	52285	4856	29 454	062500	051708	16 000	19 339
176	354 873	54472	4960	30 063	063402	051382	16 025	19 462
177	363 003	56991	5076	30 738	062304	051049	16 050	19 589
178	371 199	59386	5194	31 428	.062205	050715	16 076	19 719
179	379 598	62097	5322	32 180	062112	050378	16 101	19 851
180	388 121	64942	5456	32 967	062015	050035	16 126	19 987
181	396 815	67985	5602	33 810	.061915	.049693	16 152	20 126
182	405 686	71265	5758	34 717	061816	049338	16 177	20 269
183	414 674	74703	5921	35 666	061719	048985	16 202	20 415
184	423 904	78419	6102	36 718	061621	048626	16 227	20 566
185	433 191	82430	6288	37 796	061520	.048265	16 253	20 719
186	442 793	86911	6501	39 029	061424	.047898	16 278	20 879
187	452 456	91535	6721	40 302	.061320	.047529	16 303	21 041
188	462 374	96731	6967	41 729	061224	.047154	16 328	21 209
189	472 422	1 0227	7230	43 251	061149	.046774	16 353	21 381
190	482 668	1 0835	7519	44 918	.061033	.046391	16 379	21 557
191	493 111	1 1510	7843	46 768	.060963	.046003	16 404	21 739
192	503 808	1 2230	8190	48 730	.060867	.045611	16 429	21 925
193	514 745	1 3067	8579	51 021	.060774	.045213	16 454	22 119
194	525 .358	1 3931	8990	53 402	.060681	.044813	16 480	22 319
195	536 744	1 4998	9494	56 302	.060588	.044405	16 505	22 527
196	548 093	1 6108	1 0023	59 469	.060496	.043997	16 530	22 742
197	559 731	1 7456	1 0718	62 978	060404	.043586	16 556	22 963
198	571 519	1 8912	1 1559	67 024	.060311	.043169	16 581	23 166
199	583 511	2 0629	1 2471	71 496	.060219	.042742	16 606	23 393
200	595 771	2 2480	1 3467	76 304	.060127	.042306	16 631	23 638
201	608 155	2 4968	1 4584	82 197	.060036	.041868	16 656	23 896
202	620 921	2 1087	1 5665	91 768	.059944	.041425	16 682	24 182
203	633 867	3 1189	1 7186	100 499	.059855	.040982	16 707	24 481
204	646 841	3 6509	1 9139	115 216	.059764	.040533	16 732	24 825
205	660 111	4 2272	2 1562	133 028	.059674	.040075	16 758	25 194
206	673 719	4 9276	2 4681	155 272	.059585	.039610	16 783	25 648
207	687 480	6 0698	3 1236	190 [illegible]	.059495	.039143	16 808	25 540
208	701 516	7 6744	3 8918	125 621	.059406	.038673	16 834	25 860
209	715 895	11 1295	6 2189	162 091	.059317	.038192	16 859	26 280
210	730 267	15 8174	13 9148	562 054	.059228	.037318	16 884	26 796

Kg per cu. meter × .062428 = lb. per cu. ft.

Results of these calculations are given for every degree Fahrenheit in Table I (Humidity Table).

These results (Col. 3, Table I) differed by as much as 4 per cent from the various published figures, but showed a surprising agreement (¼ per cent due to allowance or 0.04 per cent CO_2) with the figures of the United States Weather Bureau. The reason for this was inquired into. Quoting from the Psychrometric Tables of the United States Weather Bureau of 1900 (prepared by Dr. C. F. Marvin, Professor of Meteorology), it will be seen how carefully the Weather Bureau's tables were prepared:

"The vapor pressures employed were derived from Broch's reduction of Regnault's observations. All the values given by these tables at temperatures below the freezing point are noticeably higher than Regnault's observations. In view of this systematic discordance and the further circumstance that Regnault's experiments did not include observations at the extremely low temperatures frequently recorded at Weather Bureau stations, the writer (Dr. Marvin), in 1891, made a new determination of the maximum pressure of aqueous vapor at low temperatures.

"In the course of these experiments it was found an easy matter to reduce the temperature of the water employed many degrees below 32° Fahr., without freezing it, and in these cases the vapor pressure was higher than the pressure from ice at the same temperature. Independent experiments in Sweden, by Julius Johlin, at about this time, led to the same results.

"A comparison of the vapor pressures derived from the several sources mentioned shows that Broch's computed values at low temperatures do not agree at all well with Regnault's experiments, from which they are derived, whereas the experimental results of Regnault, Johlin, and the writer (Dr. Marvin), agree very closely.

"At temperatures below 32° Fahr., therefore, it has been considered necessary to reject Broch's values, and the vapor pressures over ice, as deduced from the writer's experiments, have been used in the calculation of the tables. At temperatures above 32° Fahr. the values taken from Broch's tables are employed. At 32° Fahr. the value for the vapor pressure found by the writer from the mean of a large number of experiments was identical with that of Broch; hence, there is no break in the continuity of the two tables at the point of junction."

The calculated weight of water per pound of air does not agree with actual determinations, but shows progressive differences, possibly owing to association of molecules in the vapor, which are too small in magnitude, however, to affect our present purpose.

Regarding this difference Dr. Marvin writes:

"In all ordinary circumstances it is assumed that the expansion and contraction of partially saturated aqueous vapor is in accordance with the same laws as apply to air and ordinary gases, which do not easily condense to the liquid state.

"The adopted density of saturated aqueous vapor is not determined directly from experiment, but is deduced theoretically from the observed fact that two volumes of hydrogen and one of oxygen combine to produce two volumes of water vapor.

"The weights of unit volumes of hydrogen, oxygen, and dry air are accurately known, from which the specific gravity of aqueous vapor is found to be 0.6221."

The compared figures from various sources were submitted to Dr. Marvin, but he replied that, "The Weather Bureau has not attempted to measure the quantity desired directly; in fact, the difficulty of experimentally maintaining a condition of absolute saturation and avoiding supersaturation, or mechanically suspended moisture on the one hand and superheating on the other, is so great as to perhaps render the results thus obtained less acceptable than the theoretical values."

We must, therefore, accept these figures as generally recognized throughout the United States and at least as well established as our purpose requires.

New Definitions of Humidity, etc.

In the next place a graphic method of calculation was desired, for rapidity and freedom from gross error, much more dangerous than slight inaccuracy. Also it was sought to visualize as far as possible what actually occurred to simplify comparison. A number of trials proved it far more satisfactory to disregard entirely the question of volume until it finally became necessary to calculate cross-sections of air ducts, passages, etc.

A great gain was made by regarding "humidity" as "the pounds of water carried by 1 lb. of air (or equivalent weight of other gas having the same volume) when saturated at a given temperature and pressure," and regarding the "percentage humidity" as "the weight of water actually carried by a pound of air divided by the maximum which could be carried by that pound of air under the same conditions."

This suggested as a substitute for specific heat something which we called the "humid heat," namely, "the b.t.u. required to raise 1° Fahr. 1 lb. of air, plus whatever water vapor it might carry when saturated at the temperature and pressure," and a "percentage humid heat" varying regularly with the "percentage humidity" between that of air alone and that of the saturated mixture corresponding to the same conditions.

Similarly we obtained a new "humid volume," i. e., the volume of 1 lb. of air when saturated under known conditions, and a corresponding "percentage humid volume" varying with the percentage of saturation between the specific volume of 1 lb. of dry air and the volume of 1 lb. of air saturated under the same conditions. We use the usual "density" plotted as shown and a "percentage density" (which does not vary proportionally with percentage humidity) only for calculating velocities in $V = k\sqrt{h\frac{d-d}{D}}$, for gravity circulation plants where k is always uncertain and damper regulation can be provided.

It has been simply delightful to find the ease and facility with which calculations could be carried out when dealing with conditions thus defined.

Given a mixture (let us say saturated 90°), all we need to do for the heat capacity is to pass horizontally across to the "humid heat" curve (see chart I accompanying this issue as a supplement) and if we want the heat capacity of any unsaturated mixture, we find the corresponding temperature and percentage of saturation, dropping vertically downward (merely a change in temperature) until we reach the saturation curve and then horizontally across to the heat capacity of this saturated mixture. If we want the volume or density of any saturated mixture it is directly given by horizontally crossing to the "humid volume" or "density" curves.

One difficulty (not yet overcome in any other way so far as the writer has been able to learn) is presented by the question: "How much moisture will a saturated (or unsaturated) mixture, which has been heated to a certain temperature and is used for the drying of any material, take up before it becomes saturated (or 90 per cent or 80 per cent saturated, if we prefer) and to what temperature will it fall?" It is necessary, at present, to assume the specific heat of water vapor as constant,[1] 0.475, though there is both theoretical and experimental evidence that under these conditions it acts as superheated steam with a variable specific heat.

But assuming it constant, the law of cooling is obviously

$$\frac{dx}{dt} = \frac{1.438\,S}{1601 - t}$$

in which x represents the weight of water per pound of air, t represents the temperature (Fahrenheit) at which evaporation is stopped, and S the specific heat of the mixture which we are heating and which is to be cooled by the evaporation it causes. But we have not any similar equation to express the line of saturation, 90 per cent saturation or any other degree, and we cannot solve our problem mathematically. The latest German

[1] "Specific Heat and Total Heat of Superheated Steam," Geo. A. Orrok, "Power," Aug. '04, p. 426.

works on this subject give us nothing but a clumsy method of interpolation which requires a preliminary calculation of cumbersome tables and repeated trial calculations to select values which will approximately satisfy the two requirements.

We can, however, represent the direction of this "cooling by evaporation" graphically by the "lines of cooling." It is apparent that no matter what the maximum temperature may be, if the condition of our preliminary mixture is represented by a point on any one of the curves of percentage saturation, the action of heating it is fully portrayed by elevating that point directly upward to the line of maximum temperature (neither its specific heat nor "humid heat" is changed).

The direction of its fall in cooling dx/dt will be the same no matter whether saturated at 75°, or 50 per cent at 96°, dependent upon its original air and water components expressed by the distance from the axis of Y (as governed by the numerator $1.148S$) and upon the temperature where it finally ends up in the course of its cooling (as governed by the denominator $1601-t$). In considering the cycle it is perfectly correct to assume all the evaporation to have taken place at this final temperature.

We have, therefore, drawn for various final temperatures on our diagram a series of straight diagonal lines on the same Humidity Curve Sheet representing dx/dt for various initial mixtures, the water content of which is given by the distance from the Y axis where the left-hand end of each line ceases. On the left-hand side of the curve, owing to the slight difference of inclination of corresponding lines for successive final temperatures, it is necessary to draw only one of these fan-shaped sets of lines and these need have been drawn only for each successive 10° of temperature.

Here the saturation curves show that the temperature varies rapidly and water content (with corresponding specific heat) varies slowly. On the right-hand portion of the curve, where the variation of "water absorbed" (and of specific heat) is greater, they are needed every 5°.

The lines for intermediate points can be drawn if desired, but it will be noted that a difference of 5 produces a difference of value of $dx\ dt$ of about ½ of 1 per cent.

Now, it is apparent that with these lines upon our curve we may take any given mixture which is to be heated (whether saturated or not to start with), run up the vertical to the maximum allowable temperature and selecting from the nearest group of lines of cooling that one which starts from the nearest position with reference to the Y axis, we may follow down parallel to this to the saturation curve.

If we are not considering the complete saturation, but only the 70 per cent saturation of the final exit mixture (before cooling and condensation) this is only a first approximation. Upon noting at what temperature the diagonal intersects the curve of 70 per cent saturation we must then redraw our incline, parallel to the incline corresponding to this mixture for this nearest final to the one thus discovered.

Example Showing Use of Curves.

As an illustration of the use of these curves, assume 50 lb. per minute of water to be removed at a temperature not exceeding 110° Fahr. with available air at 80° Fahr. and 90 per cent saturated. Assume the material to be so distributed and exposed that the air will leave the dry room at 80 per cent saturation. How much air will be required? What will be its final temperature, etc?

On the chart we find 80° Fahr. ordinate crosses the 80 per cent humidity dotted curve at an abscissa of 0.0200 lb. water per pound of air. If heated to 110° the change of condition is represented by increased ordinate with no change of abscissa (moisture content), the point rising vertically to 110°, where we find it between 50 per cent and 60 per cent humidity curves, or approximately 54 per cent saturated.

On exposure to the material being dried, the air will cool and take up moisture, changing its condition parallel to the inclined straight line to the immediate right of the point, 0.0200 lb. 110°.

This diagonal shows that it will be 50 per cent saturated at 101°, 70 per cent saturated at 96°, and 80 per cent saturated at 90°, with an abscissa of 0.0248 lb. water per pound of air, having picked up $0.0248 - 0.0200 = 0.0048$ lb. water per pound of air.

To remove 50 lb. water per minute would require $50 \div 0.0048$, or about 10,000 lb. of air per minute. To ascertain the heat required for this entering air we require the heat capacity of air at 80°, 90 per cent saturated.

Starting from the same point as before, we drop to the 100 per cent saturation curve, and find that our mixture has cooled to 77¼° Fahr., and passing on the horizontal of 77¼° toward the left, to the Humid Heat curve, we find that each pound of air requires 0.248 b.t.u. per degree Fahrenheit to heat it and its water of saturation. We have then $0.248 \times (110 - 85°) \times 10{,}000 = 62{,}000$ b.t.u. per minute.

To find the volume of 10,000 lb. of air, let us say at the exit, where each pound is carrying 0.0248 lb. of water with it and is 80 per cent saturated, we have only to pass across from the point 80 per cent, 0.0248 lb., 90° Fahr. (where we ended the first operation), along the horizontal of 90° to the Specific Volume and the Humid Volume curves. The specific volume of dry air at 90° is 13.86, that of saturated air (Humid Volume) 14.55, and 80 per cent of this difference is $0.69 \times 0.8 = 0.552$ added to the volume of dry air $= 13.86 + 0.55$, or 14.41 cu. ft. per pound $= 144{,}410$ cu. ft. air per minute at the exit.

The lines at present shown upon the sheet have amply sufficed for all purposes of practical design and have made it possible almost at a glance to select the conditions best suited to a given problem. They show vividly, for instance, the great advantage secured by applying heat between successive partial saturations by progressive exposure to the material being dried —also the enormous economy of restricted ranges of temperature.

Finally, we are developing therefrom, with little labor or calculation, a series of curves showing for every 10° Fahr. entrance temperature, the maximum conditions of efficiency, assuming various saturations of air before and after passing over the material, and a locus of maximum efficiency for these entrance and exit assumptions.

Heat Losses from Pipes.

The auxiliary curves (see chart II. Loss of Heat from Pipes, accompanying this issue as a supplement) are intended to simplify calculations of heating pipes and present a graphic solution of the law of Dulong and Petit, as modified by Peclet and Box. The use of the curves is extremely simple and is shown by arrows. They enable us to obtain the two components, radiation and convection loss, from pipes ⅛ in. to 12 in. U. S. standard size, per foot of length singly or in any of the 10 positions which they may occupy in a bank.

The fundamental basis of these calculations (0.64 b.t.u. radiation and 0.05 b.t.u. convection) are not accurate by several per cent, and cannot remain constant, as the pipes rust or gather dust. No effort, therefore, has been made to secure greater accuracy than given in the figures of Box. A safety factor has been added to the result given by the curves and the size of this factor is being gradually reduced by experience.

In our designs hitherto, the velocities of gases across the heating pipes has always been practically that due to natural circulation, and no provision is shown here for variable velocities. It can easily be added by a single inclined line if desired.

(The original paper of Dr. Grosvenor contains here several tables of figures representing the derivation of the curves of Chart III. These tables are here omitted, since the results are given completely on Chart II.)

The losses by convection for each standard size pipes are plotted with temperature difference as ordinates and b.t.u. per hour as abscissæ in the lower half of the curves of Chart II.

The presentation of radiation is somewhat complicated by the fact that the correction factors vary both with temperature and with difference of temperature. These correction factors are represented by the fan-shaped set of curves on the upper half of the diagram.

It is frequently desired to use pipes in banks wherein position greatly affects radiation, though not influencing convection. Modified factors reduced in proportion to the radiating angle obscured by the other pipes of the bank are, therefore, calculated (assuming Crane's Standard malleable return bends are used for assembling) for the various possible positions indicated by Nos. 1 to 10 on the diagram accompanying the chart. These various factors for the isolated and for the various partly obscured positions are given on the chart. The point to which the diagonal for this graphic multiplication should be drawn is indicated on the main guide line, but only a few of the diagonals to these points have been drawn, as our own particular interest centers mainly on combinations of 1-in. pipe. The other lines can easily be drawn when required.

The simplified calculation of wall, ceiling and window losses was found to be well worked out in curves drawn by Mr. Burt S. Harrison and published in the *Heating and Ventilating Magazine* of November, 1907.

Similarly the capacity and drop in pressure in steam mains has been carefully worked out by E. C. Sickles and published in *Power*.

The design of the condenser remains for consideration. It is a surprising fact, but a most thorough search has revealed no data applicable to this case. We use banks of horizontal 1-in. pipe with water or brine circulating within them. If they were merely cooling air we could find ample data for design. They would transmit about 1.5 b.t.u. per square foot per degree per hour. If they were condensing practically pure steam, some quite recent work would be available. They would transmit from 150 to 300 b.t.u. per square foot per hour. But they do neither and yet they do both. A formula has been developed upon purely theoretical grounds similar to those of the law of mass action. It will be submitted later (not as a solution, but as a view for criticism and test), together with examples showing how far it is justified by actual practice.

Notes on Electrochemistry and Metallurgy in Great Britain.

(From our Special Correspondent.)

Societies and Institutions.

The Faraday Society has delayed its opening meeting of the season until November 30th, and the Physical Society's annual exhibition of apparatus has also been fixed for a later date than usual, the 14th of December. The inaugural address of Dr. Gisbert Kapp, president of the Institution of Electrical Engineers, dealt with generators, transformers, power transmission and electric railways, but did not touch on any metallurgical subject. His remarks as to the necessity of educating the potential users of electrical plants are worthy of consideration; but many will share the view expressed in a recent article in the *Times* that "it is to be regretted that Dr. Kapp, instead of covering such an extensive field in his survey, did not devote his address almost exclusively to the enormous benefits to be obtained through the more rapid application of electricity to textile and other factory purposes."

The Reduction of Some Refractory Metals from Their Chlorides.

In a paper recently presented to the Chemical Society by Messrs. J. Norman Pring and W. Fielding, the authors have described their investigations of the reactions by which certain refractory metals may be prepared by the action of electrically heated carbon rods or filaments on their volatile chlorides. When tungsten chloride was used without admixture, under diminished pressure, decomposition occurred at temperatures between 1200° and 1500°; and with molybdenum chloride, below 1300°, adherent deposits of the pure metals being formed on the carbon. At higher temperatures carbides were produced; but when small amounts of aqueous vapor were present the metals were reduced free from carbide. Pure crystalline silicon was obtained at temperatures between 1700° and 1850° by addition of hydrogen to the vaporous chloride. At first a thin layer of carbide was formed; and this prevented carbonization of the subsequent deposit until the temperature was raised to 1900°, when carbide alone was formed. In the case of boron, the carbide B_6C, was always deposited at temperatures between 1800° and 2000°. Approximate photometric determinations of the radiating power of the metal-coated carbon showed a very great increase in luminosity for a fixed consumption of power, amounting with tungsten and molybdenum to nearly three times the radiation from carbon rods. Deposits of silicon and of boron carbide caused only a very slight increase.

A New Use for Electrolytic Hypochlorite.

Dr. F. W. Alexander, medical officer of health for Poplar, has been carrying out a series of experiments on the purification of the water in the borough swimming baths by the addition of electrolytic disinfecting fluid. The large first-class swimming bath has a capacity of 75,000 gallons, and it was found that the addition of 100 gallons of electrolytic hypochlorite solution in four installments—40 gallons on the Saturday, 40 gallons on the following Monday, 10 on Tuesday night and 10 on the Wednesday—had the effect of rendering the water perfectly clear and fit for use, whereas the bath is usually emptied and cleaned on Tuesday and fresh water run in. After a total of 899 persons had used the bath on Wednesday the water became slightly clouded, but the bottom of the bath was distinctly visible. On the Thursday night after 1159 persons had used the bath the water was discolored and the bottom somewhat obscured. The superintendent reported that the effect of the addition of the electrolytic fluid was to preserve the water in as good a condition after five days' use as it would have kept in one day's use only without such addition; and the secretary of a swimming club stated that the members felt fresher, not languid, and that they had lost the usual "dead feeling" on coming out of the bath.

One of the laundresses employed in cleaning the swimming costumes reported that those articles had no unpleasant odor since the fluid had been used, but previously they gave off a most unpleasant smell. The chemical and bacteriological report by a Fellow of the Institute of Chemistry on the bath water without hypochlorite shows that the chlorine present as chlorides was 2.5 grains per gallon as against 2 grains per gallon present in the East London water supplied. Some nitrites were present; the amount of oxygen required to oxidize the organic matter present was 0.1 grain per gallon, which is almost the figure given by an ordinary slightly polluted water. Qualitative tests showed the presence of B. Coli communis, Streptococci, and Staphylococci; but B. Enteritidis Sporogenes was not present. Quantitative tests showed approximately 10,000 organisms per cc. capable of growing at blood heat, and 2000 capable of growing at ordinary temperatures. The preponderance of the former indicates that they are chiefly of animal (human) origin. Samples of the water to which hypochlorite had been added were found to contain no nitrites; free chlorine estimated at 0.5 mg. per litre; and after incubation for four days no organisms were found growing in the cultures made, which result might have been anticipated from the fact that free chlorine was present.

In the course of these experiments a peculiar fact came to light, namely, that the water used by women and girls was always dirtier than that used by the same number of men and boys; and after investigation the conclusion was arrived at that this was due to dust accumulated in the women's long hair. The other baths of the Corporation were also treated with hypochlorite, and the same satisfactory results were obtained; and

the masters and mistresses who attended with the boys and girls from the schools said the water was fresher, more invigorating, and had lost its body odor. It is proposed to use about 38 gallons of electrolytic fluid to the bath of 75,000 gallons. The cost of this quantity would be about 55 cents.

For some time past the public health officials of the borough of Poplar have been using electrolytic hypochlorite instead of carbolic acid for general disinfecting purposes, and have found it combines higher efficiency with considerably reduced cost

Company Meetings.

The Castner-Kellner Alkali Company.

This company's fourteenth ordinary general meeting was held on November 25th, under the presidency of the Chairman, the Rt. Hon. G. W. Balfour, who congratulated the shareholders on the financial statement. The net profit—but without deduction for debenture interest or depreciation and reserve—was £134,154, which was an increase of more than £18,000 on last year's profit. This result was more satisfactory than appeared at first sight, because this year was by no means a prosperous one in the chemical trade, whilst the increase arose in almost every product which the Company turned out, which must be taken as indicating the broad basis of prosperity on which the company rested. Deducting debenture interest and the interim dividend paid in May, the net available balance is £117,711. On last year's accounts, as in previous years, loss was written off for realization of disused plant. In 1907 the amount was about £16,000. Usually it was a comparatively small sum, but this year it did not appear at all. Losses in respect of realized plant differ from usual depreciation, and this is partly the reason why the sum for depreciation has been raised from £30,000 to £35,000. On suspense account the sum of £1,370 had been written off, and as a consequence that account disappeared. A suspense account had been necessary because investigations were inevitable which it would not be fair to charge to revenue of a single year; but now that the general reserve fund was established it was to be hoped that the suspense account would not be revived. The general reserve fund was started last year, and it is proposed to carry £25,000 thereto, making its total amount £37,000 which is, in fact, a cash reserve, invested in good, easily realizable securities, and is available for the purpose of a suspense account or for dividends in less prosperous years. This arrangement being agreed to, there was a sufficient balance to pay a dividend of 9 per cent, making with the May interim dividend 14 per cent for the year, and leaving £15,840 to carry forward. The chemical trade seemed to be improving, and there was no reason to anticipate a worse result for the current year's accounts than the last year's had shown.

The report and accounts were received and adopted, and a vote of thanks to the Chairman closed the proceedings.

The Aluminium Corporation, Ltd

A scheme for the transfer of the assets of this company to a new company was sanctioned in the Companies' Court on November 23d by Mr. Justice Swinfen Eady. The petition stated that the company was incorporated with a capital of £500,000, of which £440,000 were in 7 per cent participating cumulative preference shares and £60,000 were in ordinary shares with a non-cumulative preference of 7 per cent, whilst 40 per cent of the remaining profits went to the preference shares and 60 per cent to the ordinary. The £1 preference shares issued were 257,205, of which 252,222 were fully paid. There was a debenture issue of £26,906, and a large unsecured debt. The company was wound up, and liquidators were appointed in 1908; and in May of that year a scheme of arrangement was sanctioned, but was not carried out through lack of money; nor was an attempt to realize the company's assets successful. The proposed scheme provides for the payment of the general creditors, and unsecured creditors for less than £10, in full; unsecured creditors for more than £10 would receive preference shares in the new company, and shareholders would get part-paid shares. There was no opposition; and the scheme was sanctioned, subject to slight amendment and the consent of shareholders to accept shares in the new company.

London, Dec., 1909.

Synopsis of Periodical Literature

Iron and Steel.

Electric Steel Furnace of Girod.—The chief interest in an illustrated article on the Girod furnace, by Professor W. Borchers, in *Stahl und Eisen*, of Nov. 10, lies perhaps in the somewhat personal remarks in which he expresses his enthusiasm concerning this type, which he considers to be "the simplest of the present commercial electrode steel furnaces." The keynote of the invention of Girod he finds in the way in which the steel bath is made a resistor, producing itself Joulean heat. Since the Girod furnace has been described and illustrated quite often in this journal (for instance, in our Vol. VII, pp. 9, 250, 323, 371, 458) and since Professor Borchers' article contains little that is new, only a few data on the new Girod electric steel works in Ugine (first described in our Vol. VII, p. 9, and of great interest, as being the first large and entirely electric steel works) may be given here, as Borchers has recently visited them. At present there are installed in these works three furnaces, each of two tons capacity, and two larger ones, each of 10 to 12 tons capacity. Castings up to 20,000 kg weight can be produced. Other electric steel furnaces of Girod are in use or in course of erection at the works of Oehler & Company, Aarau, Switzerland; John Cockerill Company, in Seraing, Belgium; Metz & Company, Stuttgart, Germany; Stahlwerk Becker, Krefeld, Germany; Gutehoffnungshütte, Oberhausen, Germany; Turnauer Stahlwerk von Schoeller, Ternitz, Austria; Danner & Company, Judenberg, Austria; Hungarian State Steel Works, Diosgyor, Hungary. (A Girod furnace is also reported to be in course of erection in a steel plant in this country.) The same article by Borchers was printed in *Metallurgie*, Nov. 8. The issue of Dec. 8 of this journal contains a letter by the Elektrostahl G. m.b.H., Lindenberg, which criticises various points in Borchers' article and endeavors to show the superiority of the Héroult furnace over the Girod furnace. The same issue contains a reply by Professor Borchers in which he states his reasons for considering the Girod furnace to be the superior one. Both letters are of a rather personal nature.

Armature Steel and Permanent-Magnet Steel.—A suggestive study of steel for the electrical industry is given by G. Mars in *Stahl und Eisen*, Oct. 27 and Nov. 10. For armatures and transformers a steel is wanted which has a high permeability and a low hysteresis loss—that is, metallurgically speaking, a steel which has a high content of ferrite and a coarse structure indicating a loose agglomeration of the molecules (so as to permit them to follow rapidly the reversals of magnetic polarity). This condition is fulfilled by a steel containing 2 to 3.5 per cent silicon and a very small amount (less than 0.1 per cent) carbon. The requirements of steel for permanent magnets are, of course, quite different. Here high remanence is chiefly desired. In carbon steels the remanence increases with increasing carbon content up to 0.97 per cent C and then decreases. The hardness increases and reached a maximum at 0.8 to 0.9 per cent C and then remains practically constant. The maximum of remanence has been found by Mme. Curie at 1.20 per cent, but the author's figure of 0.97 per cent C seems more probable. Tungsten steels are, however, far superior to carbon steels for permanent magnets. The best steel for permanent magnets at present

used contains about 0.60 per cent C. and 5 per cent tungsten. This superiority of tungsten steels over carbon steels is due to the presence of more free iron in the former with equal hardness. A tungsten steel of the stated composition gets highest remanence if the hardening temperature is 930° C. to 950° C. The paper contains interesting suggestions on the relation between metallurgical structure and magnetic properties.

Effect of Titanium on Steel.—An interesting paper on this subject is published by ED. VON MALTITZ in *Stahl und Eisen* of Oct. 13. In his introductory remarks he mentions some experiments made five years ago in a basic open-hearth furnace of the Dominion Iron & Steel Company in Sydney, Canada, where he used titaniferous iron ore (containing 40 per cent Fe and 16.8 per cent titanic acid) as an addition for the purpose of increasing the fluidity of the basic slag and improving the desulphurization. Metallic titanium did not enter thereby into the steel bath, but the fluidity of the slag was considerably increased without detriment to the furnace lining; a distinct influence on the removal of sulphur was also observed. The experiments were, however, not carried far enough to reach final conclusions. The author discusses in his article especially the use of ferro-titanium for steel refining. Since this matter has been discussed at length in an article by Mr. C. V. Slocum, in our Vol. VII, p. 128, and in another article on the results obtained with titanium-treated steel rails (Kessler's curve, etc.) in our Vol. VII, p. 498, we mention only the new points in Mr. Maltitz's article. Ferro-titanium, containing from 10 to 15 per cent Ti is preferably used. It is added after the steel has been recarburized and after ferro-maganese or ferro-silicon has been added. Care is to be taken that the ferro-titanium is not mixed with the slag. It should be dissolved in the steel bath itself for the purpose of removing oxygen and especially nitrogen, the resulting compounds passing into the slag. Reference is made to the detrimental effect of nitrogen on steel according to the work of H. Braune (our Vol. V, p. 51). Maltitz found in converter steel for rails 0.013 to 0.014 per cent nitrogen. When the same steel was treated with titanium, the nitrogen content was reduced to 0.004 or 0.005 per cent. This is a very important result, since "nitrogen should be considered to be at least as detrimental as phosphorus." Some figures from practice are given showing that a very homogeneous steel is obtained by means of treatment with titanium. In most cases an addition of ½ per cent of ferro-titanium containing 10 to 15 per cent Ti is sufficient. While this improvement of the quality of steel is important, it should be remembered that a material which cools quietly and without developing gases, tends to the formation of pipes, like crucible steel, if care is not taken to cast at a moderate heat. This precaution is, therefore, important. Steel treated with titanium cools much more slowly than ordinary steel; the reason for this is not clearly understood. But it is also a fact that articles of steel which have been treated with titanium are heated less than ordinary steel under otherwise equal conditions; for instance, cast-iron or steel rolls, which have been treated with a small amount of titanium, can be machined more quickly than ordinary steel rolls, on account of the smaller increase of temperature due to machining. As was noticed in our former articles, some remarkable results have been obtained with titanium-steel rails on difficult sections of railroads. These are reviewed at some length by Mr. Maltitz.

Oxy-Acetylene Welding of Sheet Iron.—In *Stahl und Eisen* of Nov. 17, H. RINNE reports on the results obtained in practice by the sheet-rolling mill of the Schulz Knaudt Company in Essen, who have used for many years the water-gas welding process. On account of its reliability and cheapness they continue to use this method for longitudinal cylinder seams. During the last years they have also employed oxy-acetylene welding, which has been found especially useful for repairs of leaks developed in seams produced by the water-gas process. The main advantage of oxy-acetylene welding in comparison with water-gas welding is that the former process permits to make the seam at will thicker and stornger than the iron sheet itself. Some care must be taken, however, in oxy-acetylene welding; the acetylene must be pure and must not contain dust or impurities; the dimensions of the burners must correspond to the thickness of the sheets; the additional steel to be put on to the weld must not be of inferior quality and should not be added until the edges to be welded together are thoroughly fused. Of main importance are the hammering of the weld and the annealing after the weld is completed. Finally, wherever it is possible, the two sheets should be welded together from both sides. If carefully carried out, the results obtained with oxy-acetylene welding are at least as good as with water-gas welding. That seams made by oxy-acetylene welding are very reliable was proven by hydraulic-pressure tests made with hollow receptacles. The published statement that only thin sheets (up to 8 mm with oxy-hydrogen and up to 10 mm with oxy-acetylene) can thus be welded is wrong, since the author has used oxy-acetylene welding with best success with sheets of 30 mm or more. There is no danger of oxidizing the material during the welding operation, if care is taken to use a reducing flame (an excess of acetylene or hydrogen), but this is quite easy.

Precious Metals.

Precipitation by Zinc.—In an interesting paper read before the Chemical, Metallurgical and Mining Society of South Africa (*Journal* of the Society, March, 1909), Mather Smith claims that the generally accepted principle is wrong that it is necessary to have 1 cu. ft. of zinc shavings for the precipitation of one ton of solution in 24 hours; he further claims that more zinc than is necessary is being used, that an excess of zinc is both harmful and costly, that precipitation boxes are often constructed on a wrong principle, and that a comparison of the rate of the flow of solution per cubic foot of zinc, which does not also include or consider the difference in area and number of compartments in the precipitation boxes, is misleading. He gives examples from his practice to prove that the superfluous zinc does a great deal of harm, "as it must occasion an unnecessary consumption of cyanide and charge the solutions with an excess of zinc, which in circulating through the plant will in a great measure be deposited as white precipitate at the heads of the boxes." The principle on which he recommends to construct precipitation boxes is to allow one square foot in area for the precipitation of one ton of solution in 24 hours, with a maximum height of 24 inches from the top of the zinc frames to the top of the solution, and with a maximum of five compartments in each box. In the discussion of the paper (April issue of the *Journal* of the Society), J. H. Johnson says the problem resolves itself more into a matter of time the solution is in contact with the zinc than the depth or amount of Zn required to a ton of solution. In the old-style boxes and in the boxes recommended in Smith's paper the time for which the solutions are in contact with the zinc was from 20 to 30 minutes or more, while in many cases now the solution is only in contact for a period of 6 to 8 minutes. Depth and amount of zinc shavings per ton of solution have to be increased when the time of contact decreases.

Zinc Dust Versus Zinc Shavings.—An interesting comparison of precipitation of gold from cyanide solutions by zinc shavings or by zinc dust was given by A. J. Clark in a paper before the Chemical, Metallurgical and Mining Society of South Africa (*Journal* of the Society, Jan., 1909). The comparative results were obtained in the No. 1 sand plant of the Homestake Mining Company on a working scale with solutions practically identical in composition, during 12 months of 1906 with zinc shavings and during 9 months of 1908 with zinc dust. The average strength of solution precipitated by zinc shavings was 0.03 per cent; if below this figure the solution was strengthened by addition of barren solution of 0.07 per cent strength. The average strength of solution precipitated by zinc dust was 0.0242 per cent; the minimum strength encountered and success-

fully precipitated by dust was 0.015 per cent. The principal comparative figures are as follows:

	Shavings	Dust.	Shavings by dust.
Tons precipitated	5909	5585	
Dollars received total	$1513	$1513	
Per cent precipitated	91 +	94.4	
All costs, labor and supply	$243.28	$76.93	$166.35
Gold in sands in barren solutions	$126.45	$99.41	$27.04
Value of cyanide in wasted barren solution	$709.00	$540.60	$168.40
Totals	$1,078.73	$716.94	$361.79

Telluride Gold Ore.—The metallurgy of the telluride gold ores of the Cripple Creek district in Colorado has been the subject of a report (edited by Thos. B. Crowe) of the Portland Section of the Chemical, Metallurgical & Mining Society of South Africa. It was published in the May issue of the *Journal* of the society. Besides a large tonnage of ore of exceptional richness, the Cripple Creek district yields ore which, on account of its refractory nature, is left standing in the mine stope or thrown upon the waste dump. The refractory nature of these ores is due largely to the occurrence of the gold in combination with tellurium, forming a compound ($AuAg)Te_2$. After a trial of other methods, which turned out failures, roasting followed by barrel chlorination and concentration has become the popular treatment. This method works very well as far as the extraction is concerned, and will continue to be used so long as the present grade of ore is maintained; but as the amount of low-grade ore standing in the stopes and upon the dumps becomes larger year after year, it becomes more apparent that a cheaper method of treatment must be found, the problem resolving itself into one of raw treatment at the mine, rather than transporting 50 miles by rail to the cheap fuel in the valley. With this end in view the Portland Gold Mining Company has carried on extensive experimentation.

An interesting feature of these experiments was the use of potassium persulphate, the object being to dissolve the tellurium. When potassium persulphate is alone in solution, it acts as a strong oxidizing agent. But when mixed with cyanide solutions, its oxidizing influence is greatly retarded. It then acts as a "slow oxidizer or depolarizer, thereby greatly increasing the dissolving power of cyanide solutions and when used in small proportions, 1 lb. to 10 lb. per ton of solution, it does not destroy the cyanide to any great extent." Potassium persulphate, when mixed with cyanide solution, completely dissolves the telluride of gold. Three processes, based on this principle, were tried.

(1) A preliminary treatment by an alkaline persulphate, followed by a treatment with a cyanide solution. The results were poor, since extreme amounts of persulphate must be used to obtain a good tailing.

2. A treatment by a solution containing both alkaline persulphate and potassium cyanide. The process was not found to be economical.

3. A treatment by a solution of an alkaline persulphate, an alkaline iodide, and potassium cyanide, contained in one solution. The object was to obtain the reaction $KCy + 2I = ICy + KI$ and thus produce cyanogen iodide (which is an exceedingly good solvent for the telluride of gold) and not to depend directly upon any of the ingredients other than cyanide and cyanogen iodide as solvents. In using this process, good tailings were invariably obtained, even with very weak solutions. But it was found that the solutions after leaving the ore and standing in contact with the air lost their cyanide, especially when heavy sulphide ores were treated. However, in passing through the zinc boxes, the deterioration of the cyanide is stopped, due undoubtedly to the reducing action encountered there. Alkaline persulphate may be produced economically by electrolysis of salt cake. "Potassium iodide, although an expensive chemical, would remain as such in the solution, the persulphate causing it to give up its iodide, this iodide combining with the cyanide to form cyanogen iodide, with the alkali to form an alkaline iodide or alkaline iodate, but always finally reverting back to an alkaline iodide, as any iodate formed would be reduced to iodide in the zinc box."

The report is discussed in the July issue of the *Journal* by D. J. Moir and C. H. Cropper and in the September issue by T. B. Stevens. Moir criticises some of the chemical statements in the report, while C. H. Cropper gives some notes on successful practice with telluride ores at Kalgoorlie, W. A. There is a wide difference in the solubility of the gold in telluride minerals. The ore being treated at Kalgoorlie is for the most part a sulphotelluride. In the early days of the field, when the grade of ore was high and the extractions low, it was discovered that tellurium was a factor to be reckoned with in future treatment. Bromo-cyanide enabled better extractions to be obtained, although the residues contained high values. At the present day the average grade of ore has fallen considerably; the highest may be taken at 55s, the lowest at 28s. In the constant effort to reduce the value of the residues the economic factor has prevailed, inasmuch as an extraction of 85 per cent on a refractory ore may show a better cost sheet than a 95 per cent. As to which of the two processes in vogue is the better, viz., dry crushing or wet crushing, is a matter of opinion, but the facts are that with the dry crushing and all roasting plants the residues are under 1 dwt. without the use of BrCy, while the wet crushing residues are 1½ dwt., the raw slime only being treated with BrCy. Total treatment costs, as they are to-day, range from 8s. 8d. per ton to 10s. 8d. for wet crushing, from 10s. up to 11s. 8d. per ton for dry crushing. The capital outlay on a dry crushing plant is greater than for wet crushing, but the difference would be compensated for in the lower residues obtained; greater improvements in roasting furnace design have advanced dry crushing practice. With the wet crushing plant, concentration of the mineral is employed, but there is a proportion of very fine mineral which is carried away in the slime and which is the cause of the higher values in residues.

T. B. Stevens asks whether the authors have made any comparative experiments, using bromo-cyanide instead of potassium cyanide and persulphate, and if so, whether the latter method gives a very much better extraction. The cost of the latter would be higher. The cost of chemicals for bromo-cyanide treatment at Kalgoorlie has always been one great disadvantage of an otherwise successful process. The writer's experiments with Cripple Creek ores showed that a good extraction could be obtained by bromo-cyanide treatment, fine grinding being a sine quâ non. At Kalgoorlie it is found most economical to take out the pyritic portion of the ore by concentration, and treat it separately by roasting and cyaniding; otherwise, in treating this very refractory material together with the bulk of the ore, the cost for bromine salts, to get a good extraction, is found to be excessive. With Cripple Creek ores the writer found this also to be the case, and, in the latter district, he believes that the small amount of high-grade concentrates could be more profitably shipped to the smelters instead of treating it at the mine.

Continuous Collection of Sand for Cyaniding.—A paper on an interesting new development in gold metallurgy on the Rand was presented by W. A. Caldecott before the Chemical, Metallurgical and Mining Society of South Africa and is printed in the August issue, 1909, of the *Journal* of this society. The new process is the outcome of long extended experiments, the object of which was to effect some saving in the very high capital expenditure for a modern cyanide plant. On consideration it appeared that the most likely prospect of success lay in eliminating the sand collecting vats, or rather in utilizing them for treatment purposes instead of merely as sand storages, which take the place of the old tailing dams. The upper vats of the superimposed type of sand plant were on their first introduction before the Boer war utilized as treatment vats as well as sand collectors, but the results of contamination of mill water with cyanide caused this practice to fall into disuse, and

in any case a considerable time elapsed between the filling of a vat and its charge yielding gold-bearing solution to the boxes. Hence, although it reduced cost of transfer to a minimum, this type of plant fell into disuse as being expensive, through having only one-half its capacity available for extraction purposes, and was replaced by separate blocks of collectors and leachers (with belt or truck transfer) in the ratio of 1 to 2 or 2 to 5, which is now the common practice on the Rand. On considering how to utilize sand-collecting vats for treatment purposes and to retain their advantages, including minimum ground space required, while avoiding the difficulties mentioned, the problem became essentially that of continuously removing the water and slime from the tailing pulp so as to leave the same in proper condition for cyanide treament. The solution was found in the use of conical classifiers in connection with a specially designed slowly rotating horizontal vacuum filter. The first filter of this kind, constructed by the author, was 10 ft. in external diameter and consisted of an annular launder 12 in. wide containing a filter cloth upon a grating as a false bottom. The filtering area was thus 28.3 sq. ft. The space under the filter cloth was connected by means of radial pipes to a central hollow spindle, in which a vacuum was maintained from a receiver.

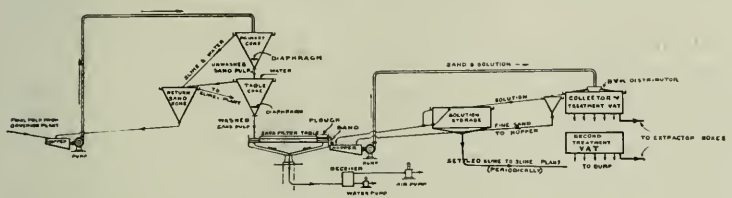


DIAGRAM OF CONTINUOUS SAND FILTRATION PROCESS.

To the lower portion of the receiver was connected a water pump and to the upper a vacuum air pump. Various other filters of the same type, but of larger capacity, have since been installed and have proven thoroughly successful in large-scale operation. Considerable details are given on the arrangement of the plant and comparative working costs. The accompanying illustration shows what is in Mr. Caldecott's opinion at present the most suitable arrangement of plant in elevation, with superimposed primary and table cones, as well as superimposed collecting and second treatment vats. The superimposed vat system under these conditions is no longer open to the objection of contaminating mill service water, while all the vats, both upper and lower, are utilized for treatment, and the cost of installing and operating transfer belts or trucks is done away with. The collecting vats used to receive the almost slime-free sand solution pulp may be of larger dimensions than usual, as settlement of slime in layers is not to be apprehended, and fewer larger units thus employed in place of more smaller ones. For the same reason the bulk of the lime used may be crushed with the ore in the battery, thus reducing the separate grinding of lime in a ball-mill or other machine to occasional periods when unusually acid ore is being delivered to the battery. Naturally the use of sand filters greatly increases the capacity of existing plants whether of superimposed or other type, by converting all the collectors into treatment vats. "The advantages which sand filters present are considerable saving in capital expenditure both for sand vats and for belt or truck transfer installation, slightly better extraction than the ordinary system unless prolonged treatment is given, a saving with superimposed vats of belt or truck transfer cost, and more slime to be treated as such at, say, 6d. per ton less than sand in vats of slightly increased diameter."

Copper.

Calculation of Blast Furnace Charges.—In the *Mining and Scientific Press* of Nov. 13, PERCY E. BARBOUR emphasizes the advantages of calculating blast-furnace charges. He describes two methods; in the first, the slag desired is assumed and the charge is calculated; in the second, the charge is assumed and the slag is calculated. The first example is based on the practice of the U. S. Smelter in Salt Lake Valley, which treats its own ores (dry and sulphide ores) together with purchased custom ores. The furnaces are 42 in. by 180 in. at the tuyeres. The conditions at the smelter require on the charge 300 lb. of ore "A," 1200 lb. of ore "B," 500 lb. of ore "C" and 300 lb. of converter slag, "D." (The assays for silica, iron, lime, sulphur and copper are given in the article.) In choosing a slag to be made, it is not the practice at this plant to make an exact singulo-silicate ($2RO \cdot SiO_2$) or other exact class slag, but to make that slag which experience has demonstrated is best suited to local conditions. This experience-dictated slag, however, approaches a singulo-silicate. Lang, in his "Matte-Smelting," considers the best slag to be generally a bisilicate of lime ($CaO \cdot SiO_2$) and a singulo-silicate of iron ($2FeO \cdot SiO_2$) in equal parts. The assumed slag in the present case is 44 SiO_2: 25 FeO = Fe 19.5; CaO 0.23. The problem is to determine the quantity of dry (silicious) ore containing high gold and silver content, and 2 per cent copper, which can be put on the charge, and the amount of lime necessary to be added. The assays of these two materials are known, and are entered on the charge-sheet. Let the required amount of silicious ore be represented by x and the amount of lime by y. The total silica on the charge is that in ores A, B, C, D (which in this case is 765 lb.), plus 69 per cent of the weight of the silicious ore, x, plus 3 per cent of the weight of the lime-rock, y; 69 and 3 being respectively the silica content of the silicious ore and the lime-rock. Therefore, the total SiO_2 on the charge is SiO_2 in pounds $= 765 + 0.69\,x + 0.03\,y$.

In like manner the total copper on the charge is the copper in the ores A, B, C, D (in this case 42 lb.), plus 2 per cent of the weight of the silicious ores. Therefore, the total copper in the charge is Cu in pounds $= 42 + 0.02\,x$. Further, in this special case it is calculated that 393 lb. of sulphur are in the ores of the charge, and if it is assumed that 85 per cent will be volatilized, there will be left 15 per cent for the matte, which amounts to 59 lb. In forming matte, Cu combines with one-quarter of its own weight of sulphur to form Cu_2S, because

At. wt. $Cu = 63.6$ $Cu_2 = 127.2$

At. wt. $S = 32.07$. $127.2 \div 32.07 = 4$.

Since the amount of Cu in the charge is $42 + 0.02\,x$, the amount of sulphur required by this amount of copper is $10 + 0.005\,x$, and the amount of sulphur left to combine with iron is $49 - 0.005\,x$. In forming the matte the sulphur combines with 1.75 times its own weight of iron to form FeS, because

At. wt. $Fe = 55.9$

At. wt. $S = 32.07$. $55.9 \div 32.07 = 1.75$.

The amount of Fe required to satisfy $(49 - 0.005\,x)$ lb. of S is, therefore, $86 - 0.009\,x$. The total Fe in the charge as per calculations, is 599 lb. from ores A, B, C, and D, plus 4 per cent of the weight of the silicious ore x, or $599 + 0.04\,x$. Therefore, the total Fe left to go into the slag will equal $(599 + 0.04\,x) - (86 - 0.009\,x) = 513 + 0.049\,x$.

The total lime (CaO) in the charge, all of which goes into the slag is 69 lb. from A, B, C and D, plus 6 per cent of the weight of the silicious ore x, plus 51 per cent of the weight of the lime-rock y. This equals $69 + 0.06\,x + 0.51\,y$. We have, therefore, three equations for the pounds of silica and iron and lime in the slag:

$$SiO_2 = 765 + 0.69\,x + 0.03\,y$$
$$Fe = 513 + 0.049\,x$$
$$CaO = 69 + 0.06\,x + 0.51\,y$$

In the chosen slag the ratio of SiO_2 to Fe is $44 \div 19.5$, while

the ratio of SiO_2 to CaO is 44 : 23. This gives the two equations

$$11.25\,x - 0.58\,y = 7700$$
$$13.20\,x - 21.71\,y = -14.570$$

From this we find $x = 1129$ and $y = 1056$, the pounds of siliceous ore and limerock respectively to be added to the charge.

Hence, the charge will consist of 2500 lb. ores A, B, C, D, 1129 lb. siliceous ore and 1056 lb. limerock, hence total weight 9685 lb. From the above equations we find that this charge will contain 1231 lb. SiO_2, 624 lb. Fe, 646 CaO, 55 Cu. The 55 lb. Cu in the charge will require 55 ÷ 4 = 14 lb. S for the Cu_2S in the matte. The remaining sulphur 50 − 14 = 45 lb. will require 45 × 1.75 = 079 lb. Fe for the FeS of the matte. Therefore, the matte will contain 55 lb. Cu, 50 S and 79 Fe. Assuming that these elements will comprise 95 per cent of the matte, the other 5 per cent being substances not considered, there results 2.3 lb. of matte, equivalent to a matte fall of 203 ÷ 3985 = 5.1 per cent. The resulting matte will assay 55 ÷ 2.3 = 27 per cent Cu. The slag will contain all the SiO_2 in the charge, 1231 lb., all the CaO in the charge, 646 lb.; the Fe in the charge, 624 lb., taken up by the S in the matte = 545 lb., which is equivalent to 699 lb. FeO. The slag is then composed of 1231 lb. SiO_2 = 43.9 per cent, 699 lb. FeO = 24.9 per cent, 646 lb. CaO = 23.1 per cent, which is the slag intended to be produced.

The method of calculation formerly used at the Garfield smelter is finally described. It consists in assuming the charge from past experience and the amounts which must be smelted, and calculating the resulting slag. The charge consists of 2000 lb. of bed-ore, 2500 lb. of ore B, 500 lb. siliceous ore, 500 lb. slag, and 500 lb. limerock. The amounts of silica, iron, lime, sulphur, and copper are calculated and carried out on the sheet. The volatilization of sulphur is assumed at 60 per cent. There is then left for matte 167.4 lb. sulphur. Copper in the matte requires one-quarter of its weight in sulphur, or 62 lb. Thus, 167.4 − 62 = 105.4 lb. S is left to combine with the iron of the matte. Sulphur requires 134 or 7/4 its weight of iron to form FeS. Therefore, 105.4 × 7/4 = 164.1 lb. Fe required. This amount leaves 995 lb. to go into slag. Since Fe is 7/9 of FeO, 995 × 9/7 = 1278.9 lb. of FeO in the slag.

The slag has now been determined. It contains 1786.0 lb. SiO_2, 1278.9 lb. FeO, and 699.0 lb. CaO. Assuming that these form 92 per cent of the total slag, and that the other 8 per cent is composed of Al_2O_3, ZnO, etc., 4319 lb. of slag result, whence the above quantities give 41.1 per cent SiO_2, 29.6 FeO, 21.0 CaO. The matte will contain 2.48 lb. Cu, 164.1 Fe, 167.4 S. Assuming that these constitute 95 per cent of the total, there is a matte fall of 610 lb., equal to 10 per cent. The copper in the matte is 40.6 per cent.

The above article is critically reviewed by Prof. L. S. Austin in the *Mining and Scientific Press* of Nov. 27. He remarks that in copper smelting, quite differently from silver lead smelting, a great variation in slag composition is permissible, and hence has risen a carelessness in the preparation of the charge. But this is no reason why not to compute a charge, however, when this has been done, the slag which is produced should be controlled by the eye as well as by analysis. Referring to the single silicate slag, which Mr Barbour proposes to use, Professor Austin says that in pyrite practice the metallurgist is chiefly anxious that his slag should carry specified quantities of silica (in this instance 44 per cent), and that the bases go as far as possible with the addition of no more dead limestone flux than needed. The total of SiO_2, FeO and CaO adds up in the present case to 92 per cent, while in practice the sum varies between 85 and 90 for Utah ores.

Various objections are raised by Professor Austin against details of the calculation of the Garfield slag. He states that he has "given up the work of the Metallurgy of the Common Metals (to charge calculations for iron, silver lead, and copper charges) based upon the trial method, which he has found practically the most used by smelter men. Since, in the final summing up, this computation is used to check the results, why not use it from the beginning?"

Zinc.

Effect of Impurities on Roasting and Distillation Process.—The various reactions which are caused by impurities in the roasting of zinc sulphide ore and in the reduction and distillation of zinc oxide are the object of a paper by V. Lepiarczyk in *Metallurgie* of July 8, 1909. The experiments were made on a rather small scale, the charges were made up not from ore, but from pure materials, and the influence of different additions was investigated in succession. In roasting pure zinc sulphide it is possible to drive off all the sulphur, if the conditions are such that any zinc sulphate which is formed during the roasting process is destroyed and SO_3 and SO_2 are carried off. If in the roasting process in practice not all the sulphur is driven off, this is due to the presence of iron (Fe_2O_3 or FeS) and limestone ($CaCO_3$). The iron is less detrimental. The presence of Fe_2O_3 manifests itself in two directions: firstly, it causes some of the ZnS to remain unchanged; secondly, there is a reaction between ZnS and Fe_2O_3 yielding FeS. The presence of lime is far more detrimental. While it does not prevent the complete change of the zinc sulphide into the oxide, yet the largest part of the $CaCO_3$ is changed into $CaSO_4$. This $CaSO_4$ not only represents a loss of sulphur, but it causes an agglomeration of the roasted blende when it is being stored so that it has to be crushed again; further, the $CaSO_4$ is reduced in the retort in the distillation process as will be shown later on. If iron is present in form of FeS together with $CaSO_4$, it prevents a small part of the zinc sulphide from being completely changed into oxide; the sulphur from the FeS itself is completely removed, but the $CaSO_4$ is not completely changed into sulphate. The results of this part of his research are summed up by the author in the statement that it is the Fe_2O_3 which prevents the complete oxidation of the ZnS, while the lime causes the formation of sulphate.

In the reduction process in the retort the reactions between the different materials which may be present were studied separately. If nothing but carbon and $CaSO_4$ are present, not the whole $CaSO_4$ is reduced to CaS, as some remains unchanged; but there are two reactions going on at the same time, namely, $CaSO_4 + 4\,C = CaS + 4\,CO$ and $3\,CaSO_4 + CaS = 4\,CaO + 4\,SO_2$. In the second experiment nothing but carbon and zinc sulphide were present, while it is generally supposed that zinc sulphide is not reduced, this was found to be erroneous; some of the zinc and sulphur evaporates and the quantities of Zn and S evaporated are in the ratio of their atomic weights. In the third experiment the reaction between carbon and FeS was studied and it was found that FeS is even more attacked than ZnS. It seems that the two reactions which take place in the second and third experiment are $2\,ZnS + C = 2\,Zn + CS_2$ and $2\,FeS + C = 2\,Fe + CS_2$. This is interesting for practical purposes since it shows that the FeS is really not so dangerous as is generally supposed because only little FeS remains back in the roasting process, and the greatest part of this is decomposed in the retort. In the fourth experiment the reaction between Fe_2O_3 and ZnS was investigated, and it seems that the complete progress of the reactions is given by the formulas $2\,Fe_2O_3 + 7\,ZnS = 4\,FeO + SO_2 + Zn + 3\,ZnS = 3\,FeO + Fe + ZnO + SO_2 + 3\,ZnS = 3\,FeS + Fe + 4\,ZnO + SO_2$. The author then studied the reactions between more than two materials. If carbon, zinc sulphide, and Fe_2O_3 are present together the reaction is $Fe_2O_3 + 2\,ZnS + 4\,C = 2\,Fe + 2\,Zn + 3\,CO + CS_2$. If carbon, zinc sulphide, and FeS are present the reaction seems to be $2\,ZnS + 2\,FeS + C = ZnS + Zn + FeS + Fe + CS_2$. While pure $CaSO_4$, when reacting with C, yields CaS and CaO and some non-decomposed $CaSO_4$, yet when $CaSO_4$ is present together with ZnO, FeO, and MnO, there is formed only CaO besides some non-decomposed $CaSO_4$. The author concludes that the following reactions take place: (a) $CaS + ZnO = CaO + ZnS$, or (b) $CaSO_4 + ZnO = ZnSO_4 + CaO$ and, for

ther, $ZnSO_4 + 4C = ZnS + 4CO$, (c) $CaS + FeO = CaO + FeS$, or (d) $CaSO_4 + FeO = FeSO_4 + CaO$, and, further, $FeSO_4 + 4C = FeS + 4CO$, (e) while in presence of Mn the following additional reaction occurs, $FeS + Mn = MnS + Fe$.

The chief conclusion of the author which is of some interest for the practice is that the iron content in zinc sulphide is slightly detrimental in the roasting process because it causes a small part of the ZnS to remain unchanged. In the reduction and distillation process, however, the iron is very useful if it is present in form of Fe_2O_3 for two reasons: First, it changes any ZnS which has not been oxidized during the roasting process into oxide and thus makes it available before reduction and distillation. Secondly, when the $CaSO_4$ is reduced in the retort, the sulphur thereby liberated is caught at once by the iron oxide so that it cannot combine with zinc. If roasted blende rich in lime is to be reduced it seems useful to add some iron oxide in order to increase the yield of metallic zinc. But this would be unnecessary if it were possible to remove the lime from the ore before the roasting process, and this would be most advantageous, both for roasting and distillation. It should be tried to remove the lime by washing the ore in one process. If no calcium sulphate was present the content of Fe_2O_3 in the ore alone would be sufficient to change any remaining zinc sulphide to zinc oxide and thus permit its reduction and distillation.

Chemical Engineering.

Multiple-Effect Distillation.—At the recent Philadelphia meeting of the American Institute of Chemical Engineers two papers were presented on multiple-effect distillation. In a paper by F. J. WOOD (mechanical engineer of Marx & Rawolle's glycerine refinery), the various steps leading up to the present development of the art of glycerine distillation are described with particular reference to the fuel economy. The consumption of coal per pound of glycerine output is greater now than with the earlier system, as it has been found more economical to save glycerine from decomposition by the use of a flow of steam in a vacuum which greatly lowers the boiling point. In the multiple-effect apparatus, a superior economy of fuel is obtained by using but one flow of steam to a number of stills and condensers, the stills being arranged in series in regard to the steam flow—that is, the steam from the steam jet of the first still flows through a series of stills alternating with condensers, the condensers being maintained at a temperature that will condense glycerine out of the steam, but not the steam itself. The glycerine is forced through heaters by means of centrifugal pumps. The heaters are connected with the boilers and supply the heat necessary for the vaporization of the glycerine. From the heaters the hot glycerine is sprayed down through the steam.

The combined vapors of steam and glycerine pass into boiling water condensers whose temperature is maintained at such a point by means of a pressure regulator, that the glycerine, but not the steam, is condensed. The glycerine in condensing gives up its latent heat of evaporation to the boiling water. The steam passes on to the next still in the series to take up its quota of glycerine. Finally, after the last of the series of stills is passed, the steam, together with the volatile impurities, are condensed as sweet water in a cold water condenser. In order to maintain a uniform vacuum throughout the system, there must be a free flow of steam and glycerine vapor without obstruction. In this system, instead of forcing the steam down through the body of glycerine, the glycerine is pumped up and sprayed down through the steam and no back pressure to the flow of steam results.

The paper by A. C. LANGMUIR, also presented before the American Institute of Chemical Engineers, deals with the general advantages and applications of multiple-effect distillation. In the author's opinion the wood process of glycerine distillation marks one of the most important advances made in recent years in this line. The patent rights have been acquired by Marx & Rawolle, New York, in whose plant the process is now in successful use on a large scale. (See abstract above.) The fuel economy is marked, a considerable increase in output per still being obtained at a fuel consumption of less than half of that formerly used. A set of six stills in series is successfully at work with no diminution of efficiency in the last still as compared with the first.

The chief advantages of the process, aside from the fuel economy, consist in the ease of working, uniformity of product with varying crudes and the scientific control possible at all times. The process is largely automatic, less labor and less skill in handling being required. Only a small body of glycerine is present in the heaters, stills and connections, and the prolonged heating of large bodies of glycerine with resulting loss in the average still is avoided. The spraying of the glycerine through the steam avoids the shooting and foaming often met with when distilling certain crudes in the still generally used. The process is applicable in any distillation where steam is used and the product obtained has a boiling point higher than that of water, such as the distillation of the heavier products of coal tar and petroleum, of aniline, essential oils and the fatty acids of the candle industry.

Smokeless Combustion and Boiler Efficiency.—At the recent Philadelphia meeting of the American Institute of Chemical Engineers a paper was presented by A. BEMENT, Chicago, Ill., giving the results of tests carried out to ascertain the loss of heat due to incomplete combustion. The results are given in a table showing the comparative heat balance of an ordinary water-tube boiler in which the combustion was incomplete and an improved boiler arranged for the reception of a fire-brick tile furnace roof which made it necessary for the gases to travel to the rear of the boiler before coming in contact with the water tubes. By this arrangement, complete combustion was secured, as shown by the absence of smoke. The loss of heat due to incomplete combustion in the ordinary boiler was found to be 9.3 per cent, while in the improved boiler there was no loss. The method of making the determinations and calculations was given in full. In the second part of the paper, the causes governing the efficiency of a boiler were discussed, on the basis of the results of the efficiency of three boilers. The first was a well-known water-tube boiler in which the gases entered among the tubes at the rear and passed diagonally across the tubes to the exit. In the second boiler, a baffle was inserted among the tubes so that gases flowed twice the length of the tubes. In the third boiler, two baffles were inserted so that the gases flowed three times the length of the tubes. The fuel saving in the single-baffled over the plain boiler was 8.68 per cent, while the saving in the double-baffled was 20.68 per cent. Complete data with reference to the coal, temperature of gases, steam, draft, power development, etc., were given.

Oil and Grease Extraction.—In a paper read at the Philadelphia meeting of the American Institute of Chemical Engineers, Mr. W. M. BOOTH discussed the properties of the solvents which can be used for the extraction of oils, giving the solvent power and the cost of each. On this basis the cheapest solvent is found to be naphtha. The construction of building tanks and extraction apparatus was discussed. Methods of operation were given, also an account of the methods in use for the recovery of grease from garbage tankage and other waste products is given; drawings of extraction apparatus, and a number of tables and curves giving the properties of various fats and oils.

Utilization of Waste India Rubber.—In a paper read at the Philadelphia meeting of the American Institute of Chemical Engineers, Mr. S. P. SHARPLES gave a résumé of the various methods which have been employed in recovering the waste rubber of commerce, calling particular attention to the processes of the late Nathaniel Chapman Mitchell in the Philadelphia Rubber Works. Mr. Mitchell was the first to make a commercial success in this country of methods for utilizing worn-out rubber articles, such as boots and shoes. This was largely due to the fact that he discovered that sulphuric acid of 15° B.

to 25 lb. could be employed to remove the fiber from these goods without injuring the rubber. The process was at once stolen and utilized all over the country until at the present time it is used in many works.

Glass and Iron.—Although the formation of a special Chemical Reichsanstalt in Germany is contemplated some chemical research work is carried out at present at the Physikalisch-Technische Reichsanstalt in Berlin. From the report of the work done in 1908 given in London *Engineering* of Dec. 17, we notice briefly some of the more interesting chemical investigations. They concern first the weathering of crown glass by means of the cosin reaction, the glasses must not be dipped into the solution for much more than a minute, lest the lead be attacked in addition to the alkali. The firm of C. A. F. Kahlbaum has purified large bulks of iron nitrate, $Fe(NO_3)_3 + 9H_2O$, which is melted at 50 deg. Cent. and exposed to gaseous ammonia; dense red iron oxide and ammonium nitrate result. The latter is washed out and the oxide reduced and the powdery metal melted in the oxy-hydrogen flame. An iron rod thus prepared contained 0.001 per cent of manganese, 0.004 per cent of copper, the same amount of nickel or cobalt, traces of carbon and silicon, and a few hundredths per cent of oxygen. Reduced oxalate, also from Kahlbaum, was less pure. The physical tests of this "nitrate" iron and of other iron gave the following results:

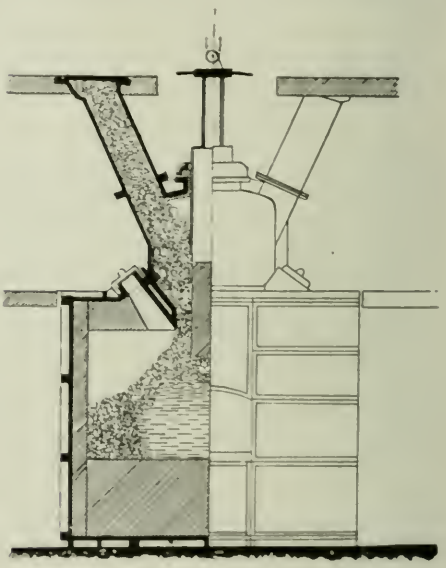
	Nitrate iron.	Kreusler wire.	Iron 0.1 per cent of carbon.	Steel 1 per cent of carbon.
Electric conductivity at 18 deg. Cent.	10	9.6	8.3	5×10^{-4}
Temperature coefficient of this conductivity	5.9	5.5	4.5	3.7×10^{-3}
Magnetic coercitive force	1.4	2.6		

This nitrate iron, which is superior to most electrolytic iron, will be valued by chemists for preparing their titration solutions and by physicists, but it is not a "pure" iron yet, of course.

Electrochemistry.

Large Electric Furnaces.—The tendency in electric furnace construction is toward a rapid increase of the size. This is clearly shown in the present development of the electric steel furnace and it is well known that the chief difficulty encountered in increasing the size is found in the construction of the electrodes. In the *Zeit. f. Electrochemie* of Aug. 1 R. Taussig describes the design of A. Helfenstein for large furnaces for calcium carbide manufacture. The size is limited by the amount of power which a single electrode or a compact electrode bundle can carry into the furnace. According to the results obtained in actual practice the maximum so far obtained is 2500 or 3000 kw. per electrode (30,000 to 40,000 amp. at 75 to 90 volts). For a three phase furnace with three electrodes this means a total capacity of 7500 to 9000 kw. or 10,000 to 12,000 hp. A further step has been made in the design of a double three phase furnace in which six instead of three electrode bundles are placed in the same furnace shaft. This means two separate three-phase units in the same shaft, so that the capacity of such a furnace is 15,000 to 18,000 kw. corresponding to a production of 80 to 110 tons of carbide in 24 hours in one furnace. With such large furnaces the heating effect of the reaction gases of the furnace becomes very great and must be taken care of in order to protect the workingmen and the connections to the external current. Further the smoke becomes troublesome (with ferro-silicon even more than with carbide). The greatest difficulty is, however, found in the method of charging the furnaces so as to arrange the charge properly around the electrode. It seems advisable with large furnaces to build them air-tight so that the smoke trouble is overcome and the heat of the gases of the reaction can be usefully employed. The adjoining illustration shows a vertical section of a single electric hearth for 8000 to 10,000 hp. About 8 or 10 meters above the bottom of the furnace a special charging floor is arranged. At the top of the furnace there is placed a large and tightly closed mixing receptacle or hopper for the charge, through the center of which the electrode passes down. Its weight is 3000 to 14,000 kg. for 10,000 hp. During the operation the electrode is surrounded on all sides by the charge. The charging receptacle has a capacity of 5000 to 7000 kilograms and is connected by means of long pipes with the charging floor. The charge is continually supplied through these pipes into the charging receptacle and passes from there into the electric furnace. In order to control the exact supply of the charge into the furnace, but especially for the removal of any furnace troubles, there are provided in the furnace at the side of the hopper broad slits which can be tightly closed. For a large three-phase furnace of 25,000 hp. three separate electrode bundles are used. Each electrode corresponding to one phase is provided with its own charging arrangement. It does not seem, however, that this design of large furnaces has already been tried out in practice.

Silundum.—At the last meeting of the German Bunsen Society Dr. R. Varberg read a paper on the manufacture of silundum in a carborundum factory in Bodeo, Switzerland. The



LARGE CALCIUM CARBIDE FURNACE.

paper is printed in *Zeit. f. Electrochemie*, October 1. They prepare amorphous carbon articles by pressing or by other treatment so that they get the desired form and then heat them in electric resistance furnaces of the simplest type so as to subject them to the action of carborundum and sand, or carbon and sand, or to both acting together. The two reactions are $SiO_2 + 2SiC = 3Si + 2CO$ and $SiO_2 + 2C = Si + 2CO$. The silicon permeates into the carbon article and forms silundum. (Chemically this is the same material as what is called siloxicon by Dr. Acheson.) As has been mentioned in our columns, silundum is useful as an electric conductor at high temperatures, for instance, for electric heating apparatus, etc. The resistivity of silundum is about six times that of carbon. The temperature coefficient of the resistivity is negative, but very small. At 1000 deg. Cent. silundum according to its constitution, has a resistivity of one half or two-thirds that at ordinary temperature. Silundum articles can stand temperatures of 1600 deg. for a long time without crystallizing and for short periods even temperatures up to 1700 deg. Above 1700 deg. silicon evaporates, yielding a white vapor of SiO_2, while the carbon remains back as graphite. Silundum can be heated to white heat and then immediately put into cold water without

cracking. It is also proposed to treat coke in the same way and to use the coke which has been changed into silundum instead of ferro-silicon in steel works. Silundum has been used as an electrode in electrolytic processes, but experiments on its use as a unit in sodium chloride electrolysis have not yet been concluded. Air attacks silundum even at the highest temperatures only in so far as a film of SiO_2 is formed. This protects, then, the internal parts.

Calcium Cyanamide.—It is well known that the formation of calcium cyanamide from calcium carbide and atmospheric nitrogen (which must first be separated from the oxygen) requires a temperature of at least 1000 to 1100 deg. Cent. in order to change all the carbide into cyanamide. It is true that the same reaction goes on at a lower temperature, like 800 or 900 deg. Cent., but it soon stops and it stops the earlier the lower the temperature. However, if a few per cents of calcium chloride are added to the carbide, a complete transformation of the carbide into the cyanide can be obtained at a lower temperature, like 700 or 800 deg. (process of Polzenius, German patent 163,320). Also, if some calcium fluoride is added to the carbide (process of F. Carlson) it is possible to obtain a product rich in nitrogen at 900 deg. Cent., while at 800 deg. Cent. the fluoride shows its favorable effect, but does not change all the carbide into cyanamide. A paper by F. FOERSTER and HANS JACOBY in *Zeit f. Electrochemie*, Nov. 1, is quite interesting as it shows that the action of the calcium fluoride addition is essentially different from that of the calcium chloride addition. If the quantities of calcium fluoride which are added are varied it makes practically no difference if only the heating process is carried on for a long-enough time. Whatever the amount of calcium fluoride added, the absorption of the nitrogen by the calcium carbide remains limited at 800 to 860 deg. The amount changed to cyanamide is the same whether 1, 2, 5 or 10 per cent of fluoride are added. The amount of the addition of fluoride determines only the speed of the transformation of the carbine into cyanamide. The formation of calcium cyanamide evolves considerable heat, but it is remarkable that the temperature attained by the charge containing calcium fluoride is less than that of charges without any additions or that of charges to which calcium chloride is added. In the latter cases the temperature may rapidly rise to 1000 deg. Cent. If, however, an addition of calcium fluoride is used, the temperature remains near 900 deg. Cent. The addition of calcium fluoride has, therefore, not only the advantage that it reduces the reaction temperature by 100 deg. Cent., but it permits a more quiet and more uniformed reaction. This is of importance when the retorts in which the process is carried out are heated from the outside. This has formerly been tried since in view of the relatively low temperature required electric heating was thought to be unnecessary. But as was described in our articles on the large new calcium cyanamide works in Odda, Norway, (our Vol. 7, pages 212, 309 and 360), it has been found preferable to use relatively small units and to heat them from the inside by electric resistors. Under these circumstances it is not necessary to use any addition in order to reduce the temperature of the reaction. But the present authors think that the addition of calcium fluoride might have the advantage of providing greater uniformity and regularity of the process. It is known that when nitrogen reacts with carbide (especially barium carbide) not only cyanamide but cyanide is formed. In this respect the addition of calcium chloride has again a different effect from the addition of calcium fluoride. With the fluoride some cyanide is obtained, with the chloride none.

Nickel Wire as Resistor for Laboratory Furnaces.—A paper by Prof. M. LE BLANC, read at the last meeting of the German Bunsen Society and printed in the *Zeit f. Electrochemie*, Sept. 15, gives details of an electric laboratory furnace in which nickel wire is used as a substitute for platinum. The results have been very satisfactory. In the discussion several speakers (among them Prof. Nernst) stated that they had obtained good results with nickel, but others complained of the great fragility of nickel. (The nickel-chrome alloy wire, made and used quite extensively in this country, would probably prove far more satisfactory for such work than pure nickel; it does not yet seem to have been introduced in Germany.)

Electrolysis of Alkali Chlorides.—A sketch of the development of the mercury cathode process in Europe in four successive types is given by R. TAUSSIG in *Zeit. f. Electrochemie*, Sept. 1. The original type—the old Castner-Kellner cell—was rocked. The cell had the great disadvantage that its units were too small, the current used for each cell being only 400 or 500 amp. The electrolytic plant with all accessories became therefore very large and somewhat complicated and the installation and attendance was expensive. The second type is the Kellner cell, which consists of three compartments, the chlorine compartment being in the center. The three compartments are connected together by siphons and pressure can be applied on the mercury in the two outer compartments by means of compressed air so as to force the mercury from one outer compartment into the middle compartment and then into the other outer compartment. When passing through the middle compartment the mercury alloys with the sodium which is being set free there. The anodes are platinum nettings, size 50 by 50 mm. They are arranged in electrode boxes of concrete which have proved more successful than the old clay boxes. Each box contains 88 platinum nettings so that each chlorine cell contains 528 platinum nettings of 525 grams weight, their capacity being 4000 amp. In the two outside compartments cathodes of cast-iron grids are placed. The third type is the Kellner-Solvay cell, which is now in use in Jemeuppe in Belgium, Brescia in Italy, and Lubimof in Russia. The mercury is circulated with the aid of a wheel. This cell represents the largest unit. It was originally built for 6000 amp, but is operated now at 10,000 amp and the current may be increased even to 15,000 amp. The fourth type is a modification of the latter cell and the only difference is that the mercury is circulated in it not by means of a wheel but by by means of an Archimedean screw.

Electrolytic Bleaching.—The electrolytic bleaching process as used in this country employs essentially electrolysis of sodium chloride for the production of caustic soda and chlorine, the latter being changed to bleaching powder; bleaching solutions are then made from the bleaching powder. On the other hand, in Europe the direct production of bleaching hypochlorite solutions directly in the cell in which the sodium chloride is electrolyzed has assumed considerable proportions in practice. On the basis of considerable experience it is claimed that the directly produced hypochlorite solution has a better bleaching effect on cotton and textile fabrics than solutions made from bleaching powder. (A hypochlorite cell of this type is made and used in this country for laundry work.) On the other hand, there has been a difference of opinion concerning the bleaching of sulphite pulp. In a paper by B. FRASS in the *Papier Fabrikant*, Vol. 7, No. 10 and 11 (also in *Zeit. f. Electrochemie*, June 15) he reports on an investigation from which he concludes that in sulphite cellulose bleaching, too, the directly produced hypochlorite solution is more effective than a bleaching powder solution. The behavior is as though the active chlorine in form of the hypochlorite solution was more effective than in form of a bleaching powder solution. An exact explanation has not yet been found.

Formation of Storage Battery Plates.—G. JUST, P. ARKENASY and B. MITROFANOFF gave in *Zeit. f. Electrochemie*, Nov. 15, an interesting account of extended researches on the use of an addition of nitric acid or nitrates to sulphuric acid electrolytes for the rapid formation of positive lead accumulator plates. The formation of plane lead plates was first studied in pure sulphuric acid according to the process of Plante; the development of capacity was investigated and charge and discharge curves were plotted. Analogous experiments were then made with an addition of nitrate to the sulphuric acid. They

showed that the capacity could be quickly increased in this way and also showed a change of form of the charge and discharge curves. The authors endeavored to find the best amount of nitrate to be added, but had some difficulty because the active mass had a tendency to drop off from the plane lead plates. Nevertheless, the experiments showed that approximately a concentration of 30 grams of potassium nitrate per 2.8 grams of sulphuric acid per liter was specially favorable. If too much nitrate is added the tendency of the active mass to drop off is increased. The less the addition of nitrate the more adherent becomes the active mass. They also found that when nitrate is added a white lead sulphate film is first formed on the positive plates. All attempts to improve the adhesion of the active mass to the plane lead surfaces were successful. But it was found that when grids such as used in practice with a large surface (Grosse-Berli, chemnitzer) were treated in the same way with the addition of nitrate, it was possible to get the formation of an active material which would be sufficiently adhesive in practice. The more nitrate is added, the better is the porosity of the active mass formed, while the durability is diminished. Since both effects act against each other in practice, there must be a certain concentration of nitrate which is most suitable.

Recent Metallurgical and Chemical Patents.

Iron and Steel.

Removal of Phosphorus.—Probably one of the most interesting advances made in recent years in steel refining is disclosed in a patent by Mr. Ernest Humbert (Dr. Héroult's metallurgist who is at present operating the Héroult electric furnace plant of the U. S. Steel Corporation in South Chicago). The invention consists of removing the phosphorus, not as usual, as phosphate, but as phosphide. The claims made for the process are that there is less loss of time and labor than in the old process, that there is an increase in the amount of steel produced, and that there is a saving in the electrical energy required for the electric furnace. The invention consists in the use of an oxidizing slag in such a way as to combine with the phosphorus and form a phosphate, and the adding of a reducing material to the slag containing the phosphate to convert the latter into a phosphide. The reducing material, usually ground coke, floats on top of the slag and reduces the phosphate without interfering with the metal below. "The strength of combination of the phosphide is so great that the phosphorus cannot be separated out of it by the iron without its first being changed back to phosphate, and this cannot occur in a reducing atmosphere." The fact that the reducing atmosphere is required seems to necessitate the use of an electric furnace. The following example of the operation of the process in practice is given. An impure charge of steel from a bessemer converter was used, analyzing approximately phosphorus 0.10 per cent, sulphur 0.16, manganese 0.10, silicon traces, carbon 0.07. This was transferred to a Héroult electric furnace. A black slag containing 400 lb. of mill scale and 400 lb. of lime was added to a 15 ton charge of steel, making an oxidizing slag, which became fluid when the steel below was thoroughly oxidized and all the phosphorus had passed into the slag as phosphate of lime ($P_2O_3(CaO)_3$). Upon this slag containing the phosphate of lime was added ground coke, which floated thereon and reduced the phosphate therein, forming calcium phosphide (P_2Ca_3). Thereupon, without removing the slag, the desired additions (silicon, manganese, etc.) were made to the molten steel. There was no return of phosphorus to the steel, and the final product contained phosphorus 0.015 per cent, sulphur 0.005, with carbon from 0.05 to 1.50 and silicon and manganese ad libitum. (943,192. Dec. 14, 1909.)

Electric Steel Furnace.—As was noticed in our June issue, 1909 (vol. 1, page 251), Mr. C. A. Keller has abandoned the type of electric steel furnace formerly employed by him, in which the current passes from one electrode above the bath into the bath and out of it again into another electrode above the bath (the well-known Heroult type). He now employs, like Mr. Girod, only one electrode at the top, or if he uses several electrodes they are all connected in parallel; the other terminal is the bottom or the hearth of the furnace. The current, therefore, passes from the electrode at the top into the bath and leaves the furnace at the bottom. For this type a durable construction of the furnace bottom is, of course, all important. In a recent patent of Mr. Keller a bundle of iron bars is placed vertically in the furnace bottom and are all connected electrically to a conducting plate arranged below their bottom ends. Between these bars and around the bundle which they form is strongly rammed in a refractory earth, preferably magnesia earth. The furnace bottom consists, therefore, half of vertical iron rods and half of magnesia earth. The latter also becomes a conductor at high temperatures. (941,419. Nov. 30, 1909.)

Electric Reduction Furnace for Iron Ore.—The Swedish electric reduction furnace for iron ore which has been described repeatedly in this journal (see especially the article by Yngstrom, page 11 of this issue) is patented by E. A. A. Grönwall, V. R. Lindblad and O. Stalhane. It will be sufficient here to quote the first claim, which relates to "the combination in an electric furnace with the heating chamber and the stack communicating therewith, the chamber being organized to provide a free space between the charge and the top of the heating chamber when the charge is introduced into the chamber and into the lower portion of the stack, of electrodes entering through the wall of the chamber and extended to pass through the said free space with their end portions immersed in the charge, and means for drawing gas from the upper part of the furnace and passing such gas into said free space portion of the furnace." (943,224. Dec. 14, 1909.)

Copper.

Converter Construction.—In customary practice of bessemerizing copper matte, a silica lining is used; it has the disadvantage that it reacts with the iron present to form a slag, so that the lining must be frequently renewed. If a non-corrodible basic lining is used, made of magnesite or chrome brick, etc., there results a considerable expansion under the influence of the high heat, and this expansion continues practically throughout the life of the lining, which should last for several months. The usual result is either disintegration of the lining or rupture of the converter shell. To overcome this difficulty, Mr. Wm. H. Peirce (Baltimore Copper Company) proposes to arrange the basic lining so that it is free to shift relatively to the shell so as to expand under the high heat developed in the course of treatment and at the same time preserve its continuity, while the tuyeres embedded in the lining have full delivery connection with the blast supply. In the first arrangement employing this scheme the tuyeres were flexibly sustained to shift with the lining to thereby avoid the breaking down of the tuyeres or the lining. Leaks through the lining, which would tend to disintegrate it, were thereby avoided. This arrangement has now been changed by Mr. Peirce in so far as the lining is shiftable as before relatively to the shell but the section of the lining in the immediate neighborhood of the tuyeres is anchored to the shell to form a zone in which there is no substantial shift. The cylindrical horizontal shell is open at its upper side to permit expansion of the lining, and is provided with a tuyeres, cylindrical, non-corrodible lining arched over at the open top of the shell and free to expand therein, and the set of blast tuyeres, extending

through the lining, is arranged longitudinally at one side of the shell. In such a converter the expansion of the lining is chiefly in a vertical direction or transerve to the axis of the shell. To prevent this shift in the immediate neighborhood of the twyers the section of the lining at one side of the converter and at the longitudinally extending twyer belt is anchored to the shell, the lining being otherwise free to expand in opposite direction from the twyer belt. The twyers are embedded in the lining and an abutment, or a series of abutments of metal, secured to the shell and extending into the lining, form a longitudinally extending shelf or barrier adjacent the set of twyers. (942,661, Dec. 7, 1909.)

Zinc.

Electric Zinc Smelting.—Mr. F. T. Snyder, who has already patented quite a number of different designs for electric zinc furnaces, has recently patented another one operated by three-phase currents. The furnace is of a narrow oblong shape and the three electrodes suspended from above are arranged in a row so that the side walls may be close to the source of heat. The hearth of the furnace is filled during operation with molten lead upon which rest the molten matte and slag. The furnace is open at the top to introduce the charge and permit the escape of the carbon monoxide gas which is burned at the top. The furnace is water-jacketed up to the top of the furnace to cool the upper portion of the charge. Powdered coke is introduced through a hopper surrounding the top of the electrode, while the main body of the charge is put into the furnace between the outer edges of this hopper and the water-cooled sides of the furnace. The zinc vapor rises in this outer part, but condenses on the water-jacketed walls "in liquid form rather than as dust." The condensed liquid zinc is withdrawn through drains consisting of thick hollow tubes of graphite, inserted in the walls and provided with a special water-jacket. The bores of these tubes are small so that they are completely filled by a small stream of liquid zinc. The zinc drops off at the outside into a well, which surrounds the lower part of the furnace and is at the bottom in connection with a molten lead receptacle in the hearth of the furnace. The advantage is that any of the heavier metals such as lead which may have been volatilized with zinc will settle in the well and join the main body of the lead in the bottom of the crucible. (942,110, Dec. 7, 1909.)

Deposition from Zinc Sulphate Solution.—In depositing zinc from sulphate solution by electrolysis, the electrolyte becomes gradually richer in free sulphuric acid and this results in part of the zinc deposit being redissolved from the cathode. Messrs. V. Engelhardt and Max Huth have found that this is due to impurities in the electrolyte (for instance, metal dissolved from the anode). They, therefore, employ a pure solution of zinc sulphate and an anode from which nothing is dissolved during electrolysis. This is pure massive or compact lead peroxide on a non-metallic support. If every other metal and even carbon be completely excluded the content of free sulphuric acid may be allowed to exceed 10 per cent without conversion or dissolution of the zinc already deposited. The patent is assigned to Siemens & Halske Company. (935,250, Sept. 28, 1909.)

Precious Metals.

Chlorination Process.—For the chlorination of refractory ores Mr. Wm. Koehler proposes to subject them to the action of hydrochloric acid in the presence of hydrogen at an elevated temperature, which must be below the temperature necessary for the volatilization of the metallic chlorides formed. The process is preferably carried out in a tube mill, hydrochloric acid entering together with the hydrogen. For commercial purposes very good results are obtained by using hydrochloric acid together with water gas, in which case the acid is acting in the presence of hydrogen and a carbon compound. (942,569, Dec. 7, 1909.)

Aluminium

Welding of Aluminium.—The Schoop process of aluminium welding by means of a special flux which simply serves to remove from the surface the otherwise present oxide film, but does not introduce any foreign materials into the weld, has been described in various articles in former volumes of this journal. A new improvement in the constitution of this flux is described in a recent patent of Mr. M. U. Schoop. It is a mixture of alkali chlorides with fluorides; for instance, potassium chloride 60 grams, sodium chloride 12 grams, lithium chloride 20 grams, cryolithe 6 grams. The materials are melted together and the molten mass is collected and ground to powder and then mixed to a thick paste with an aqueous solution of glycerine containing 5 to 10 per cent of glycerine. The proportions of the constituents of the welding compound are so chosen that the compound fuses at a temperature close to that of the melting point of aluminium. (943,164, Dec. 14, 1909.)

Light Metal.—Mr. J. G. Mellen patents a metal, consisting of aluminium 79 parts, magnesium 11 parts, phosphorized copper 10 parts. By phosphorized copper he means a metal consisting of 99.5 per cent copper and 0.5 per cent phosphorous. The metal, which he calls "duralum," is said to be adapted for general use as an anti-friction metal for bearings, etc., and for use in castings for motor cars, etc. It has approximately the same specific gravity as aluminium and "is very much harder and stronger, it being particularly adapted to resist great crushing strain. For the purposes for which light castings are required it is much superior to iron, as it will stand even greater compression and does not dent under blows, and while fully as light as aluminium it is wholly free from the objectionable qualities of aluminium." The metal is said to run freely in casting and to machine well. (936,339, Oct. 12, 1909.)

Aluminium Solder.—An aluminium solder, patented by Mr. G. Hartmann, consists of 80 per cent tin, 17 per cent aluminium, 0.7 per cent nickel and 2.3 per cent metallic magnesium. (938,422, Oct. 26, 1909.)

Another aluminium solder, patented by Mr. L. Goppman, consists of 49.05 per cent tin, 3.43 antimony, 26.09 lead, 20.31 zinc and 1.10 copper. (939,494, Nov. 9, 1909.)

Tin.

Detinning.—A new chlorine detinning process is referred to in a patent of Messrs. W. W. Murray and H. M. Fernberger, in which tin scrap is detinned by means of chlorine gas dissolved in carbon tetrachloride. The resulting stannic chloride also dissolves in the carbon tetrachloride. After the detinning process is completed the solvent containing the stannic chloride is withdrawn from the scrap. But a considerable percentage will be retained in the numerous interstices of the scrap, which is therefore washed again with fresh carbon tetrachloride. The stannic chloride may be separated from the wash liquid by water and the solvent recovered, an aqueous solution of stannic chloride resulting from the treatment of the wash liquid with water. After the wash liquid has been removed from the residues, the portion adhering to the residue may be removed by further washing the residue with water and by the application of heat. (943,986, Dec. 21, 1909.)

Tin from Tin-Lead Waste.—Mr. A. Nodon recovers the tin from tinned lead-plate waste, etc., by subjecting the metal waste to the action of a hot solution (40° C.) of stannic sulphate with the addition of sulphuric acid, mixed with an alkaline stannic haloid salt (ammoniacal bichloride of tin), which solution has no chemical action on lead. The tin is dissolved in the form of ammoniacal chloride of tin. The solution is electrolyzed between graphite anodes and pure tin cathodes with a cathodic current density of 1 amp to 2 amp per square decimeter. (940,471, Nov. 16, 1909.)

Electroplating.

Nickel Plating.—Dr. E. F. Kern, who has done considerable work on the theory and practice of electroplating, has invented a process of depositing nickel. A smooth, dense, coherent and adherent deposit of nickel may be obtained by the

electrolysis of a bath containing fluosilicate of nickel in solution, and particularly good results may be obtained by the electrolysis of a bath containing in addition to the fluosilicate of nickel either an alkaline fluoride, as ammonium fluoride, alone, or an alkaline fluoride and a soluble fluosilicate which is more basic than nickel, preferably aluminium fluosilicate. The electrolyte recommended by him is made as follows:

Water	100	or more	parts
Fluosilicate of nickel	10	"	"
Ammonium fluoride	5	"	"
Aluminium fluosilicate	5	"	"

Ammonium fluoride should be added to the bath from time to time during the operation of the process, in order to restrain the separation of the silica. (942,729, Dec. 7, 1909.)

Galvanizing Barrel.—To reduce the cost of electrogalvanizing by making the operation almost entirely automatic is the object of a patent granted to L. Potthoff, whose work along these lines has been described in an illustrated article in our August issue, 1907 (Vol. V, page 331). The arrangement contains a revolving barrel lined inside with cocoa-matting, in which the articles to be plated are constantly tumbling over each other, so that they are plated on all sides. When a sufficient coating of material has been formed, the direction of rotation of the barrel is reversed, whereby by an automatic device the plated articles are discharged from the barrel and passed through a chute into the washing drum. Upon further charging of the plating barrel with another lot of articles, and when the barrel is now being again rotated in the opposite direction, the articles collected in the washing drum are automatically discharged and carried successively through the draining drum, the sorting drums, and finally to the receptacles for shipment or storage. The operations of galvanizing, washing, drying and sorting (various shapes and sizes of articles being galvanized at the same time) are therefore automatic, and since a single attendant can manage a number of electroplating machines, the cost of attendance is greatly reduced. (941,535, Nov. 30, 1909.)

Nitric Acid.

Fixation of Atmospheric Nitrogen.—In the October issue, page 430, we gave an illustrated description of the Innsbruck plant where the Pauling process for the fixation of atmospheric nitrogen is in commercial operation. In this connection two recent patents of Harry Pauling are interesting. Since the process was described in detail in the article referred to, we simply give here the claims of the two patents. Patent 939,441 refers to "apparatus for producing electric discharges having in combination stationary electrodes with a distance between them large enough to prevent ignition, a conductor successively passing, and thereby alternately increasing and decreasing the distance between the said electrodes and blast pipes adapted to extinguish the discharges before short circuiting between the electrodes may occur." Patent 939,442 refers to the "process consisting in elongating the voltaic arcs produced at the place of greatest proximity of the electrodes by causing them by suitable means to propagate along the electrodes and in passing the gas to be acted upon by electrical discharges in currents of equal quantity and velocity in inclined direction against each other and meeting in the plane of the electrodes." (939,441 and 939,442, Nov. 9, 1909.)

A patent granted to Prof. Fritz Haber and A. Koenig, of Karlsruhe, and assigned to the Badische Company, is quite interesting because it digresses for good reason from orthodox principles. The chief features of their new process are the use of a reduced pressure and the intentional cooling of the arc. Lord Rayleigh had formerly investigated the effect of a change of pressure, but had found the yield to be almost independent of the pressure. The present authors have confirmed this if nothing but the pressure is changed, and if as high a temperature as possible is employed. They found that it is different, however, if the arc is cooled by artificial means. The reduced pressure assists this cooling since the discharge which at ordinary pressure possesses the shape of a thin rod, expands and approaches the walls of the apparatus, which, being cooled, help to remove the heat from the arc. In this way discharge flames can be produced which practically no longer possess the well known brown zone in which the nitric oxide is believed to decompose partially back again to nitrogen and oxygen. They use metallic electrodes or electrodes of magnesium oxide, calcium oxide, etc. The pressure is maintained at not more than half an atmosphere, while the apparatus is of such a shape that the inside is practically filled with arc flame, while the apparatus is cooled either by the natural escape of heat or by additional means of cooling. By this means the concentration of the nitric oxide in the gases leaving the apparatus is increased "and it is easy to produce gases containing from 9½ to 10 per cent nitric oxide when using air, and from 13½ to 14 per cent when a mixture of equal quantities of oxygen and nitrogen is employed." These yields of nitric oxide are obtained with it being necessary to cool the gases rapidly by removing them quickly from the sphere of reaction, for instance, by withdrawing them through metallic coolers. For instance, the aforementioned yields were obtained when the arc was burned inside a tube of quartz which was cooled with water and the air or mixture of oxygen and nitrogen entered and left the tube at a pressure of 90 mm mercury, and with the low velocity of 1.5 m per second. In order to prevent decomposition of the nitric oxide at the red-hot electrodes, the entry and exit tubes for the gases can be so placed that the layer of gases around the electrodes is not renewed. The sectional current density, i. e., the strength of the current per unit area of the cross section of the tube, should preferably be as high as possible. There are no sharp limits within which the pressure should be maintained. But no advantage is gained by reducing the pressure considerably below 80 mm mercury. In the first three claims the pressure is defined as "not exceeding that of half an atmosphere." (938,316, Oct. 26.)

Alloys.

Ferro-Titanium.—If ferro-titanium is produced with carbon as reducing agent the resulting alloy is rich in carbon, which is a clear disadvantage. If, however, silica is present the silicon reduced simultaneously with the titanium acts as a carbon excluding agent, and the alloy produced is low in carbon, but high in silicon. The silicon may be removed, however, by a following reduction with iron oxide, for instance hematite ore. Mr. F. M. Becket (Electro Metallurgical Company) proceeds, therefore, in the following way. He produces a low-carbon high-silicon ferro-titanium in an electric furnace from a charge of titanium oxide and silica, carbon, and iron or iron ore. To the high-silicon alloy thus produced is then added iron oxide, either in the same furnace or in a separate vessel, to effect the oxidation of silicon in the alloy with simultaneous reduction of the iron oxide. A low-carbon and low-silicon ferro-titanium is thereby obtained. For instance, a product containing about 30 per cent of titanium and 2 per cent of carbon, substantially free from silicon, may be made by reacting with ferric oxide on an initial alloy containing about 50 per cent of titanium, 40 per cent of silicon, 3 per cent of carbon and 7 per cent of iron. A product containing 30 per cent of titanium and 5 per cent of carbon, substantially free from silicon, may be similarly made from an alloy containing 41 per cent of titanium, 22 per cent of silicon, 7 per cent of carbon and 30 per cent of iron. (941,553, Nov. 30, 1909.)

Calcium Nitride.—A modification of the well-known Acker process is the subject of a patent of Mr. C. F. Acker. Instead of acting on the fused lead-sodium or lead-calcium alloy with steam (in order to produce the hydroxide) he acts on it with nitrogen and produces a nitride. The nitride is usually lighter than the lead and rises to the top, where it collects as a floating layer. Other nitrides can be made in the same way. (941,132, Dec. 14, 1909.)

A New Type of Distillation Machinery.

BY WALTER E. LUMMUS.

A continuous distilling column of unique design recently built by the firm of Walter E. Lummus, of Boston, Mass., for the U. S. Forest Service, and known as a three-product distilling machine, has shown results on operation which appear to warrant a brief general survey of this system of distillation.

The acceptance of the machine referred to depended entirely upon the successful outcome of a series of trial runs with a variety of materials especially selected to demonstrate the amplitude of operation. The results achieved appeared creditable in every respect and seemed to meet with the requirements of the Forest Service as well as to justify the hopes of the builder. The machine is now set up at the new Forest Products Laboratory lately established at Madison, Wisconsin, and is being used in the refining of large samples of crude wood turpentine. This work is being carried on in connection with a series of experiments undertaken by the Bureau primarily intended to lead to a more thorough utilization of the waste material resulting from the lumbering of the Southern yellow pine.

Machine Versus Apparatus.—Practically all distilling apparatus has hitherto been designed to obtain profitable results through the skill and judgment of expert operators. The financial return from such operations varies therefore with the diligence and care with which the work is conducted. But in all the more progressive arts, automatic machinery is taking the place of hand work because more reliable and more economical. The object of the Lummus distilling machine design is a compact, self-contained, and automatic machine, capable of performing work equal to that of the most skillful operators and of performing the work.

Construction.—The accompanying cut illustrates a machine capable of turning out not less than four refined products, simultaneously and in marketable condition. It is a form better adapted to general manufacturing conditions than the one built for the Forest Service, which was specially designed for experimental purposes.

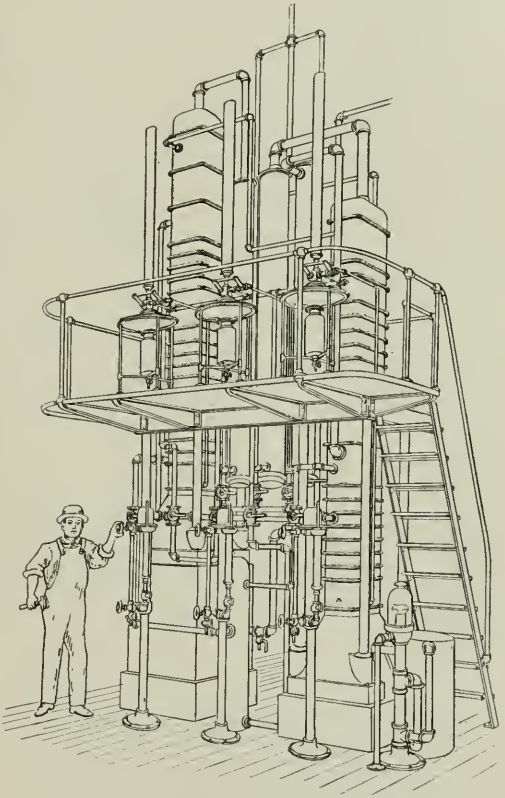
A complete description of this machine and the nature of the operations by which its work is accomplished would involve the study of multiple-effect cells and an understanding of the conflict of whirling masses of foam and vapor in what might seem hopeless complication. Really the process is one of orderly and selective evaporation and the apparatus has been reduced by scientific elimination to a sober and practical solution of what has been heretofore a vexatious and difficult task.

The machine as shown consists of two columns composed of unit sections made of suitable metals and built upon a unit system which provides for modifying the apparatus to the most favorable arrangement for the work to be performed. In the distilling chambers comprised in these columns are devices for causing the vapor to undergo fractional distillation and condensation, to the end that the ingredients of the crude material automatically assort themselves "like by like" and go their separate ways in the proportions in which they occur in the crude. Between the columns or upright series of chambers are suspended condensers at appropriate heights so that the products as they issue may flow by gravity into raised tanks from which, in turn, they may be drawn with convenience for shipping.

In front of the machine and at a height convenient for operation, are speed gauges or spiritometers especially designed for the purpose which show the rate of flow of each product, its gravity and temperature. Between the gauges is the switchboard or bank of operating valves, arranged with simple and methodical relation to the gauges so that the operator may observe his gauge and manipulate the regulation of the machine to the exact result desired.

The makers emphasize specially the simplicity of operation, the compactness, and workmanship of these machines in contrast with the complicated construction and enormous bulk of foreign apparatus intended to turn out similar products. In regard to compactness it may be well to add that the particular machine illustrated occupies a floor space about 3 ft. 6 in. x 7 ft. 6 in. and is 17 ft. over all. The capacity, on the other hand, is between 600 and 720 gals. of crude material in 24 hours, or 25-30 gals. per hour.

Operation.—Continuous distillation is not new in Europe, and there are simple continuous "beer stills" in modern American distilleries. The latter, however, are not depended upon to make finished products but only to concentrate material that is to undergo later more thorough rectification. The advantages of continuous operation will be obvious to everybody.



MULTIPLE-PRODUCT DISTILLATION MACHINE.

The distilling machine described is operated continuously and no boiler or kettle is employed, nor any system of tanks to hold liquors in various stages of completion. When once properly adjusted the material travels without pause through the various chambers, emerging from appropriate condensers according to composition. The operation may be continued indefinitely so long as a uniform supply of crude material, steam, and cooling water are maintained. Very sensitive regulators control the pressure and temperature and even the gradually diminishing head of the supply tank from which the raw material flows to the machine. To this feature of absolute control the uniformity and improved quality of products are largely due.

The most remarkable feature of this machine, however, and one which perhaps indicates an entirely new development of industrial engineering, is the feature of "multiple products."

Not only is the main product isolated in pure condition from the complex extract in which it occurs as "crude," but advantage is taken of the favorable condition of the residue while yet hot to further assort the mixed gases in compartments designed for the purpose. In this manner each component of commercial value is likewise isolated and flows from its respective condenser in a manufactured condition.

Economy.—The economy of operation of these machines is especially worthy of attention. Since the raw material enters only as fast as the products leave, it follows that the quantity of material present in the machine at any one time can be reduced to that actually undergoing distillation and the steam and water consumption are correspondingly small. By a further economy, without compensating disadvantage of any kind, the raw material is heated to boiling temperature before entering the machine proper, with the waste heat of the products or effluents. In ordinary stills, a quantity of material sufficient for the entire operation must be maintained at the boiling point throughout the entire period, thus consuming many times the fuel required for evaporation, and no practical means has been found to utilize the heat of residues or distillates except in rare and unimportant instances. The machines, as built, have a capacity, according to size, far in excess of all previous records. The interaction of gases and liquids, upon which the efficiency depends is here produced by devices employing several new principles of distillation, and the superior yields and quality are partly due to these improvements.

Cost.—Not only is the cost of manufacturing with these machines far less than with the usual equipment of stills and tanks, but even the first cost of installation is less with a consequent saving of interest, depreciation and repairs. The Lummus distilling machines require no special building construction.

To sum up, there are two main distinctions between these distilling machines and the ordinary apparatus of well-known form.

Firstly, this new type may be termed a machine in every sense of the word, being compact, self contained and automatic. It is especially designed to meet American conditions, which, owing to the high price of labor, tend directly toward automatic machinery.

Secondly, the elastic unit system employed in construction permits the adapting of these machines to the special requirements of any crude material. The most unusual demands may be met with an exactness that has heretofore been considered impossible.

These two points alone explain the ready application of the Lummus machines to a vast number of distillation problems. Wood alcohol, turpentine, petroleum, fusel oil, tar oils, etc., which for years have been manufactured by tedious periodic processes are with these machines easily separated in one continuous operation into their ultimate constituents or groups of constituents, conforming to commercial specifications. Furthermore, products that have heretofore defied economical manufacture on account of prohibitive cost of periodic refining, are now brought within the range of profitable production.

Slime Filtration.

A new filter press placed on the market by the Sweetland Filter Press Company (Ernest J. Sweetland, General Manager, Union Trust Building, Los Angeles, Cal.) is of special interest, as it is a pressure filter, but is designed to have the chief advantages of vacuum filters, especially from the point of accessibility and the convenience of having to open the filter chamber for cake discharge.

The general appearance of the Sweetland filter press is shown in Fig. 1. The press is assembled on a plant of about one inch per foot, while the filter leaves hang vertically.

In order to gain the advantages of accessibility, the Sweetland filter press (Fig. 1) is built up of a plurality of sections. Each section is practically complete in itself, and, all being alike, they are interchangeable. In this way the capacity of the filter may easily be enlarged at any time it should be desired, for the enlargement of the filter merely requires the addition of more sections, and any number may be inserted by lengthening the tie rods sufficiently to accommodate them.

Each section consists of a cast-iron frame, the upper portion of which is in the form of a rectangle, while the lower walls

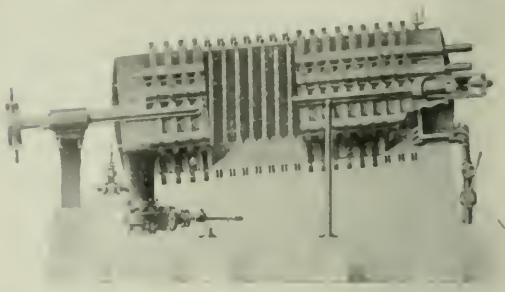


FIG. 1.—FILTER PRESS.

converge, forming a triangular opening in the lower part of the frame below the filter leaf (Figs. 2 and 3).

A special feature is the form in which the frames are cast; they have an I-beam section (instead of a rectangular or square section, as the old type of frames). In other words, there is a strengthening rib on the outer surface of the frame. This rib is in the form of a thin web of metal cast upon the outer wall of the frame, at right angles to it, with a bulb cast upon its outer edges. This construction results in great strength and light weight. While the main wall of the furnace is only about $\frac{3}{4}$ in. thick, the frames stand an internal pressure of 100 lb. per sq. in. without injury.

Fig. 2 shows one of the frames. Both sides of the frames are planed to a smooth, flat surface. On one side a groove $\frac{1}{4}$ in. deep and $\frac{1}{8}$ in. wide is cut into the frame, into which a rubber gasket $\frac{1}{8}$ in. square is inserted, which abuts against the flat side of the adjacent frame. This gasket is firmly impressed within

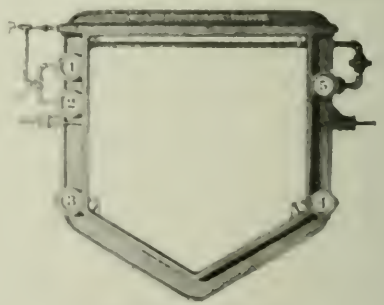


FIG. 2.—FRAME SECTION.

the groove, and forms a water tight joint up to 100 pounds pressure per square inch.

The spray pipe is clearly shown in this figure, extending horizontally across the top of the frame just below the upper wall.

There are five circular openings or passageways through each frame. When the frames are aligned in position in the press these passageways form continuous conduits throughout the length of the press and terminate in pipe connections at the upper head.

When the filter leaf is in position (Fig. 3) it is connected by a union to the frame at the upper right hand corner. The upper

left-hand corner is held in place by a small clip, while a pair of lugs attached to the frame support the two lower corners. The union and pipe connection at the upper right-hand corner brings the interior of the filter leaf into communication with the frame. There is a horizontal passageway through the top of the frame, which extends from one side to the other and terminates at the left-hand side in a pipe connection. Other pipe connections are provided leading into conduits Nos. 1 and 2 (Fig. 2), and join in a lever-handled three-way valve, the third opening of which connects with the pipe communicating with the filter leaf. The filter leaf may therefore be brought into communication with either conduit No. 1 or No. 2, according to the way the three-way valve is turned. The arrangement is plainly shown in Fig. 2.

A test-cock is shown on the pipe connection, the object of which is to provide separate communication with each leaf, so that in case a leak should occur at any time the filter leaf containing it may be immediately located and shut off from further filtration until such time as it is convenient to repair the leak or replace the leaf.

During filtration the three-way valve is turned to allow the filtrate to drain into conduit No. 2. At all times conduit No. 1 is kept filled with compressed air under moderate pressure, or water under a 12 to 2-foot head supplied by a pipe connected with the upper head of the press, and when it is desired to discharge the slime cake from the filter leaf the three-way valve is turned to allow the compressed air or water to enter the interior of the filter leaf from conduit No. 1, which causes the cake to drop instantly.

Each frame is provided with two small nozzles located near the lower corner of the leaf, which communicate with conduits Nos. 3 and 4 (Fig. 2), respectively. The object of these nozzles is to remove the last traces of slime which may adhere in the corners of the press after discharging. If the pressure of the pump is turned into these conduits for about 30 seconds after the bulk of slime has been discharged, the body of the press is left absolutely clean for the following cycle of operations.

Conduit No. 5 (Fig. 2) supplies the spray pipes with liquid. Each spray pipe is provided with an individual valve connection between the pipe and conduit, so the full pressure of the water supply available may be turned into all the spray pipes simultaneously, or into a few of them at a time, as may be desired. These spray pipes are not used in the ordinary cycle of operations, but for occasionally washing the filter leaves.

The filter leaf which is shown in place in the frame (Fig. 3)

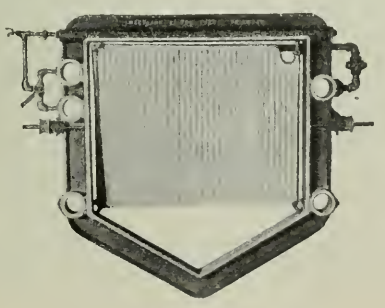


FIG. 3.—FRAME, WITH FILTER CLOTH.

comprises a series of vertical corrugated wooden strips covered with a layer of canvas, which forms the filtering medium. Each leaf is provided with a drain pipe terminating in a union outside of the leaf. To place the leaf in the frame it is only necessary to connect the union after inserting the leaf between the lugs at the other three corners. The filter leaves (standard size, 3 ft. 2 in. by 4 ft.) are so easily taken out and replaced in the frames that two men can easily remove a set from a 40-frame press, replace them with fresh ones, and have the press ready for operation again in about an hour.

The spray pipes are clearly shown in Fig. 1 between the filter leaves in the upper part of the press, and the discharge nozzles are shown entering from the upper head of the press, in line with the lower walls of the frame. There are five discharge nozzles, each ½ in. to ¾ in. in diameter, depending on the length of the press. All are controlled by a single quick-open-

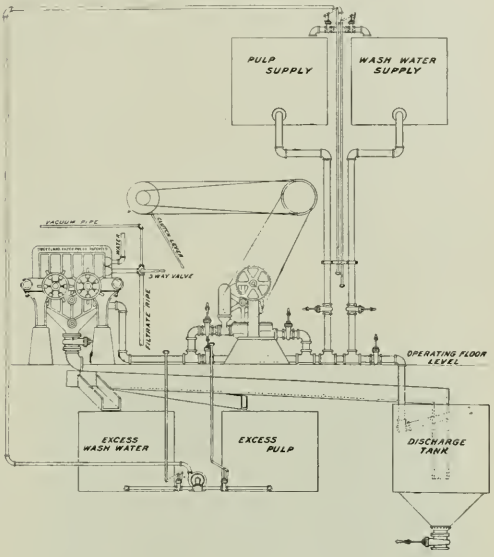


FIG. 4.—ARRANGEMENT OF FILTER PLANT.

ing valve on the discharge line of the plunger pump. The purpose of these nozzles is to flush the slime residues from the press when caused to fall from the filter leaves.

The discharge valve is plainly shown in this figure, attached to the hopper on the lower head of the press. This is an 8-in. quick-opening valve, and is used whenever it is desired to empty the press either of excess pulp or wash water, or when discharging the residues.

When the press is installed, there are three launders below the discharge valve; one for excess pulp, one for excess wash water and one for discharge. By means of a revolvable switch (not shown in Fig. 1) the discharge from this valve may be directed into any one of the three launders.

A suitable arrangement of a filter press plant intended for 100 tons' daily capacity is shown in Fig. 4. In this arrangement the filter press and pump which operates it are placed on a level below the pulp-supply tank and wash-water supply tank, and on another level just below the filter press are the excess pulp tank and excess wash-water tank, and a small centrifugal pump to return the excess pulp and wash water to the supply tanks on the upper level. By this arrangement the filter press is both filled and emptied by gravity, and the necessity of a large pump for rapidly filling and emptying the press is eliminated. A difference in level of 25 ft. between the supply tank floor and the excess tank floor is sufficient to permit of this arrangement, although more fall can be used to advantage if available.

The pump shown on the operating floor level is a triplex outside-packed pump, the size of which varies according to the size of the installation. This pump serves two purposes, viz., that of forcing the sludge or wash water into the press during filtration and washing, and to deliver water to the discharge nozzle while the press is being discharged. As a greater volume is required

during discharging than during filtration, the pump is so arranged as to have two speeds.

The filter press is first filled with pulp from the pulp supply tank through the by pass under the pump.

As soon as the press is filled the low-speed clutch of the pump is thrown in which forces the pulp into the press under 40 or 60 lb pressure. A relief valve is provided, which keeps the pressure from rising above the desired point. As the pulp is forced in, the clear filtrate passes through the canvas of the filter leaves and out through the conduit No. 2 into the filtrate pipe, and the slime is deposited on the surface of the filter leaves, forming a cake from 1 in. to 2 in. in thickness, depending upon the length of time the operation is continued.

When the cake has reached the desired thickness the pump is stopped and the three-way valve on the filtrate pipe turned so as to bring the vacuum pipe in communication with the interior of the filter leaves, the object of which is to hold the cakes intact on the filter leaves while they are relieved of pressure. A vacuum of 5 in. only is rquired to hold the cakes in place. The swivel joint or switch, just below the discharge valve, is now turned to direct the flow into the launder leading to the excess pulp tank, and the discharge valve is opened to allow the excess pulp to rapidly drain out the press.

The discharge valve is then closed and the press filled with wash water (or barren solution, as the case may be) through the by-pass. When the chamber of the press has been filled the pump is again started at low speed, pumping the wash water through the cakes to displace the dissolved gold or silver values. The displacement is very rapid under 40 to 60 lb pressure, and usually takes about ten minutes.

The excess wash water is drained from the press into the excess wash-water tank, and the slime cakes may now be discharged as follows.

The switch below the discharge valve is turned into the discharge launder. The valve on the suction line of the pump, which leads to the cone-bottom discharge tank, is opened. The pump is started at full speed, which delivers a stream of water from the top of the discharge tank to the discharge nozzles under about eighty pounds pressure. There are five of these nozzles, all of which are in operation at the same time, and a stream of water of high velocity is caused to sweep the bottom of the press. With these streams in operation all that is necessary is to turn down the lever handle upon each of the filter press frames, which causes the water or compressed air in conduit No. 1 to enter the filter leaves, causing the cakes to fall from the leaves to the bottom of the press, one at a time. As fast as they fall they are partially disintegrated and carried away by the streams from the nozzles into the discharge launder, which deposits the residues in the cone-shaped tank.

Each cake is given about ten seconds to fall before the next one drops, which is ample time for the leaf to become thoroughly cleaned by the reverse current through the pores of the canvas, and for the cakes to be carried away. Fresh water is not required for use in discharging the residue, but the discharging water is merely circulated from the top of the cone bottom discharge tank through the bottom of the filter press back to the discharge tank again.

In time a cleaning of the canvas becomes necessary, due to the formation of a deposit of lime and silica etc., which requires acid treatment and scrubbing of the canvas. For this purpose the Sweetland Filter Press Company have developed a special revolving brush for cleaning the filter leaves.

From tests made with the Sweetland filter brush it appears that the rate of filtration is practically in direct proportion to the pressure at which the liquid is applied to the filter surface.

Brazing Plant.

BY RICHARD N. HART.

Unless a broken iron casting is replaced by a new piece, there is one kind of repair that will restore its original strength. This is by brazing. A well-brazed fracture in cast iron adds from 10 to 25 per cent to the initial strength. This is because the brass amalgamates or alloys with both broken surfaces. Brass is stronger and tougher than iron.

It is generally much cheaper and quicker to braze a fractured casting than it is to buy a duplicate part. For this reason some of the larger foundries and machine shops have brazing apparatus of their own, or an independent brazing shop will be set up in the midst of a community of factories. Such a shop can make a large margin on its repair works and save a still larger sum for the manufacturer. Occasionally, when the part to be repaired is part of an apparatus of foreign make, the saving in time, anxiety and money may be very great. Every mining camp of any importance, at a distance from supplies, should certainly have a brazing outfit.

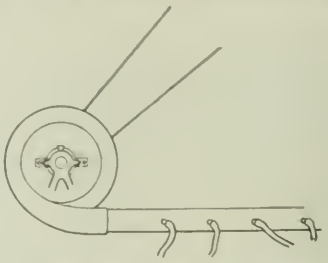


FIG. 1.—BLOWER AND AIR TUBE FOR BRAZING PLANT.

The brazing outfit here described can be installed in large shops, or can be set up in an independent repair shop. It consists of torches using gas and air blast, a makeshift brick oven and miscellaneous tools, such as a vise, hammer, files, wire brushes, etc.

The city gas at a pressure of about two pounds is used. The air blast is furnished by a rotary fan playing into a long cylinder, as shown in Fig. 1. The fan is driven by a 2-hp or 3-hp motor, and gives a pressure of six to eight ounces. As many as six burners can be attached to the air cylinder at once; for heating large castings this may be necessary.

The torch is a two-way gas pipe, Fig. 2, mounted on a tripod with a swivel joint and connected to the gas meter and the air cylinder by two wire wound rubber tubes. These tubes should be long enough to allow the burners to be moved about freely.

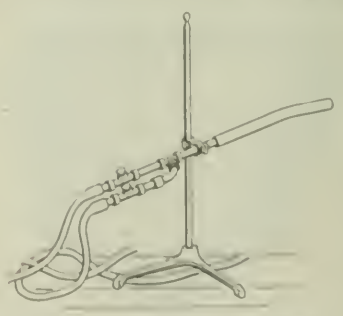


FIG. 2.—AIR-GAS TORCH FOR BRAZING.

The furnace may be made of loose brick built up around the casting so as to hold it in position and to contain the heat well. The table can be protected by asbestos cloth. The roof of the brick furnace can be held up with iron bars, as shown in Fig. 3, unless the casting is small enough to allow the roof to be made of one brick. The front of the furnace is left open and the torches are played directly into the front. No vent is needed

for the gases of combustion, because the crevices between the bricks are large enough, and the heat is thus better retained.

Briefly stated, a broken casting is repaired as follows: The fractured surfaces are brushed with a stiff wire brush, and the surfaces near the fracture are cleaned of paint, rust and grease with file, brush or waste. Paint the fracture surfaces with an emulsion of bronze filings, sal ammoniac, borax and water. Fit the broken pieces together and build the bricks up around the

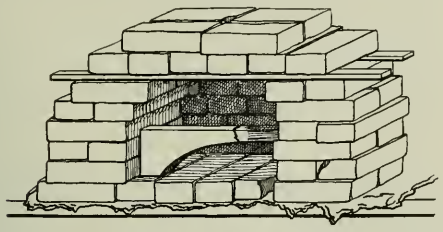


FIG. 3.—MAKESHIFT BRICK FURNACE FOR BRAZING, SHOWING BROKEN CASTING IN POSITION.

casting. Be sure that the pieces are fitted as firmly and closely as possible. They may easily separate slightly from the motion of the bricks while you are making the furnace. Next bring the mouth of the burner up to the open front of the furnace, point it toward the break in the casting and light the gas.

Regulate the air-gas mixture so that the flame has a distinct blue cone—*not purple*. If the casting is large, use two, three or more torches. When the metal is red-hot, apply additional flux to the fracture joint with an iron rod and rub it well along the crack. The flux is calcined borax or a mixture of calcined borax with sal ammoniac, salt, etc. Then shovel on a spoonful of brazing spelter, which will melt and run wherever the flux has cleaned the metal. Use plenty of spelter. Half-and-half brass may be used, or, better still, a brass of 70 per cent copper, which will be stronger and less apt to lose its zinc by burning.

The casting may now be cooled and cleaned. The cooling, like the heating, must be very carefully done if the casting is

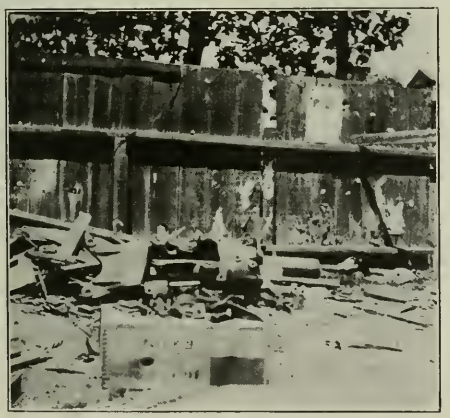


FIG. 4.—BROKEN MOTOR CYLINDER.

large or delicate or liable to break through expansion. Cast iron will gain in strength from 10 to 25 per cent by brazing, and will not break again in the same place. Steel, malleable iron, brass and bronze can also be brazed.

Figs. 4 and 5 show a motor cylinder that was broken into 33 pieces and brazed together by the Cast Iron Brazing Company, of Los Angeles, Cal. It was necessary to hold it together with wire. The entire operation took a little over two days. Care is necessary in heating up motor cylinders, otherwise they will have to be reground because of unequal expansion. Or if the flame is oxidizing (purple) it will rust the bore of the cylinder.

The cost of the apparatus can be brought within $400, or even $300. If the outfit is put up in the machine shop, and the power is taken direct from the shafting, a $150 motor can be dispensed with. If the outfit is put up at a mine, the gasoline blower from the assay office will take the place of motor, fan, cylinder and torch. In fact, it is a question whether a good gasoline torch would not be as safe, cheap and easy going as the outfit already named.

The Cast Iron Brazing Company estimate their gas and power cost at from 15 to 20 cents per hour for one burner, and about 10 cents additional for each extra burner. A casting of, say, 25-pound weight can be brazed in about one hour, counting cleaning before and after. The labor cost would be about 40 cents. The approximate cost of a one-hour repair job, provided the plant is in steady operation nine hours a day, is roughly as follows:

6 per cent interest on $400 apparatus	$ .01
10 per cent yearly depreciation on same	.02
Rent $200, taxes, insurance	.10
Power and gas	.20
Labor	.40
	.73

Between this cost and the average value of the casting to be

FIG. 5.—REPAIRED MOTOR CYLINDER.

mended there is a very wide margin, allowing a substantial profit.

Whenever this brazing outfit is needed it is badly needed. A mine, or any isolated plant using machinery, is sure to be tied up occasionally by the breaking of parts. Delay is always tremendously costly and exasperating. A trifling outlay, such as just described, will prevent breakages from demoralizing the plant.

Gas Velocity Recording Device.

A new velocity gauge for gases, which is being placed on the market by the Hydro Manufacturing Company, 720 Lewis Block, Pittsburgh, Pa., is interesting, as it is based on a novel principle.

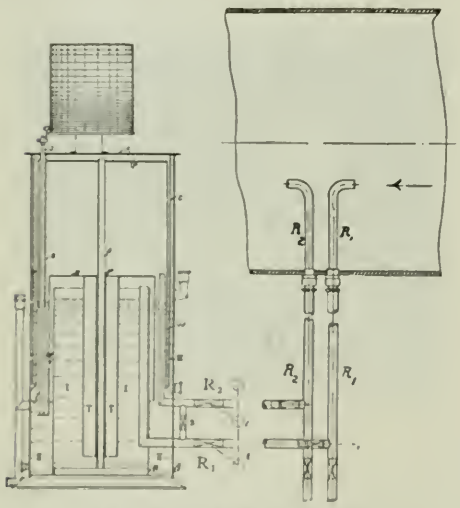
The gauge is shown in the adjoining diagram. The gas, the velocity of which is to be measured flows through the flue shown on the right hand, in the direction of the arrow. Within the path of the gas the ends of two pipes, R_1 and R_2, are placed. The open ends of both pipes are bent, but in opposite directions. That of R_1 is bent opposite to the flow of the gas, that of R_2 is bent in the same direction as the flow of the gas. This causes a difference of dynamic pressure in the two pipes R_1 and R_2,

and this difference of pressure depends solely on the velocity of the gas. By measuring the pressure difference, we therefore get an indication of the velocity of the gas.

The pressure difference is measured in the apparatus shown at the left. It consists in the main of two cylindrical compartments, one within the other. Both are partly filled with liquids, of which the inner one serves as operating medium and the outer one as a water seal. The inner liquid *I* carries a float *T* to the top of which is fastened an inverted cup-shaped cylinder *D*, which dips with its lower open rim *M* into the outer sealing liquid *II*. In this manner two independent gas chambers are formed, both of varying capacity, depending upon the depth of the immersion of the bell, which in turn depends directly upon the variation of pressure within and without.

A rod *S* that is centrally fastened to the bottom of the main cylinder serves as a guide for the operating bell. Its upper end passes through the removable cover plate *P'* of the apparatus.

The movement of the float or bell is carried outwardly by means of a second rod *J*. This rod is hinged to the lower portion of the bell and it passes through a tube *K*. The top end of this tube is secured air-tight to the cover, and its lower end dips into the sealing liquid *II*. The rod carries at its upper end a writing pen which bears against a rotating cylinder operated by clockwork, and thus serves to trace the varying differential pressures of the system upon a chart.



A VELOCITY METER AND RECORDER

An additional water seal *H* is provided for the cover by a cylindrical portion *C* extending therefrom downwardly into the sealing liquid *III* of an annular space, formed within the upper portion of the main body.

Communication between the gas main and the apparatus is established by two pipes *R*, leading into the air space below and *R*, leading into the space above the movable bell. Each of these pipes is provided with a stop-cock, 3, and both cocks are linked together to make them work simultaneously. By these means it is made possible to distribute the pressure to both sides of the bell at the same time. A three way cock serves to make communication of both air spaces in the apparatus with the atmosphere.

The filling of the three independent liquid chambers is accomplished through pipes which lead out from the main body of the apparatus. Plugs are provided for at the lower end of these pipes, which allow easily draining the receptacle if this should be necessary.

The velocity of the gas in the flue causes, as explained above, a difference of pressures in the two pipes R_1 and R_2, and this pressure difference causes the movement of the bell which is transmitted to the writing pen on the recording device.

This recording velocity gauge should be useful in gas works, producer gas installations, coke oven plants, blast furnace plants, and in general in metallurgical and chemical works in which the success of the process depends on a definite rate of supply of air or gas. Naturally the instrument is also useful in boiler plants, where it is employed to check the correct air supply for the most economical combustion of the coal; in this case the record chart indicates every operation of the fireman or of the automatic stoker, and serves as a guide to the fireman in which manner to obtain the best results.

Notes.

The **Zaremba Company**, builders of evaporators and chemical machinery, announce the removal of their engineering department and Eastern office from Philadelphia, Pa., to 1044 Ellicott Square, Buffalo, N. Y.

Corubin.—A valuable by-product, obtained in the production of chromium by the thermit process, is known as "corubin" (registered trade-mark), which has proved to be one of the best abrasive materials obtainable. It is used extensively abroad in the manufacture of high-grade emery wheels, emery cloth, and for other purposes where a first-class abrasive is required. Arrangements have now been completed for supplying the American market with this material, and the Goldschmidt Thermit Company, 90 West Street, New York, will be glad to give further particulars.

Helion Resistors.—The Parker-Clark Electric Company announces the retirement of Mr. W. G. Clark from his connection with the Company and the election of Dr. George M. Miller to the office of president and director. Prof. H. C. Parker continues at the head of the laboratory work. The offices of the company have been removed to the Lincoln Square Building, 1960 Broadway. The company is now placing on the market the helion resistors which are being made by the action of silicon on carbon for use as resistors in electric heating apparatus, etc. Anticipating a large demand for these resistors, the company plans to start a factory for their manufacture as soon as a good location can be selected.

A Study of the Open Hearth is the title of a very interesting little book just issued by the Harbison-Walker Refractories Company, of Pittsburgh. It is a concise and very clear treatise of the open-hearth furnace and the manufacture of open-hearth steel. The first and principal chapter deals with the open-hearth furnace and gives many useful data on refractories used in it. The other four chapters deal with fuels, the acid open-hearth process, recarburization, the basic open-hearth process and special processes (Talbot, Monell, Bertrand Thiel, Duplex). Some parts of the book deserve a more detailed review, which we reserve for a future issue.

The Palmetto Phosphate Company, of Tiger Bay, Florida, has recently put in operation its new producer gas electrical generating plant. The first of the three units was started Nov. 23, the second Nov. 26 and the third Dec. 2. Briefly, the equipment consists of one 1100 hp gas producer plant in three units, three 18 x 24 in. single-tandem gas engines direct connected to 800 kw. three-unit generators together with the necessary exciters, transformers, switchboards, etc. The entire plant is now electrically operated. The Allis-Chalmers Company are the builders of the apparatus.

Oxy-Hydrogen Welding and Cutting.—While the oxy-acetylene process has been introduced to a large extent in this country both for welding and cutting, the older analogous process in which the oxy-hydrogen flame is used, and which is employed largely in Europe, has been comparatively unknown in this country. A change in this condition is now to be made by the American Oxhydric Company, of Milwaukee, Wis., the owners of the American patents of the Societe l'Oxhydrique

Internationale for apparatus for the welding and cutting of metals by means of the oxygen-hydrogen flame. The American company thus has the benefit of years of experience with this apparatus in France, Germany, Belgium, Italy and Russia. The company manufactures the oxygen and hydrogen gases by electrolysis of water.

The **West Steel Casting Company**, Cleveland, O., who for the past three years have built up quite a large trade in medium and small steel castings made by the converter process, have lately enlarged their plant and are now installing crucible furnaces which will supply special small castings demanded by the automobile and general trade. The converter metal will be utilized as heretofore for casting of West's "electrical steel," which is largely used for dynamo frames and other electric castings. This same metal is also found valuable on account of its good strength and ductility for blank gears and general steel castings. The officers of the company are Mr. Ralph H. West (son of Mr. Thos. D. West), president; Mr. Everett W. Pike, vice-president, and Mr. David P. Lansdowne, secretary.

Cyanide Process in Mexico.—The first plant to be built in the famous Parral gold mining district of Mexico for the treatment of ore by the cyanide process has been decided upon by the Palmilla Milling Company, of Parral, State of Chihuahua. The plant is designed primarily for treating the ore of the Palmilla Mine; but will also be arranged for treating custom ore. The "Vetacolorada" or "red-vein" of the district is capable of producing many thousands of tons of ore of reasonable grade which can be treated successfully by cyaniding and concentration. The cyanide part of this new plant will include the best in filter and air agitation tank construction, though these latter details have not yet been decided upon. The machinery has been purchased from Allis-Chalmers Company, Milwaukee. Stamps are of latest design. The ordinary accessories such as grizzlies, crushers and conveyors are provided. All wearing parts were specified to be of specially tough steel. Three tube mills are included, which will be 5 ft. in diameter and 22 ft. long and will have the latest features of construction, such as automatic spiral feeders and ribbed liners.

Mr. **Elmer A. Sperry**, as has been well known, did a great deal of work several years ago in collaboration with Mr. C. P. Townsend to reduce to practice what is now well known as the Townsend cell for the electrolytic production of caustic soda and chlorine. This cell has now been in continuous successful use on a large scale for several years at Niagara Falls. It has not, however, been generally known until the present announcement that during this time of collaboration of Mr. Sperry with Mr. Townsend several further processes were developed and tested on an extensive scale. These include processes for the electrolytic treatment of ores in a fused bath, processes for the production of lead oxides, white lead (of low or high oil absorption), and pigments; also a process for tin ore treatment. These processes are said to have all the same characteristics as the caustic soda and bleach process of Townsend, namely, low first cost, directness of application, and economy of maintenance and power consumption. Further details concerning these processes may be had from Mr. E. A. Sperry, 40 Wall Street, New York City.

Obituary.

John Wellington Nesmith, president of the Colorado Iron Works Company, aged 76, died on Friday morning, December 17, of heart failure following an operation for gall stones. He had been in splendid health and attending to his usual business occupations until Monday, and his death came as a shock to his relatives and friends. Mr. Nesmith had been identified with the progress of the West from the early days, coming to Colorado in 1860, engaging in mining and milling in Gilpin County. He subsequently became interested in Langford & Company's foundry, the nucleus of the Colorado Iron Works, of which he became manager in 1879 and president in 1886. In the advancement in ore-treatment methods he took an active interest, and specialized in the blast-furnace smelting of lead and copper ores from the time of the first application of the process, achieving a world-wide reputation for the Colorado Iron Works in this line. Mr. Nesmith was a past master Mason and a past high priest of the Royal Arch Masons, vice-president for Colorado of the National Association of Manufacturers, a member of the American Institute of Mining Engineers, the Colorado Scientific Society, and of various other organizations.

Digest of U. S. Patents.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ELECTRIC FURNACES (Continued).

No. 643,898, Feb. 20, 1900, Harry M. Hill, St. Louis, Mo.

Resistance type. Oven for dental use, assaying and melting; an inclined crucible of fire clay or magnesia, having an external spiral groove receiving a doubled platinum wire of 36 gage, the groove being deeper than the diameter of the wire. The crucible is surrounded by windings of asbestos or mineral-wool paper, applied moist and baked. Before applying this paper the crucible is heated, and shrinks. The wire resistor is then rewound. The crucible is closed by a plate of refractory material, also grooved externally and receiving a resistance wire.

No. 645,284, March 13, 1900, Edward G. Acheson, Buffalo, N. Y.

Resistance type, for heating, roasting, and especially for purifying and graphitizing carbon. The furnace chamber is rectangular, two sides sloping at an angle corresponding to the angle of repose of the charge-mixture to an opening at the bottom, beneath which is a screw conveyor rotating in a water-cooled tube. Opposed electrodes pass through the walls of the chamber near its top. For the production of 500 lb. of graphite from coke per hour, the electrodes have a cross-section of 1250 sq. in. and are spaced apart 20 in., passing a current of 37,500 amp. They are composed of a number of rectangular bars arranged in contact. Pipes carrying water for cooling the product pass transversely through the lower portion of the chamber. The furnace has a suitable supply hopper, and vents which may lead to condensing chambers.

No. 647,614, April 17, 1900, Marcus Ruthenburg, Philadelphia, Pa.

Furnace for smelting and refining ores, comprising a chamber in the form of an inverted quadrilateral pyramid, formed of slabs of refractory material, opening beneath into a separate hemispherical crucible. Horizontal carbon electrodes pass through water jackets into the opposite sides of the chamber, preferably carrying 1000 amp per square inch of cross-section. The molten product, gravitating into the crucible, overflows through a spout at one side into a wheeled receptacle.

No. 650,040, May 22, 1900, Ernest W. Engels, Essen, Germany.

Bricks, slabs and plates of silicious, clayey or other refractory materials are rendered fire and acid proof by coating them with a layer of carborundum or other carbid, melted or fused on by an electric arc, for example, one of 200 amp at 110 volts. Either the pre-formed carborundum or the materials for producing it, sand and charcoal, may be placed on the surfaces of the bricks and then fused, uniting with the brick.

NEW BOOKS.

FINLAY, J. RALPH. The Cost of Mining. An exhibit of the results of important mines throughout the world. 424 pages, illustrated, maps. Price, $5.00 net. New York: McGraw-Hill Book Company.

BASKERVILLE, C. General Inorganic Chemistry. 364 pages. Price, $1.50. Boston: Heath.

BASKERVILLE, C., and CURTIS, ROBT. W. Laboratory Exercises to Be Used in Conjunction with General Inorganic Chemistry. 57 pages. Price, 40 cents. Boston: Heath.

KAHLENBERG, LOUIS. Laboratory Exercises in General Chem-

istry. Second edition, revised. 208 pages. Price, 90 cents. Madison, Wis. (L. Kahlenberg, University of Wisconsin.)

Peckham, Stephen Farnum. Solid Bitumens. Their physical and chemical properties and chemical analysis; together with a treatise on the chemical technology of bituminous pavements. 332 pages. Price, $5.00. New York: M. C. Clark Publishing Company.

Coyle, D., and Howe, F. J. O. Electric Cables; Their Construction and Cost. 473 pages, illustrated. Price, $5.00 net. New York: Spon & Chamberlain.

Franklin, W. Suddards. Electric Waves. An advanced treatise on alternating-current theory. 324 pages. Price, $3.00 net. New York: Macmillan.

McClung, R. K. Conduction of Electricity Through Gases and Radioactivity. A textbook with experiments, with 78 illustrations. 201 pages. Price, $1.50 net. Philadelphia: P. Blakiston's Sons & Company.

Reeve, Sidney Armor. Energy, Work, Heat and Transformations. 238 pages. Price, $2.00 net. New York: McGraw-Hill Book Company.

Malcolm, C. Wesley. A textbook on graphic statics. 328 pages. Price, $3.00. New York: M. C. Clark Publishing Company.

Molesworth, Sir Guilford Lindsey. Metric Tables. Fourth edition, revised and enlarged. 104 pages. Price, 80 cents. New York: Spon & Chamberlain.

Russell, G. E. Textbook on hydraulics. 190 pages. Price, $2.50. New York: Henry Holt & Company.

Stephens, J. Rittenhouse. Principles of Accounting. 91 pages. Price, $1.25. Kansas City, Mo.: John R. Stephens, Keith & Perry Building.

Hathaway, C. E., and Griffith, Ja. Bray. General expense and cost summaries; instruction paper. 70 pages. Price, 50 cents. Chicago: American School of Correspondence.

BOOK REVIEWS.

A Course in Inorganic Chemistry for Colleges. By **Lyman C. Newell,** Ph.D., Professor of Chemistry in Boston University. Half-leather bound, 604 pages, 89 cuts. Price, $2. Boston: D. C. Heath & Company.

General Inorganic Chemistry. By **Charles Baskerville,** Ph.D., Professor of Chemistry in the College of the City of New York. Cloth, 357 pages. Price, $1.50. Boston: D. C. Heath & Company.

A Laboratory Manual. By **Dr. Charles Baskerville** and **Dr. Robert W. Curtis.** Bound, 64 pages. Price, 40 cents. Boston: D. C. Heath & Company.

The absolutely best chemical text-book will never be written, so there is always room for a better one.

The books noted above are faithful attempts to improve on existing text-books, in the teaching of first-year college students. Of the first, we may say that it is a very carefully written work, following the style of the best of the older text-books, is a mine of information and is full of clear and convincing explanations, while not breaking away from the methods of the older text-books, we may say that it is certainly a most excellent *vade mecum* for the earnest, industrious student.

The second book is a "departure," an attempt to part company with the older type of text-book. The prevailing idea is "to accomplish the most in the least time." To this end, statements are short and precise, the style is even "sketchy," but in a bright student it should be attractive and inspiring. We would rather have seen the motive of the work "the accomplishing of the best use of the time available." The other principle is likely to result in superficiality, even in "cramming." But Dr. Baskerville's book and methods, under the charm of the personal enthusiasm of a similar teacher, will probably result in fascinating many young men with chemistry, particularly students of the quick, bright, city type, with which many schools and colleges have principally to deal.

Dr. Newell's book is conservative, excellent, reliable, and highly instructive; Dr. Baskerville's is radical, at places brilliant, not so reliable, but inspiring. The teacher will at once perceive that each book will be the best for certain classes of students.

* * *

The Elements of Metallography. By **Dr. Rudolf Ruer,** privat-docent at the University of Gottingen. Authorized translation by C. H. Mathewson, Ph.D., Instructor in Metallography at Yale University. Octavo, xiv + 342 pages, 119 illustrations. Price, $3 net. New York: John Wiley & Sons.

One is surprised and pleased to find that this work is not another contribution to the merely microscopic study of metals and alloys; in fact, the microscope is kept in the background throughout, being used for "supplementing the results of thermal investigation." The basis of the treatment is the thermal study of melting points and cooling curves, with all the deductions therefrom.

The physico-chemical theory is carefully and clearly explained. The one-component, two-component and three-component systems are systematically considered, the discussions being very lucid and interesting. This forms the bulk of the book.

Another chapter on "Practice" gives the technique of the thermal measurements, some very good advice on pyrometry and a brief description of preparing microscopic sections.

Altogether, it is a book for the thorough study and mastery of the best methods of investigation of metals and alloys. We agree with it entirely in putting thermal investigations in the first place and the microscope in the second place in the scientific study of most metals and alloys. We wish the work the acceptance and success which it richly deserves.

* * *

Lo Zinco. By Prof. **Roberto Musu-Boy.** 16mo., 230 pages, 10 illustrations, 4 diagrams. Price, 3.50 lire. (Retail price in New York, $1.25.) Milan: Ulrico Hoepli.

A well written, neatly printed, cheap book, which will undoubtedly advance the knowledge of the metallurgy of zinc in Italy. Parts I and II deal with the properties and metallurgical extraction of zinc, respectively; part III with the zinc mines of different countries; part IV with the uses of zinc and its alloys; part V with economic conditions in the zinc industry, and an appendix with the manufacture of zinc white.

While there are many shortcomings in the presentation, as far as concerns practice in other countries, there is much to be learned from the work, particularly concerning the zinc industry in Italy. We find, for instance, that Italy stands second to Germany among European countries in the value of zinc ore production, but behind six others in value of metallic zinc produced. This is because of the lack of cheap fuel and points the way to a large zinc industry as soon as the electric zinc furnace is through its "Kinderkrankheiten." About all that is said on this subject is that the furnaces of Casoretti and Bertani, working some years ago near Bergano, have a production of 3.25 tons of zinc per hp-year, using 250 kg of carbon for reduction per ton of ore and extracting 90 per cent to 92 per cent of the zinc. The author should have given considerably more space to the proposed electric zinc furnaces, because of their great prospective importance to his country.

* * *

The Wellcome Photographic Exposure Record and Diary, 1910. In cloth, 272 pages. New York: Burroughs, Wellcome & Company.

The 1910 edition of this photographic almanac contains a large amount of useful information for amateur photographers with respect to details of methods of development and exposure, the Wellcome exposure calculator chart being included. The almanac is exceedingly handsome

Metallurgical and Chemical Engineering

A Consolidation of Electrochemical and Metallurgical Industry and Iron and Steel Magazine

Vol. VIII. New York, February, 1910 No. 2

PUBLISHED MONTHLY BY THE

Electrochemical Publishing Company

239 West 39th Street, New York

Pittsburgh Office, 413 Lewis Block.

European Office, Hastings House, Norfolk St., Strand, London, Eng.

J. W. Richards, Ph.D.....*President*

E. F. Roeber, Ph.D.....*Editor and Secretary*

C. E. Whittlesey.....*Treasurer*

Yearly subscription price for United States, Mexico and United States dependencies, $2.00; *for all other countries,* $2.50 (*European exchange,* 10 *shillings,* 10 *marks,* 12.50 *francs*).

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Entered as Second-Class Matter at the Post Office at New York, N. Y., under the Act of Congress, March 3, 1879

New York, February, 1910.

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The Reaction Between Silica and Carbon

We heartily join Dr. Acheson's many friends and admirers in extending to him our congratulations on the award of the Perkin medal, and we congratulate with equal sincerity the Perkin Medal Committee on its choice. The apparently simple reaction between silica and carbon was the starting point of Acheson's life work which has since extended and included all the different reaction products between silicon, carbon, and oxygen. Besides the three elements we have now silicon carbide or carborundum, silicon monoxide (Dr. Potter's monox) and silica, carbon monoxide and dioxide, and the somewhat indefinite series of compounds which Mr. FitzGerald calls silico-carbides and denotes by the formula SiC_xO_y. The temperature and the other conditions of the experiment decide what the end result of the reaction will be. While from a purely commercial and industrial standpoint these products have been brought into a very satisfactory shape, it would seem that much remains to be done by physical chemists from a scientific standpoint—above all the establishment of the possible equilibria. When this is done, American physical chemists could pay their debt to Dr. Acheson by devising an Acheson diagram for the silicon-carbon-oxygen system analogous to the Roozeboom diagram for steel.

The Effect of Heat Leakage on the Electrode Losses

In commenting on Dr. Forssell's paper in our last issue, we pointed out that the phenomenon of electrode losses in furnaces may be conveniently considered as a superposition of three phenomena. These are, first, the simple heat conduction due to the temperature difference at the two electrode ends as though no current was flowing; secondly, the Joulean heat set free in the electrode, which flows off impartially one-half into the furnace and one-half into the water-jacket on the cold electrode end; and thirdly, the heat leakage from the electrode into the surrounding furnace walls, which may be considered as being sucked impartially one-half from the furnace and one-half from the water-jacket. It may be interesting to consider how the simple formulæ which hold good if the leakage from the electrode into the walls is neglected, must be changed to take account of the heat leakage. We will use the notation of Mr. Hering's fundamental article in the October issue of last year (Vol. VII, page 442). To represent the heat leakage into the wall we denote by c the circumference of the cross-section of the electrode in inches, and by w the wall leakage coefficient, defined as the mean value of the heat in watts given off per second per square inch of electrode surface to the surrounding walls. Since L is the length of the electrode, cL is its lateral surface and wcL represents the total heat leakage into the walls or the surrounding medium.

* * *

One fact is at once evident. While both the Joulean heat and the simple heat conduction due to temperature difference

depend only on the ratio L/S of length to cross-section of the electrode, but not on the length or the cross-section separately, this is not true of the heat leakage into the wall, since this depends on the circumference (but not at all on the area) of the cross-section. It would, therefore, now be meaningless to try to find that value of the ratio of length to cross-section, for which the electrode loss would be a minimum, or that value of the ratio for which no heat passes through the hot electrode end in either direction (the two temperatures and the current being given). However, if the cross-section and its circumference are given besides the temperatures and the current, we can investigate in what way the electrode loss and its accompanying phenomena change with varying length of the electrode. The heat which passes from the furnace into the hot electrode end is $jkTS/L + \frac{1}{2}wcL - \frac{1}{2}C^2rL/S$. This becomes zero if $L = S\sqrt{2jkT/(C^2r - wcS)}$. The difference from Mr. Hering's equation is the term $-wcS$, which is, of course, zero, if the heat leakage into the walls is neglected. Further, the total electrode loss (which is defined as the sum of the heat given off from the cold electrode end into the waterjacket plus the heat leaking from the electrode laterally into the furnace walls) is $jkTS/L + \frac{1}{2}wcL + \frac{1}{2}C^2rL/S$, and this becomes a minimum for $L = S\sqrt{2jkT/(C^2r + wcS)}$.

* * *

As long as the leakage of heat from the electrode into the furnace walls is neglected, the condition under which the electrode loss becomes a minimum is identical with the condition of temperature equilibrium at the hot electrode end, no heat passing through this end in either direction. The above formulas show that this is no longer the case, if we take the heat leakage into the walls into account. In order to obtain temperature equilibrium at the hot electrode end, we must make the electrode longer or smaller in cross-section than would follow from the simple formulas in which the heat leakage into the walls is neglected. On the other hand, in order to make the electrode loss a minimum, we must make the electrode shorter or larger in cross-section than would follow from the simple formulas in which the heat leakage into the walls is neglected. But, after all, what we really should attempt to get is not minimum electrode loss, but rather maximum furnace efficiency. If the current, the resistance of the furnace charge, the cross-section and its circumference of the electrode, and the temperatures are given, then the condition of maximum furnace efficiency requires a greater length than the condition of minimum loss. Only when the resistance of the electrode becomes negligibly small compared with the resistance of the furnace charge, the conditions of maximum furnace efficiency and minimum electrode loss become identical.

* * *

For numerical applications, knowledge of the wall leakage coefficient w is all important, while practically no data is available. Nevertheless, we can draw in a general way the following conclusions. If we would base the calculation of the length of the electrode (cross-section, circumference, current, and temperatures being given) on the condition of temperature equilibrium at the hot electrode end, we get an upper limit for the length; since this upper limit we should not go, because we would then produce a temperature maximum somewhere within the electrode, and flowing off from this point of maximum temperature in both directions; there would then be great danger of destruction of the furnace walls by this excessive temperature. Nor would it be wise to select exactly this length for the electrode, because any accidental rise of the current above its normal value during operation would cause the same trouble. On the other hand, if we would base the calculation of the length of the electrode on the condition of minimum electrode loss, we get a lower limit for the length, below which we should not go because we would then only increase the loss without any compensating advantages. Nor should we select exactly this lower limit of electrode length because by increasing the length above this value (by a proper amount) we raise the efficiency of the furnace as a whole. If we remain between these few limits, we cannot be absolutely wrong, and it is at least comforting to know that the value of length found from the simple formulas in which the leakage of heat into the walls is neglected, lies between the upper and lower limits just discussed.

An Interesting Blast Furnace Experiment.

Mr. Gayley's famous large-scale experiment on the use of dry blast has opened a new era in blast-furnace operation. An even more revolutionary experiment is soon to be tried in a Belgian blast-furnace plant, with not only a dry blast, but with practically an oxygen blast (as noticed in an article on new uses of liquid air and oxygen, published in this issue). Such a process has been strongly advocated in this country by Mr. J. E. Johnson, Jr. While we do not know how far Mr. Johnson's process, as described and illustrated in our Vol. VII, page 38, will be followed in the Belgian works, it will be apropos to recall here briefly his chief arguments in favor of an oxygen blast. The principal gain would be the possibility of doing away with the heating stoves for the blast and with many of the accessories. Mr. Johnson further proposes to feed the carbon and the iron ore, not together, but in separate columns. The intention is to oxidize the carbon, by the oxygen blast, to carbon monoxide and to force the carbon monoxide through the iron ore, resulting in the reduction of metallic iron and the formation of carbon dioxide. The latter passes off through the ore column and preheats it, but it does not come again in contact with free carbon. In this way more of the carbon can be oxidized to dioxide than in ordinary blast furnace practice, and a better utilization of the calorific value of the coal can be obtained within the furnace itself. It will be interesting to watch the experiment in Belgium. In general, it is safe to say that any further reduction in the cost of producing oxygen gas will create new applications in industrial chemistry and metallurgy and will increase the demand. As an example of the interaction between different fields of activities it is peculiar that the chief impetus for the larger use of oxygen—our principal heat producing element—should have come from the development of low temperature (liquid air) apparatus.

A Question of Business Policy.

Managers of very large enterprises have different questions of business policy to solve which smaller firms are exempt from. When a firm or trust becomes so large or powerful

that it dominates the market supply of a commodity, the question of "fixing prices" arises; in other words, it has become master to a large extent of the law of supply and demand, in that one line. Such domination is usually based upon control of the main sources of supply and of the bulk of the machinery of distribution, in addition to the manufacturing features which are the original scope of the business. Given a firm in such a commanding position with respect to a commodity which the public needs and must buy in large quantities, be it beef or wheat, steel rails or coal oil, what are the principles which should actuate and determine the fixing of prices? Are they to be fixed at the utmost extent of the public's ability to pay, or at the reasonable limit of the company's ability to furnish?

* * *

Our business friends are familiar with the term "good-will." It probably existed on earth as a business asset centuries before "the angels sang it in." Certain it is that it is as highly valuable an asset in the balance sheet now as it ever was. Can trusts or monopolies afford to dispense with it? To come down to the concrete facts, can any firm or corporation afford to squeeze its customers to the last limit of their endurance, and thus forfeit their good-will? If an extra demand arises which causes a temporary shortage, is a corporation justified in temporarily putting up its prices, although its own manufacturing costs have not increased?

* * *

Many modern business men—the "public-be-damned" type—will answer these questions without hesitation by the simple affirmation "business is business." But, business is not business, as they mean it; it is horse-sense and common-sense and good sense applied to commercial life. The doctor who usually charges a certain sum for a certain service, and who doubles his fee because he was wanted more urgently, is not a sensible man, he makes enemies instead of friends. The shopkeeper who would charge me 10 cents for a 5-cent loaf of bread because I am extra hungry, is unprincipled, and gets no more of my future custom than I can help giving. The monopoly which "fixes prices" according to the demands of the market and not according to its cost price plus a reasonable profit, is working upon equally false and unfortunate ethical and business principles. By so doing, it is forfeiting the good-will of its customers, it invites competition, is provokes hostile legislation and stringent regulation; in other words, it antagonizes the general public good, and no business can permanently succeed upon such a foundation. A more secure business, a more permanently profitable business, a more honorable business can be built by cultivating the good-will of the public, by a "public-be-pleased" policy and by avoiding exploitation of its necessities. Short-sighted, self-willed, hard-fisted, penny-wise-and-pound-foolish some business men are, and they do not see the better way, the more honorable way—and in the end the really more profitable way, which best business sense and intuition sees and acts upon.

Two Recent Technical Advances in Steel Manufacture.

It has been said by cynical observers that patent specifications are often worded in a way so as to conceal rather than reveal their exact meaning. Nevertheless a patent, to hold good, must state the invention, and in spite of all attempts of hiding, and in spite of the many foolish things for which patents are granted, it remains true that the current patent literature contains most interesting live matter. Two quite recent instances with respect to steel making and refining are, a patent of Mr. E. Humbert for removal of phosphorus in the electric furnace, abstracted on page 48 of our January issue, and two patents of Mr. R. B. Carnahan, Jr., abstracted under Recent Metallurgical and Chemical Patents in this issue, for the production of a non-corrosive dead-soft steel of remarkable purity in the open-hearth. Mr. Humbert's patent is interesting in view of the recent statement of a very distinguished steel metallurgist that the electric furnace could do nothing new with phosphorus. It shows again how dangerous it is to argue *a priori* in such matters. To explain the new method of phosphorus removal, we may point out a certain parallelism with the method of removing the sulphur.

* * *

Sulphur in steel is chemically combined with iron and manganese. In the basic converter and the basic open-hearth these sulphur compounds will pass to a certain extent into the basic slag; and their absorption by the slag will be the greater, the higher the temperature of the slag and its basicity. But this absorption cannot be complete, as it must stop when equilibrium has been reached between the steel bath and the slag, because the sulphur compounds are soluble in both of them. The essentially new reaction which permits a practically complete removal of the sulphur in the electric furnace is the removal of the sulphur in form of calcium sulphide which is soluble only in the slag, but not in the steel bath. This seems at least the ultimate reaction in the electric furnace and because the calcium sulphide is insoluble in the steel bath, it is possible to drive the removal of the sulphur to any desired extent. If this reaction should occur in an open-hearth furnace, it could certainly not be the ultimate reaction on account of the oxidizing atmosphere which would change the calcium sulphide quickly into sulphate. This would react with the iron, causing the formation of iron sulphide which then passes back into the steel bath. It is, therefore, essentially on account of the reducing atmosphere of the electric furnace as compared with the oxidizing atmosphere of the Bessemer converter or open-hearth, that reactions eliminating the sulphur to any desired degree are possible in the electric furnace and impossible in the old metallurgical furnaces.

* * *

The removal of phosphorus in ordinary metallurgical furnaces with an oxidizing atmosphere is in the form of a phosphate. Mr. Humbert, by putting ground coke on the slag, changes the phosphate to phosphide, which is, of course, possible only on account of the reducing atmosphere of the electric furnace. It is impossible to do the same in an oxidizing furnace. The method is very simple, and, from what we have heard about its working in practice, it permits a very exact control of the phosphorus and results in a reduction in time, energy consumption, and loss of steel. The chief advantage seems to be that it is no longer necessary to scrape off the slag; this avoids much trouble and is important also for the construction of furnaces of large capacity. The electric furnace process is thus being worked out successfully in all its details and the chief problem to be solved in future seems to be to divide the whole

process of steel making into such a combination of different steps in different apparatus, that each reaction is carried out in the most suitable apparatus (according to whether it requires an oxidizing or reducing atmosphere, etc.) and that the practical efficiency of the whole combination process is a maximum.

* * *

The patent of Mr. Carnahan for making a dead-soft, open-hearth steel, of remarkable purity, as described under Recent Metallurgical and Chemical Patents, is interesting both with respect to its method of manufacture and its properties. There is absolutely nothing electrical about the method of manufacture as it is completely an open-hearth process, but in its first part—the reduction of impurities to the desired very low degree by continuing the process much longer than is usual in ordinary open-hearth practice and thereby overoxidizing the metal—the process is just like modern practice in preparing steel in the converter or open-hearth, preparatory for subsequent refining in the electric furnace. The really remarkable feature of the process of Mr. Carnahan is that what he does with this overoxidized steel appears to be simply to reduce the oxides by ferrosilicon, and to remove the gases by addition of aluminium in the mold. It is true that his steel when it leaves the furnace is very much hotter than in ordinary open-hearth practice and this may help in the accomplishment of the ends. It is a fact that when attempts have formerly been made to refine steel thoroughly by overoxidation, failure has always resulted from the difficulties of treating the overoxidized steel. These difficulties were successfully overcome by electric furnace treatment, but Mr Carnahan appears to have succeeded without such treatment and it is only to be regretted that the patent says so little about this most interesting feature of the process. The steel is on the market as American ingot iron and is used especially for sheets where its non-corrosive properties are of importance. The process seems to be rather expensive and it is a question whether it would not be cheaper to use a combination of converter and electric furnace processes since there seems no reason why it should not be possible to produce a steel of the same composition by such a combination.

* * *

The claim that this dead-soft steel is non corrodible appears to have been borne out in practice as far as the comparatively short time for which this steel has been in commercial use permits to draw conclusions. If the non corrodibility of this steel is further confirmed by experience, it will be a strong point in favor of the electrolytic theory of corrosion. It will be remembered that according to this theory, as proposed by Dr Whitney and elaborated and expanded by the extensive work of Dr Cushman and Dr Walker the intrinsic cause of corrosion must be found in the non-homogeneity of steel; this is probable to a large extent to the use of ferro manganese; on account of this lack of homogeneity there is a liability of some places in the steel surface becoming anodes and other places becoming cathodes in contact with moisture which acts as electrolyte, thus we have short circuited galvanic couples and corrosion results. Evidently, if this theory is correct then an absolutely homogeneous steel, free from impurities, slag inclusions etc. would be non-corrodible, as has been repeatedly pointed out in these columns. The makers of "American ingot iron" work on this principle; they do not use ferro manganese and in general endeavor to produce as pure a steel as possible. Let us hope that further experience with this steel will confirm the first good results obtained with it. It would be an indirect practical large-scale confirmation of the electrolytic theory of corrosion, and what is more important, the essential question would be definitely settled as to the direction in which we must work to find a practical solution of the corrosion troubles.

The Use of Pulsating Current for Electrolytic Refining.

It has long been looked upon as an axiom that direct current must be used for electrolytic processes. According to Faraday's law, the weight of the desired electrolytic reaction product is proportional to the coulombs passing through the cell in the proper direction. Theoretically, considering the reaction as perfectly reversible, we may conclude that by reversing the current and passing now the same number of coulombs through the cell in the opposite direction, the electrolytic reaction product of our former process is completely undone, so that at the end the system is in exactly the same condition as at the beginning. This will be true in practice, if the reaction products of the current passing in one direction remain within the zone of electrolytic action (for instance, do not pass off as gases out of the cell, etc.), and this condition will be fulfilled the more exactly the quicker the reversals of the current follow each other or, in other words, the shorter the period or the higher the frequency of the alternating current employed. In most cases of practice it is perfectly true to say that alternating current produces no electrolytic action.

* * *

The ingenious modification of the Wohlwill process of electrolytic refining of gold and gold-silver alloys, described in an article elsewhere in this issue, is exceedingly interesting, not only because it broadens very considerably the scope of the process, but because the very simple means of accomplishing this is the use of a pulsating current—a superposition of direct current and alternating current—instead of simple direct current. Everything we said above on the necessity of direct current for electrolytic action remains true in this case. The total refined gold which is obtained by deposition on the cathode equals exactly the amount which would be deposited if only the direct current were present. With respect to the output of refined gold the alternating current could just as well be absent. Why, then, is it used? Simply to break up the silver chloride layer formed at the anode, so as to avoid the necessity of mechanically scraping it off. The direct-current component of the pulsating current determines the gold deposition, the alternating-current component keeps the anode clean. In view of its remarkable practical advantages this process deserves attention far beyond the sphere of gold refining.

Lake Superior Iron Ore Values.

It would be interesting, although probably not particularly useful, to know what would have been the course of values if our Lake Superior iron ore deposits had been discovered in a country, like England, with an old established iron industry not subject to rapid growth on account of increase in demand. One

may well conclude that the history of prices would have been quite different from that which has been written. It was not until 1869 that pig-iron production in England passed the 5,000,000-ton mark, and not until 1906, 37 years later, that it passed the 10,000,000-ton mark. In the United States pig-iron production has doubled, on an average, just a trifle more frequently than once in 10 years. It is, of course, quite an elementary proposition that an increased demand for pig iron involves an increased demand for iron ore, and that the more rapidly the demand for pig iron increases, the more rapidly the demand for iron ore increases. There is a very marked difference, however, in the character of the demand for ore according to whether the demand for iron increases rapidly or slowly. It requires time to exploit ore deposits and develop mines to commercial production. If the increase in demand is slow, there is plenty of time to look around and develop the ore production. If the increase in demand is rapid, the time required for development becomes an important element, both because the ore is needed quickly and because, usually, rapidly expanding demand means very profitable prices, so that the ore buyer can afford to pay a premium for the quickness.

* * *

When the Mesabi range was discovered it appeared that an inexhaustible supply had been discovered, and values, as determined by the four older of the Lake Superior ranges, fell rapidly. The first Mesabi ore was shipped in 1892, contributing 4245 tons to a total output of the region of 9,082,370 tons. By the close of 1895 the range had passed each of the four older ranges in point of production, and in 1893 it shipped more ore than the other four ranges combined. For six years after values had become definitely established, or for the seasons 1894-99 inclusive, Mesabi non-Bessemer ore sold at an average price of considerably less than $2 a ton on Lake Erie docks, or at less than 50 cents at mine. Undoubtedly the industrial depression from 1893 to 1898, inclusive, had something to do with the low prices, but more in the direction of prolonging the period of low prices than in producing them. Before knowledge of the range was five years old it was certain that it contained enough ore to cover the country's entire pig-iron production, at the then obtaining rate, for 30 or 40 years. The first Mesabi ore was gained by doing little more than removing the sod. Had demand for ore increased only slowly the ore might have remained a drug on the market, but the rapid expansion in demand has made it so that stripping operations involving the removal of 75 ft. or 100 ft. of overburden are in progress.

* * *

While the greater portion of the Lake Superior ore deposits have passed to the control of consumers, there remains a large annual production by merchant interests which is sold in the open market, chiefly to merchant blast furnaces. Prices are fixed for each shipping season and do not change during the season. Ordinarily the great bulk of the season ore is sold in a few days or a few weeks. The relation between the price of ore and the prices of pig iron has undergone some remarkable changes even in the past few years, or since the general trade position of the Lake Superior ores became well established. The comparison given below is calculated to illustrate these changes. There is set down, first, the price of Mesabi non-Bessemer ore for seven seasons. Next is set down the prevailing price of No. 2 foundry pig iron, f.o.b. valley furnaces, at the time the principal ore selling was done. Next is given the average selling price of the pig iron during the approximate period of ore consumption. The year named is the year in which the ore was, or will be, shipped, the price having been named either in the early part of that year or in November or December of the preceding year, while the period of average pig-iron prices runs from May 1 of the year to May 1 of the following year.

MESABI NON-BESSEMER ORE AT LAKE ERIE DOCK, AND NO. 2 FOUNDRY PIG IRON AT VALLEY FURNACES.

Season.	Ore.	Pig iron at time.	Pig iron average.
1904	$2.50	$13.00	$13.93
1905	3.00	16.00	15.95
1906	3.50	17.25	20.28
1907	4.00	21.00	19.41
1908	3.50	14.50	14.78
1909	3.50	14.25	14.89
1910	4.00	17.00	.....

* * *

The pig-iron market has, as a rule, reflected the general state of trade rather than the cost of ore. Thus, the price of ore for the season of 1906 (which was fixed in December, 1905) was upon a market of $17.25 for pig iron, but the market advanced and from May 1 following for 12 months averaged $3 a ton higher. For the following season ore prices were fixed on a $21 pig-iron market; the market subsequently advanced to $25, in May, and declined to $15 in the following April. The actual realized price on the pig iron made from the 1907 ore was much less than is shown by the $19.41 average of quotations, for the reason that the great bulk of the pig iron was sold for forward delivery before the extreme advance occurred, the actual realized price probably being well below $19. As the price of ore is made at Lake Erie dock and the cost of transport has varied but little, the fluctuations in the price of ore at mine have been almost as large, absolutely, as the fluctuations at the point of delivery, while they have been much larger in point of percentage. What is particularly striking about the presentation given is the fact that relative to pig-iron, ore is much higher than ever before. In 1905 a $3 price was made upon $16 pig iron, but in 1908 a $3.50 price was made on $14.50 pig iron, and a year later the price was reaffirmed with pig iron 25 cents lower, while in 1906 a $3.50 price was made upon $17.25 pig iron. Again, the price of $4 was made only for 1907 and 1910, but in 1907 it came upon $21 pig iron, while for 1910 it came upon $17 pig iron. Thus with the great changes which have occurred in both ore and pig-iron prices, one trend remains distinct throughout, a higher and higher price for ore relative to the price of pig iron. The asset is the ore mine and not the blast furnace. Value depends upon the cost of replacement or duplication, provided there is need. One may readily conclude that, had there not been the rapid expansion in demand which has been experienced, values of Lake Superior ores would have remained upon a relatively low basis. As said before, for a period of six years in the nineties Mesabi non-Bessemer ore brought less than 50 cents per ton at mines. The present $4 price represents $2.50 at mines, since the rail freight from mine to Duluth is 80 cents and the lake freight through Lake Erie dock is 70 cents.

Nobel Prize for Chemistry.

The Nobel prize for chemistry has this year been awarded to Wilhelm Ostwald, of Germany. A full sketch of the life and work of Dr Ostwald has appeared in these columns (Vol. I, page 253). Since the publication of this article Dr. Ostwald has resigned his professorship in the University of Leipzig and has also devoted a good deal of his attention to the foundation of a new system of natural philosophy based upon energy as the fundamental principle. The result has been a remarkable volume of lectures on natural philosophy. Dr. Ostwald has also just published his book, "Grosse Manner," in which he attempts to analyze the elements of scientific greatness.

Lately, however, Dr. Ostwald has again favored the chemical world with several volumes on general chemistry, chiefly emphasizing the logical evolution of chemical science and scrutinizing its philosophical foundations.

Dr. Ostwald's many friends and former students on this side of the Atlantic have heard, with great gratification, of this last honor bestowed upon him.

Chemists' Club Plans Filed.

Plans have been filed with the New York City Building Department for the 10-story clubhouse and office building to be erected for the Chemists' Club at 50-54 East Forty-first Street. It will be an artistic edifice, of white marble in French Renaissance style, of the Louis XVI period, finished with Ionic pilasters and balconies at the second story and similar decorative balconies at the top story.

The main floor will contain in the rear a large auditorium with a balcony, and the second will be fitted as a lunchroom, with social rooms opening off from it. The fourth and fifth floors will be devoted to living and sleeping rooms for the members, below which will be the library and museum. The five upper stories will contain laboratories. The building is to cost $200,000, according to the estimate of York and Sawyer, architects.

American Chemical Society.

The annual meeting of the American Chemical Society was held in Boston during the last week of December. Over three hundred members attended. As Dr. Willis R. Whitney had declined re election to the presidency of the society, Dr. Wilder D Bancroft, of Cornell University and editor of the *Journal of Physical Chemistry*, was chosen to succeed him in that office. Dr Albert P. Hallock was re-elected treasurer and Professor Charles L. Parsons was re-elected secretary. Mr. Arthur D. Little, of Boston, Mass., Dr Leo H. Baekeland, of Yonkers, N. Y., and Professor W. L. Dudley, of Vanderbilt University, were elected councillors-at-large.

The meeting was a great success from a scientific and professional standpoint. The papers and discussions were of a high order, they will all be published in full in the different publications of the Society. Dr. Whitney's presidential address on "Some Chemistry of Artificial Light" proved particularly interesting and suggestive

The social functions of the convention were exceedingly enjoyable. Both the dinner and the smoker were well attended At the smoker Prof. Samuel C. Prescott was the soul of Section Q. Simply to indicate the good feeling that prevailed throughout, we quote here the speeches with which seven unsuspecting scientists were presented with "honorary degrees" at the smoker

"Harvey Washington Wiley—Doctor of philosophy, doctor of laws, doctor of [illegible] Thereof he [illegible] of the congress to be [illegible] on banquets, growth of [illegible] your name [illegible] your [illegible] your [illegible] Pure Food Products' and under [illegible] you, sir, the degree of LL.D., Doctor of Legal Lunch.

"Alfred D. Elbert—Physicist, statesman, man of affairs, efficient engineer, we appreciate your [illegible] as chairman of the committee of arrangements to make our brothers from other States remember the Hub. We bestow upon you the degree of C.O.D., Doctor of Chemical Organization.

"Charles Lathrop Parsons—Bachelor of science, inorganic chemist of note, famed for his exposition of the vagaries of Berrylium, consumed by a gnawing lust for numbers you have made our society the biggest in the world, we bestow upon you the degree which you so gracefully ornament, H.H.S., Hustling, Hornblowing Secretary.

"Wilder Dwight Bancroft—Bachelor of arts, author and editor. You range the field of sciences from divariant systems to cigars, from brain storms to brass. Your famous interpretation of Gibbs is so profound that Gibbs himself could not follow your analysis. You know you are great because we cannot understand you. We bestow upon you the degree of M.S.S., Many Sided Scientist.

"Marston Taylor Bogert—Master of science, doctor of laws, the mighty wielder of unconserved language, torturer of innocent atoms. You are a past master in forcing them to roll over, sit up and beg. I confer upon you the degree of L.Q.D., Doctor of Complicated Queeralogies.

"Louis Kahlenberg—Bachelor of science, author, enemy of fanciful imaginings. We admire you as a valiant fighter in a lost cause. I bestow upon you the degree of R.I.P., Promulgator of Revolutionary Ideas.

"Willis Rodney Whitney—Bachelor of science, doctor of philosophy, leader of America's chemists, trained in pure science, an expert in its application. I confer upon you the degree of E.M.F., Every Man's Friend."

American Association for the Advancement of Science.

Dr. A. A. Michelson, professor of physics in the University of Chicago, has been elected president of the American Association for the Advancement of Science, for the meeting to be held the end of the year at Minneapolis, Minn. Dr. L. O. Howard, of Washington, D. C., is permanent secretary of the Association; Professor Frederic E. Clements, of the University of Minnesota, was chosen general secretary, and Professor John E. Zeleny, of the University of Minnesota, secretary of the council. Vice-presidents of the sections have been elected as follows:

Section A—Mathematics and Astronomy—Professor E. H. Moore, University of Chicago; *Section B—Physics*—Dr. E. B. Rosa, Bureau of Standards, Washington, D. C.; *Section C—Chemistry*—Professor G. B. Frankforter, University of Minnesota; *Section D—Mechanical Science and Engineering*—Professor A. L. Rotch, Blue Hill Meteorological Observatory; *Section E—Geology and Geography*—Dr. John M. Clarke, New York State Geologist, Albany, N. Y.; *Section F—Zoology*—Professor Jacob Reighard, University of Michigan; *Section G—Botany*—Professor R. A. Harper, University of Wisconsin; *Section H—Anthropology and Psychology*—Professor Roland B. Dixon, Harvard University; *Section I—Social and Economic Science*—Hon. T. E. Burton, Cleveland, Ohio; *Section K—Physiology and Experimental Medicine*—Professor F. G. Novy, University of Michigan; *Section L—Education*—President A. Ross Hill, University of Missouri.

Mining and Metallurgical Congress

The International Congress of Mining, Metallurgy, Applied Mechanics and Practical Geology, of which a preliminary notice occurred in our issue of July, 1909 (Vol. VII, page 398), will take place at Dusseldorf, Germany, during the last week of June, 1910. The congress will be divided into four sections 1, Mining; 2, Metallurgy; 3, Applied Mechanics; 4, Practical Geology.

The presidents of the congress are Bergrat Kleine, president of the "Verein für die Bergbaulichen Interessen im Oberbergamtsbezirk Dortmund," and Kommerzienrat Springorum, director general of the Hoesch Iron & Steel Works, and president of the "Verein deutscher Eisenhüttenleute." The general secretaries are Dr Schrödter, secretary of the "Verein Deutscher Eisenhüttenleute", and Bergassessor von und zu Löwenstein, secretary of the "Verein für die Bergbaulichen Interessen im Oberbergamtsbezirk Dortmund."

There will be two grades of membership in the congress The first class of members are those who have become patrons of the congress through payment of a contribution of not less than 100 marks ($25). The second class of members is composed of those who pay a subscription of 20 marks ($5). The first-named class of members or patrons will be entitled to receive the printed reports of all proceedings of the congress in all different sections. The members of the second class, or ordinary members, will receive the reports only of that section in which they register. The proceedings of any one of the other sections will be obtainable by payment of an additional charge of 5 marks ($1.25) in each case.

The work of the congress will be performed in general meetings, at which papers of general interest will be presented, and

sectional meetings for the purpose of discussing important problems relating to more detailed fields, and by making visits to scientific and industrial institutions of interest and by excursions to districts of geological interest.

The program of Section II, that of metallurgy, is as follows:

A.—PRODUCTION OF PIG IRON.

1. Cooking.
   (a) Ovens.
   (b) Mechanical appliances.
   (c) Recovery of by-products.
2. Ore supply.
   (a) Recent discoveries of ore.
   (b) Recent development and prospects of the ore briquetting processes.
3. Metallurgy of the blast-furnace process.
   (a) Influence of foreign substances.
   (b) Composition of slag.
4. Blast furnace working.
   (a) Ore conveyance, storage and charging.
   (b) Gas washing and purification of the waste water.
   (c) Dry air-blast.
   (d) Casting machines and mixers.
5. Utilization of waste products.
   (a) Gases.
   (b) Dust from blast-furnace gases.
   (c) Slag (for hydraulic packing, cement, stone, concrete).

B.—PRODUCTION OF MALLEABLE IRON.

1. Advances in the methods of the metallurgical treatment of iron and steel.
   (a) Air blast refining process.
   (b) Open-hearth refining processes.
   (c) Processes for the production of electro-steel.
2. Production and treatment of special alloys of steel.

C.—IRON AND STEEL MANUFACTURE.

1. Improvements in the methods of casting iron and steel.
2. Further treatment of malleable iron.
   (a) Forging and pressing.
   (b) Rolling.
   (c) Fitting.
   (d) Development of the welding processes.
3. Driving of rolling mills considered technically and economically (steam, gas, electricity).

D.—TESTING OF IRON AND OTHER METALS.

1. Chemical testing.
2. Mechanical testing.
3. Metallography and microscopy of metals.

E.—ECONOMICS OF THE IRON INDUSTRY.

1. Iron trade statistics.
2. Labor conditions and labor supply.
3. Patent rights.

All inquiries for information regarding the sections or membership in them may be addressed to the Arbeitsausschuss des Internationalen Kongresses (Committee of Organization of the International Congress) Jacobistrasse 3/5, Düsseldorf, Germany.

The Institute of Metals.

The annual general meeting of the Institute of Metals was held at the Institution of Mechanical Engineers, Storey's Gate, Westminster, London, S. W., on Tuesday and Wednesday, Jan. 18 and 19, 1910.

The program contained the following papers:

C. O. Bannister, Esq., Assoc. R.S.M., on "The Use of Carbonaceous Filters in the Smelting of Zinc, as Employed in the Hopkins Fumeless Zinc Process."

G. D. Bengough, Esq., M.A., on "The Properties and Constitution of Copper Arsenic Alloys."

T. Vaughan Hughes, Esq., Assoc. R.S.M., F.I.C., on "The Failure in Practice of Non-Ferrous Metals and Alloys, with Particular Reference to Brass Loco-Tubes."

O. F. Hudson, Esq., M.Sc., and E. F. Law, Esq., Assoc. R.S.M., on "A Contribution to the Study of Phosphor Bronze."

C. A. Klein, Esq., on "Notes on a Suggested Record of Analyses."

Dr. Richard Seligman, Ph.nat.D., and F. J. Willott, Esq., on "The Analysis of Aluminium and Its Alloys."

Ernest A. Smith, Esq., Assoc. R.S.M., on "The Assay of Industrial Gold Alloys."

The first paper we print in full in this issue. Abstracts of all the other papers are included in the synopsis of Current Metallurgical and Chemical Literature in this issue.

The new Carnegie Physics Laboratory at University College, Dundee, Scotland, has been formally opened by Sir Joseph J: Thomson, of Cambridge University.

Resignation of Professor Chandler.

Professor Charles Frederick Chandler, head of the department of chemistry at Columbia University, has tendered his resignation, after forty-seven years spent in the service of that institution.

The trustees voted to accept his resignation with regret, and provided that during the next academic year he should have leave of absence, his connection with the university to expire at the end of that time. When taking action on this matter the trustees passed the following minute:

"In receiving the application of Charles F. Chandler, Ph. D., M. D., LL. D., Sc. D. (Oxon.), Mitchill Professor of Chemistry, to be relieved of active service at the close of the present academic year the trustees wish to place on record an expression of their sincere appreciation of the distinguished service which Professor Chandler has rendered to Columbia University and to the science of chemistry during his long and active career as teacher and investigator.

"Immediately upon receiving the degree of Doctor of Philosophy from the University of Göttingen in 1856, Professor Chandler entered upon his lifework of academic teaching. From 1857 until his appointment in 1864 to be Professor of Chemistry at Columbia, Professor Chandler was head of the chemical department of Union College. On his transfer to Columbia in 1864, he threw himself heartily and generously into the task of developing the newly organized School of Mines. In addition to the heavy duties of his professorship, Professor Chandler served for thirty-three years as Dean of the School of Mines, and to his effective and conscientious administration of its business that great school owes no small share of its usefulness and prestige. Besides bearing the burden of administrative and teaching duties, Professor Chandler found time for important public service during many years' incumbency of the presidency of the Health Department of the City of New York. He has also been an enthusiastic and prolific investigator of chemical problems, particularly those arising from the application of the science of chemistry to the public health and to the various forms of industrial activity. Professor Chandler is a member of the chief learned societies, at home and abroad, in which chemists are represented, and he has received well-merited recognition at the hands of his colleagues in chemistry in England, as well as in America, together with the highest honors of the University of Oxford. The chemical museum of the university, which has been built up by his efforts and in no small part by his generosity, is unique of its kind and will remain a monument to his untiring activity.

"Professor Chandler will carry with him into his retirement the affectionate regard and esteem of two generations of students, as well as of a host of colleagues on the teaching staff of the university.

"The trustees record their grateful appreciation of this long and generous career of devoted service."

During the majority of Professor Chandler's term of service at Columbia he was identified with the School of Mines, now the School of Applied Science, and was one of the three men whose efforts made its existence a possibility in 1864. At this time he was made dean of the school and he served in this capacity until the time of his resignation in 1897. While he was at its head, the School of Mines grew from a very small department of the university to one of its largest branches. Since his retirement from the office of dean, Professor Chandler has continued his work as professor of chemistry. He has taught up to the present time and will continue to do so, until the close of the present academic year.

Professor Chandler has an enviable reputation in the realm of chemistry. He was one of the leaders in the celebration of the centennial of the discovery of oxygen, which was held at the old home of Joseph Priestley in 1874. It was at this time that the American Chemical Society was organized. In 1899 the Society of Chemical Industry honored Professor Chandler

by electing him president, the first American chemist to receive such recognition.

That the School of Mines did not seem at first a very promising project is recalled in an article in the Columbia *Alumni News*. Three professors, however, were found who were willing to begin without salaries. George T. Strong, William E. Dodge, Jr., and several others furnished about $3,000 to equip the laboratory. A fine cabinet of minerals was donated by the Hon. Gouverneur Kemble, and President Barnard, Dr. Torrey, and other trustees of the college encouraged the enterprise in every possible way. Some vacant rooms in the basement of the college were assigned for laboratories. Provision was made for twelve students, but twenty-four came the first day. The success of the School of Mines was remarkable. During the entire winter the carpenters and gasfitters were constantly at work erecting new tables for additional applicants, and the number of pupils reached forty-eight.

During the first vacation a large four-story building was placed at the disposition of the school, with funds for the equipment of laboratories and cabinets. Accommodations were arranged for seventy-two pupils. During the second year the school was thronged; eighty-nine students were in attendance. The success of the new school seemed so well assured that the trustees arranged to place it on a substantial basis as a coordinate department of the college. Professor J. S. Newberry was called to the Chair of Geology, relieving Professor Chandler of this subject, and a full faculty of professors and assistants was established. A new building was erected, and equipped with laboratory accommodations for 150 pupils.

From this time on the school continued to prosper, and was from time to time expanded by the establishment of nine courses of professional study, until it became one of the most important schools of applied science in the country and acquired an international reputation. Professor Chandler was made dean of the faculty at the beginning, and was the executive officer of the School of Mines for thirty-three years. He resigned in 1897, when the university moved to its new site. During the long period of his connection with Columbia, Professor Chandler has lectured upon all branches of chemistry—theoretical, analytical, and applied. At the present time he lectures upon general and industrial chemistry, the other branches being taught by his colleagues and assistants. During the earlier years of the School of Mines the assay department was an especial hobby of Professor Chandler, and with the aid of his successive assistants he has made it the most complete of its kind to be found anywhere. To facilitate the work of assaying gold and silver ores and to insure accuracy in calculating the results, Professor Chandler devised in 1866 the system of assay weights now in general use by all assayers.

Upon the suggestion of Professor Chandler Havemeyer Hall was provided by the Havemeyer family of New York City. Preliminary to perfecting the plans of this remarkably complete institution, Professor Chandler visited and inspected every laboratory of consequence in Europe. The chemical museum, which is the personal creation of Dr. Chandler, is the most complete [illegible] of the kind in the world, and it is unique. It gives the visitor some idea of the extent to which the affairs of man depend upon the science of chemistry.

By the resignation of Professor Chandler a serious gap is made in the instructing staff of Columbia University. A successor, and an able one, will of course be provided, but it will be difficult to find a man who will be able to act in all the capacities that Professor Chandler has served in, those of personal adviser, executive head, chemical instructor, and loyal friend.

The Iron and Steel Market.

Industry has been a dull sample in the iron and steel trade, as was December. There has been the usual mill reaction in sentiment after a prolonged advance. Not since the great boom of 1899 had the industry experienced so important an advance in prices, and never in its history has it experienced so spectacular an increase in production, since from April to December fully 10,000,000 tons was added to the annual productive rate of pig iron.

The present reaction is purely one in sentiment, and by no means touches either production or prices. It is only human nature, perhaps, to be disappointed that further good news has not come out in the industry. The sweeping nature of the improvement in the latter half of last year had the double result of leaving no reasonable room for further improvement and of discouraging men from expecting any termination to it.

As to orders, there has been a lull, relatively speaking, for two months. These are months which are customarily quiet in the industry. No large projects are undertaken at this time and there was a particular incentive for buyers to crowd all possible business into the old year, on account of the trend of prices. There was much buying and specifying on contracts at terms which cannot be duplicated.

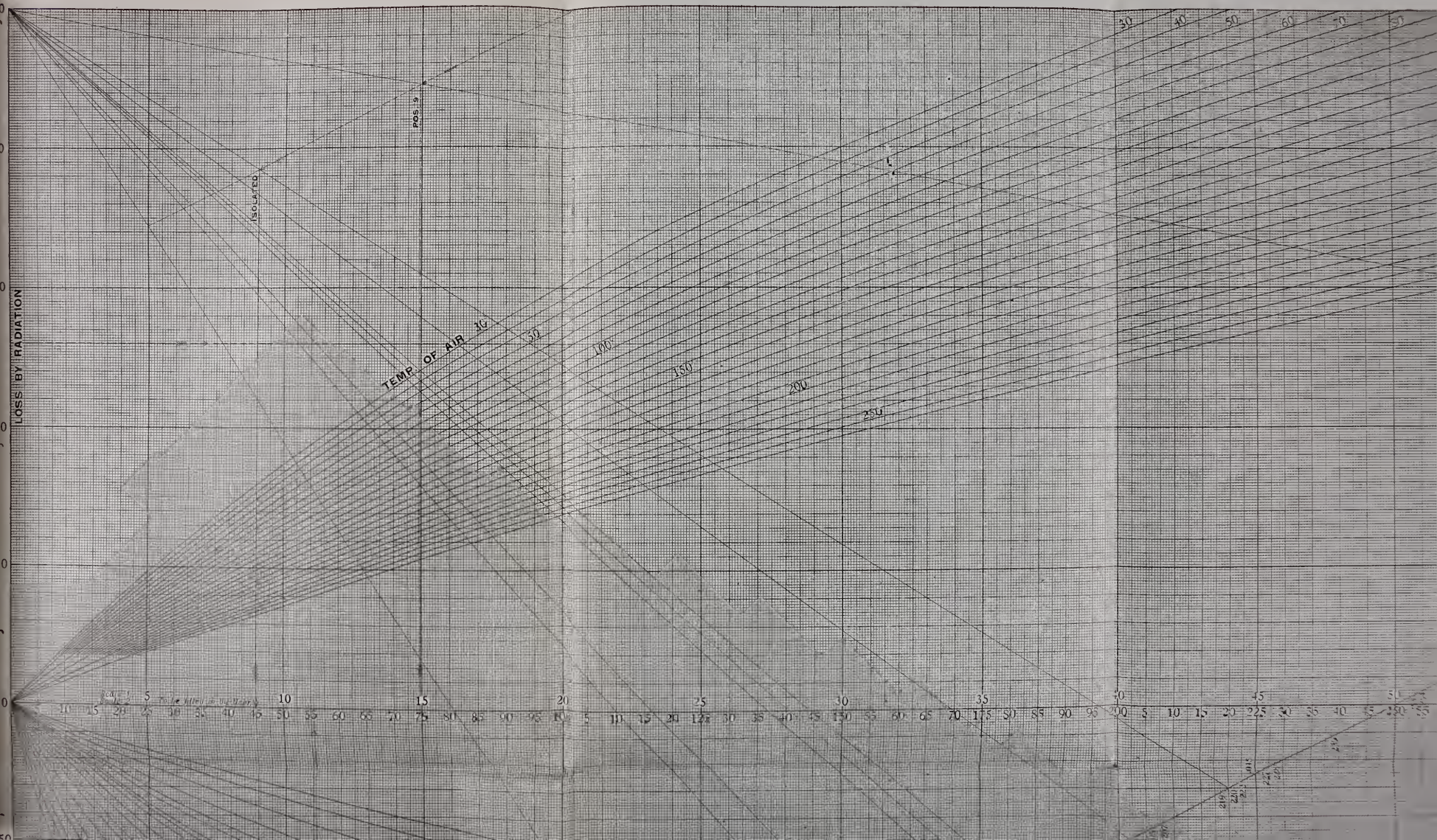
All signs point to the current year being one of minor fluctuations in demand, production and prices. There is no room for large swings in prices. The advance, while spectacular, was moderate, since it followed a break in prices extremely low considering the advances in costs of material and labor since the historic low prices of the middle nineties. There is no room for a great break in prices, and it can scarcely be thought there is room for a runaway market, considering the great increase in productive capacity. It strained the industry in 1907 to pass a pig iron production rate of 27,000,000 tons a year, but the present productive capacity is between 31,000,000 and 32,000,000 tons and new furnaces in course of erection will add 2,000,000 tons before the year is out.

It is not likely that there will be any great changes in total demand during the year. Production was brought up to a rate of between 31,000,000 and 32,000,000 tons of pig iron a year at the close of 1909 by a demand coming chiefly from the smaller lines of consumption, the demand based upon prosperity in the agricultural sections being most conspicuous. The consumption of steel in large projects which have to be financed was relatively light. The railroads really took very little iron and steel in 1909. While the year's production of pig iron closely approached that of 1907 and exceeded that of 1906, the railroads did not take, in cars, locomotives and rails, one-half as much material as they did either in 1906 or 1907.

It is fair to assume that a considerable portion of the demand of 1909 was for the purpose of replenishing greatly depleted stocks of material in the hands of jobbers and manufacturing consumers. This might, perhaps, point to a slight decrease in their requirements in the next few months. The industry could stand the reduction and still find itself with a much greater tonnage than at its best former rate, but at the present time the consumption of iron and steel for railroad purposes is very much greater than the average of 1907. The locomotive and car shops are running almost full, and the orders now on their books represent much more business than the actual output of the entire year 1909. Current reports are that the prospects of additional railroad buying are not good, but the business already placed guarantees a better tonnage for this year than there was last year.

Iron Ore.

Late in December Lake Superior ore interests fixed prices for the season of 1910 at an advance of 50 cents a ton over the schedule for 1908 and 1909, thus restoring the prices which were held for the single season 1908, these being $5 for old range Bessemer, $4.50 for old range non-Bessemer, $4.75 for Mesabi Bessemer and $4 for Mesabi non-Bessemer, all at lower lake ports, basis guaranteed 55 per cent iron (natural state) for Bessemer ores and 51.50 per cent for non-Bessemer ores. Consumers had already made large reservations of ore, particularly of Bessemer, pending the decision as to prices, and these reser-


LOSS BY RADIATION
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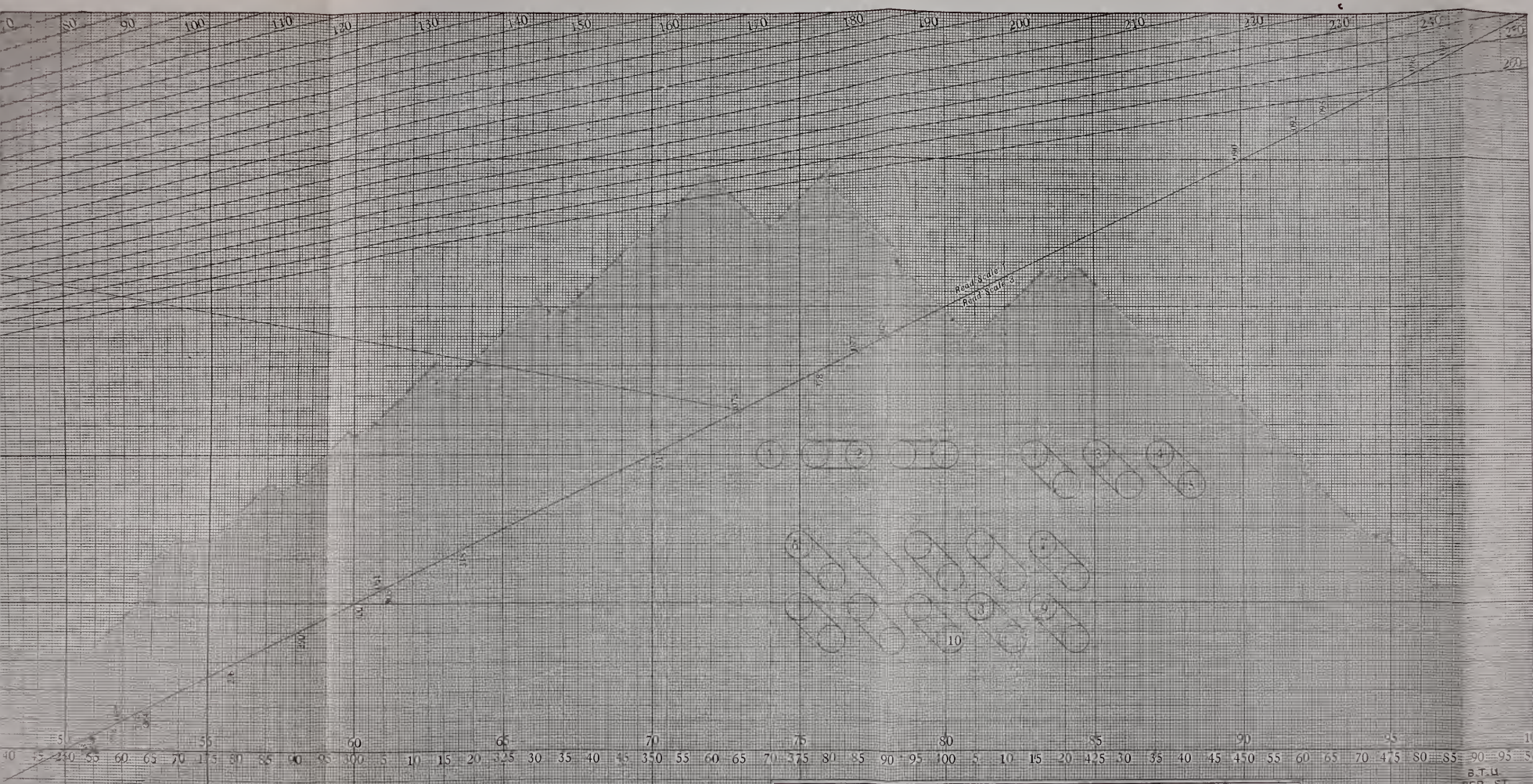
ENCE IN TEMPERATURE, DEGREES FAHRENHEIT



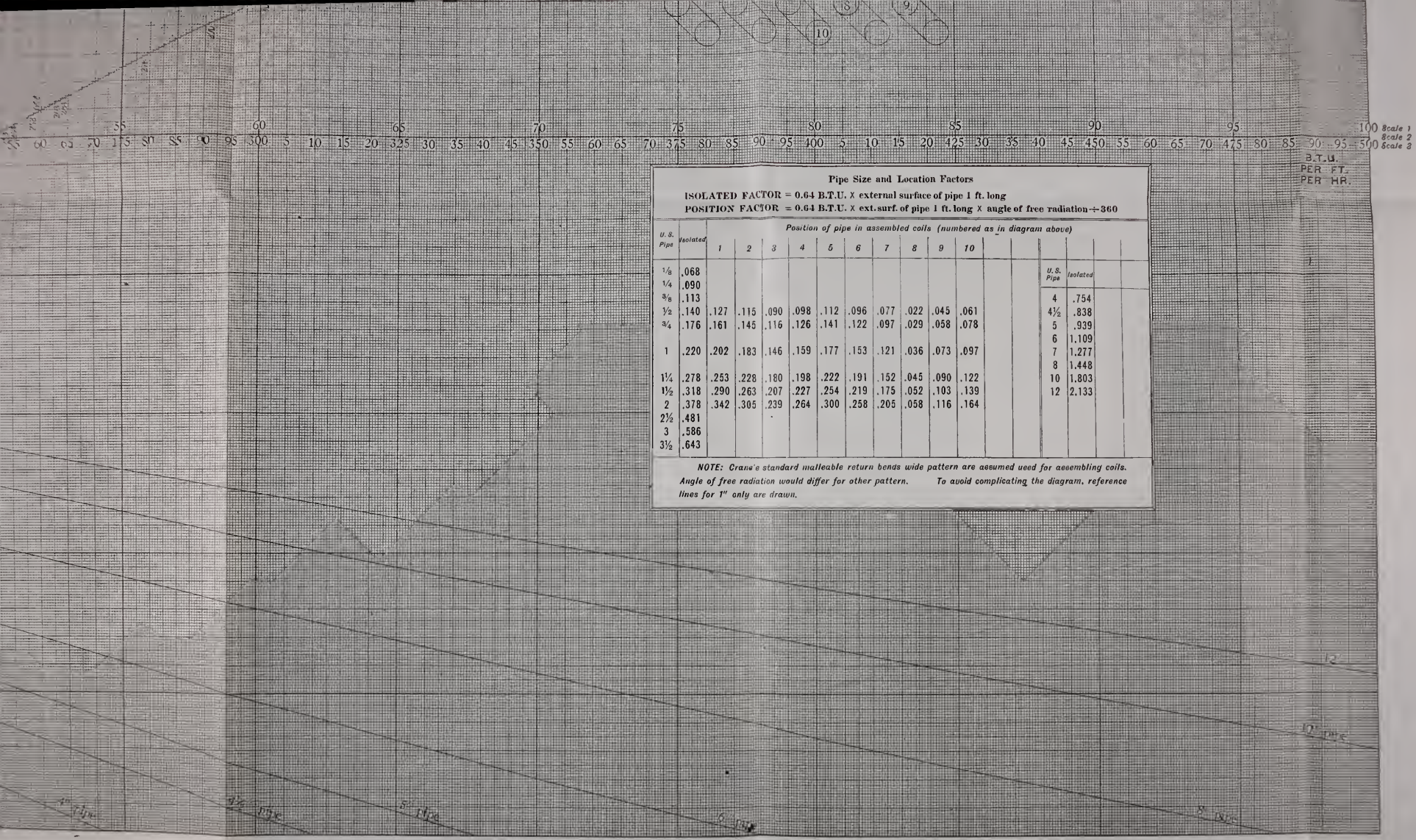
Pipe Size and Location Factors

ISOLATED FACTOR = 0.64 B.T.U. X external surface of pipe 1 ft. long

POSITION FACTOR = 0.64 B.T.U. X ext. surf. of pipe 1 ft. long X angle of free radiation ÷ 360

U. S. Pipe	Isolated	Position of pipe in assembled coils (numbered as in diagram above)									
		1	2	3	4	5	6	7	8	9	10
⅛	.068										
¼	.090										
⅜	.113										
½	.140	.127	.115	.090	.098	.112	.096	.077	.022	.045	.061
¾	.176	.161	.145	.116	.126	.141	.122	.097	.029	.058	.078

U. S. Pipe	Isolated
4	.754
4½	.838
5	.939



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U. S. Pipe	Isolated	Position of pipe in assembled coils (numbered as in diagram above)									
		1	2	3	4	5	6	7	8	9	10
⅛	.068										
¼	.090										
⅜	.113										
½	.140	.127	.115	.090	.098	.112	.096	.077	.022	.045	.061
¾	.176	.161	.145	.116	.126	.141	.122	.097	.029	.058	.078
1	.220	.202	.183	.146	.159	.177	.153	.121	.036	.073	.097
1¼	.278	.253	.228	.180	.198	.222	.191	.152	.045	.090	.122
1½	.318	.290	.263	.207	.227	.254	.219	.175	.052	.103	.139
2	.378	.342	.305	.239	.264	.300	.258	.205	.058	.116	.164
2½	.481										
3	.586										
3½	.643										

U. S. Pipe	Isolated
4	.754
4½	.838
5	.939
6	1.109
7	1.277
8	1.448
10	1.803
12	2.133

NOTE: Crane's standard malleable return bends wide pattern are assumed used for assembling coils. Angle of free radiation would differ for other pattern. To avoid complicating the diagram, reference lines for 1" only are drawn.

F HEAT FROM PIPES

Accompanying "Calculations for Dryer Design" by WILLIAM M. GROSVENOR.

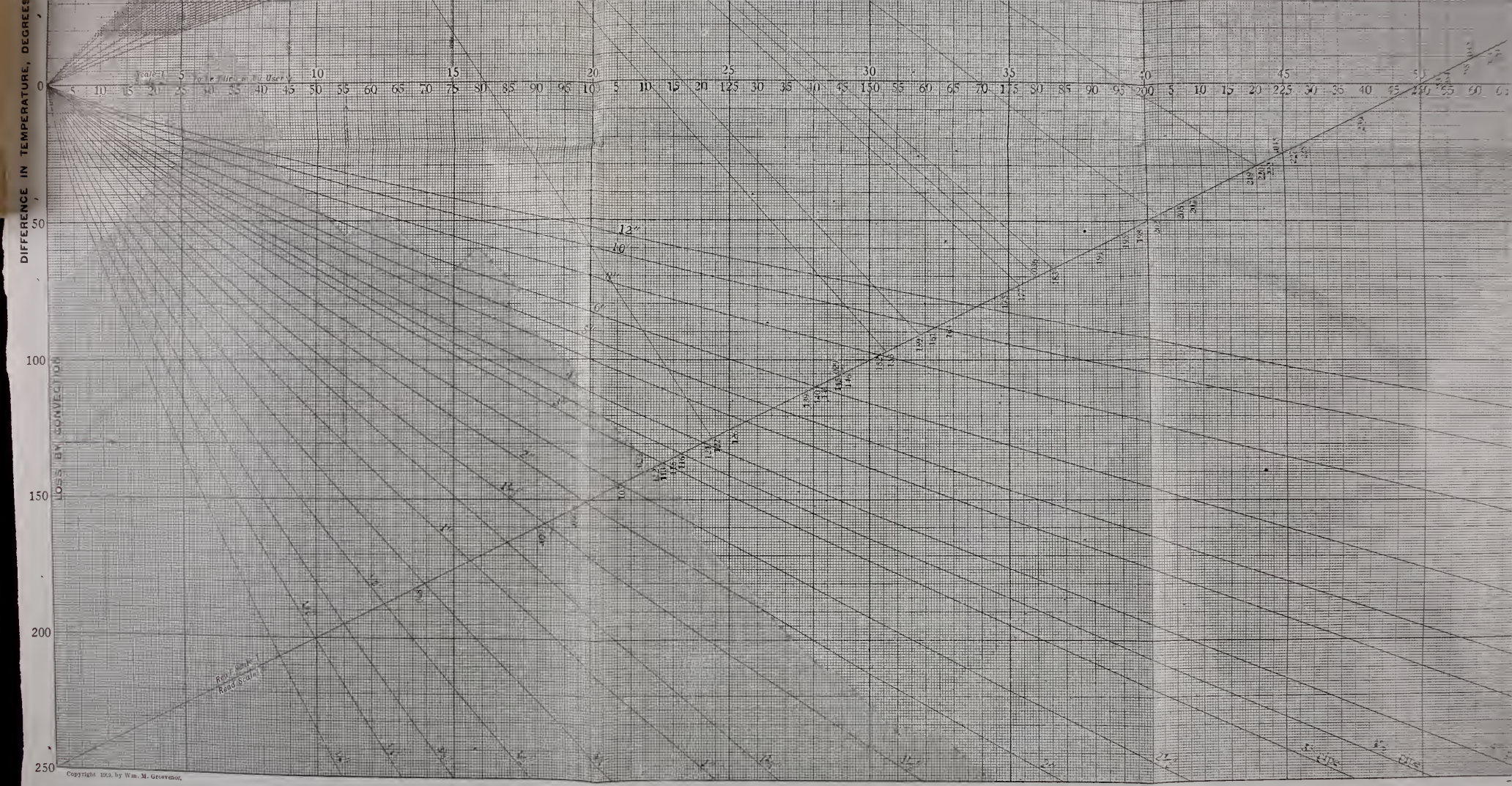
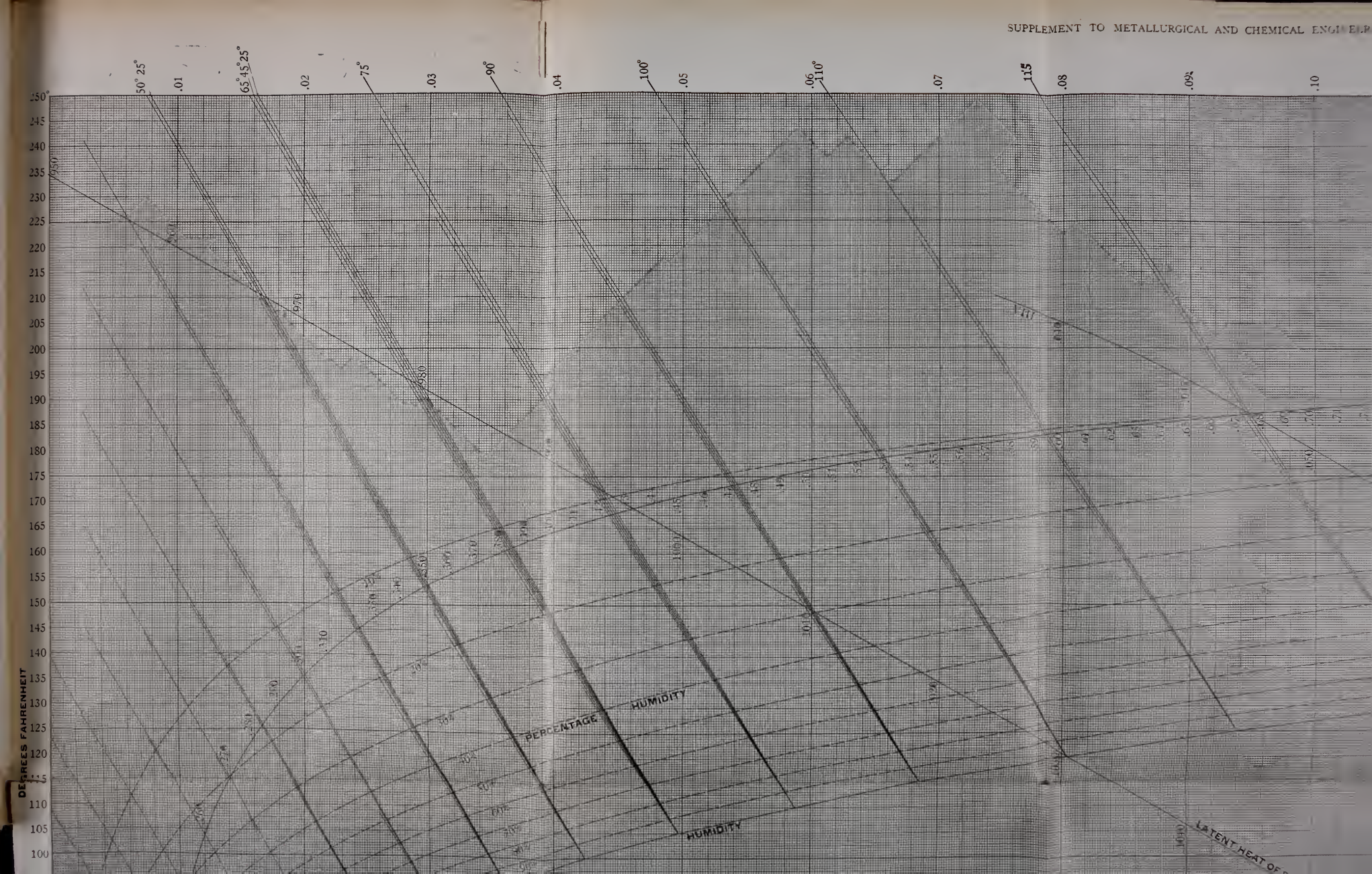


CHART II: LOSS OF HEA


DEGREES FAHRENHEIT
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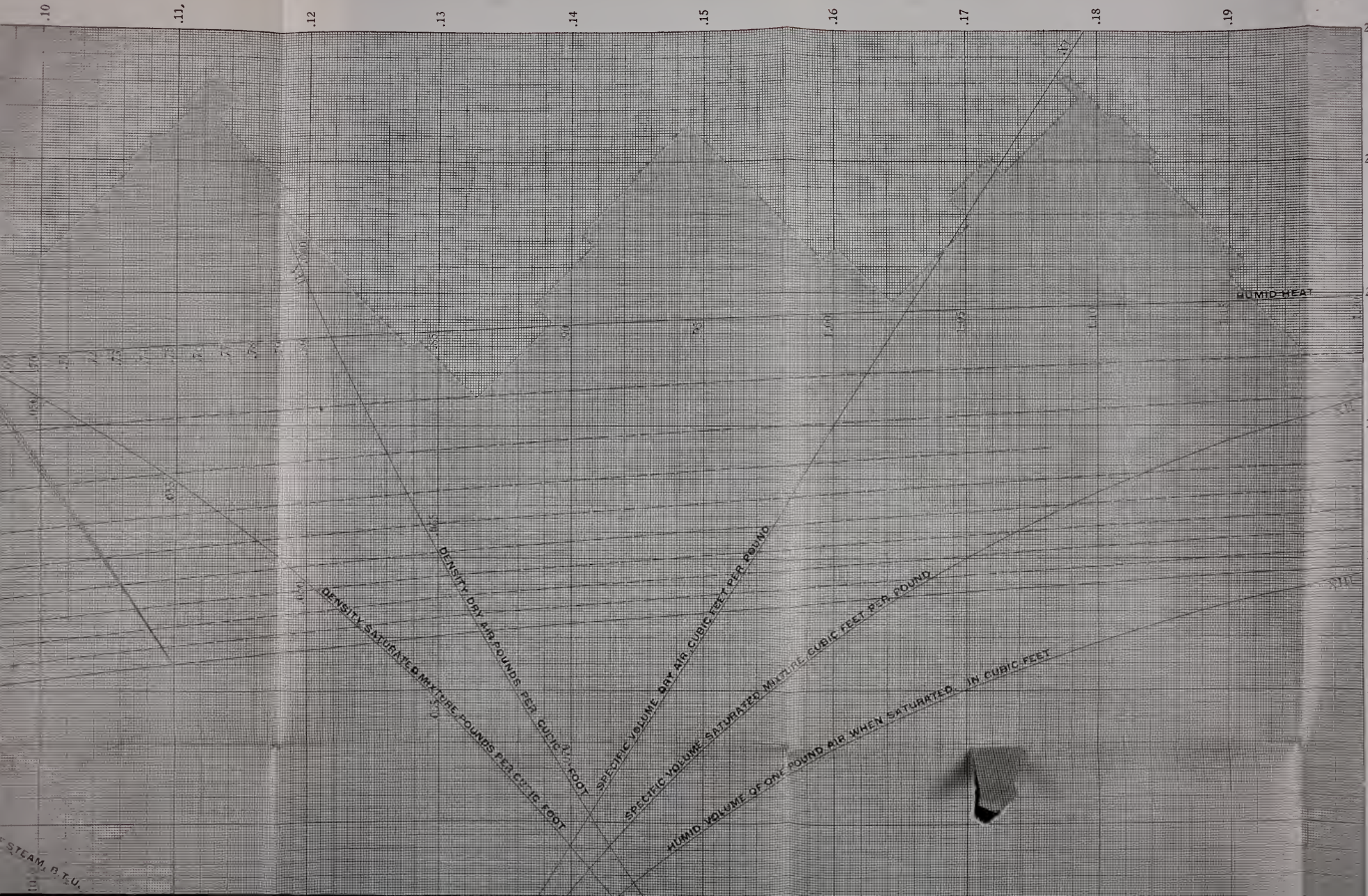
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225
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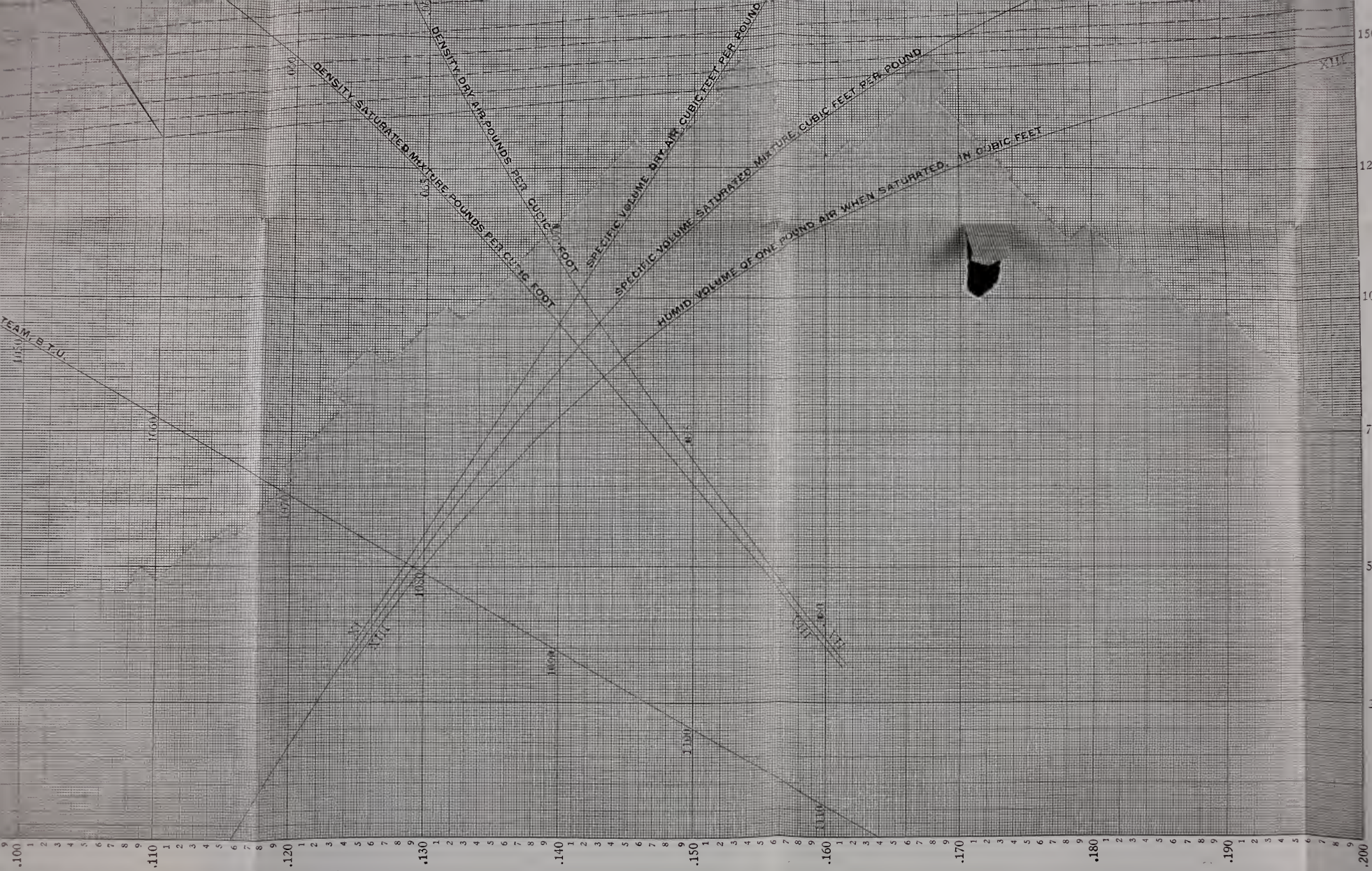
175
150
125
100
HUMID HEAT
DENSITY DRY AIR POUNDS PER CUBIC FOOT
DENSITY SATURATED MIXTURE POUNDS PER CUBIC FOOT
SPECIFIC VOLUME DRY AIR CUBIC FEET PER POUND
SPECIFIC VOLUME SATURATED MIXTURE CUBIC FEET PER POUND
HUMID VOLUME OF ONE POUND AIR WHEN SATURATED IN CUBIC FEET
STEAM, B.T.U.



IDITY CHART

Accompanying "Calculations for Dryer Design" by WILLIAM M. GROSVENOR.

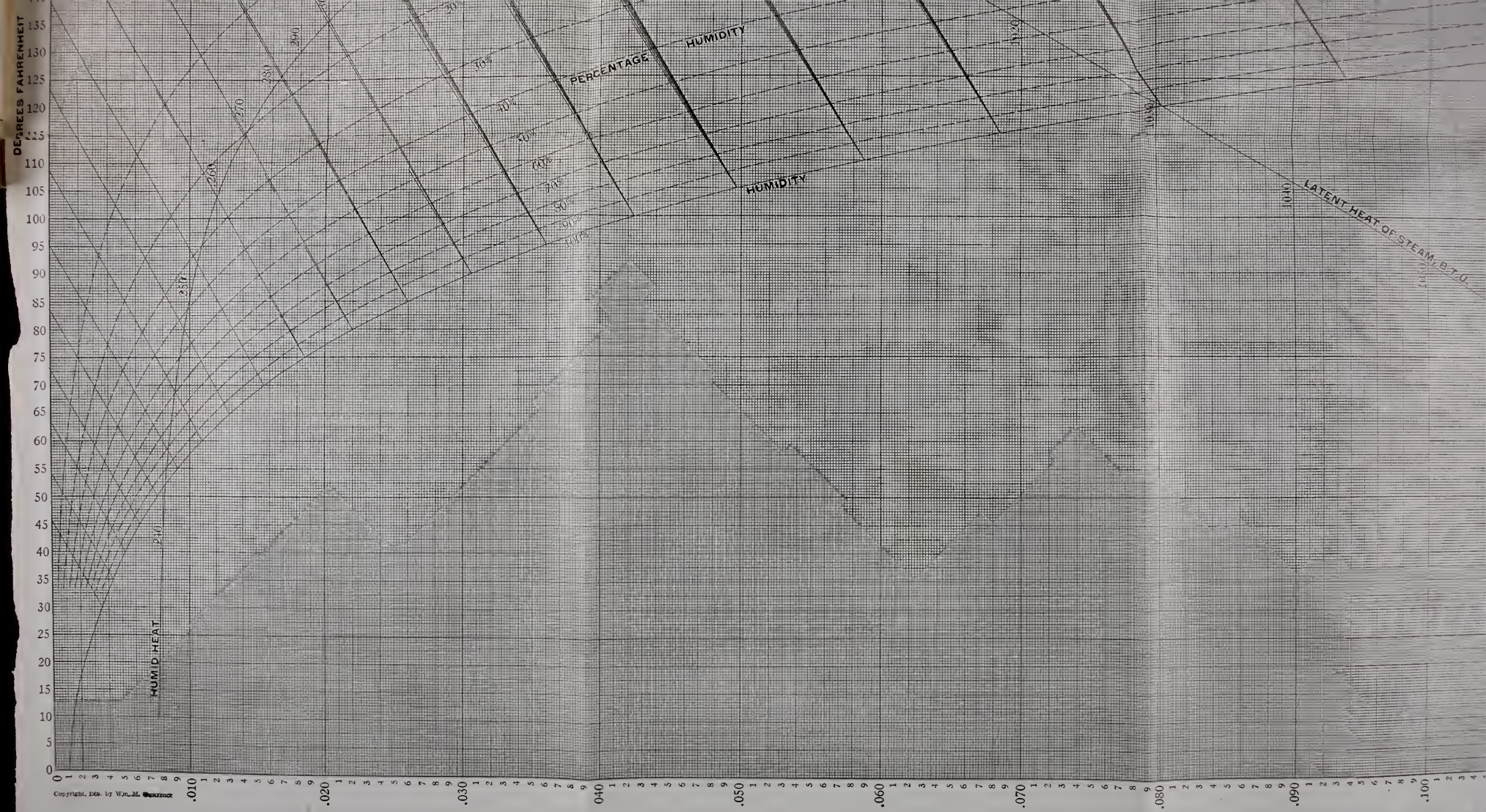


CHART I: HUMIDITY

vations were promptly converted into actual purchases. By the middle of January more ore had been sold than in any previous year at the date. All the merchant Bessemer ore that will be brought down has been sold, and by far the major part of the non-Bessemer. A feature of the buying was that three independent steel interests, normally self supporting in ore, bought a total of about 750,000 tons of merchant ore, their purchases in previous years having been very light.

The present ore prices have not been equaled since the opening of the Mesabi range, except in 1907, but a conspicuous difference in conditions is that pig iron is from $4 to $5 a ton lower than it was when the heavy buying was done for the season of 1907. By so much has the relation between Lake Superior ore and the pig iron which is made from it changed in three years.

Vessel rates from upper lake ports to Lake Erie ports were advanced five cents from the 1909 basis, to 70 cents from Duluth, 65 cents from Marquette, and 55 cents from Escanaba. At the same time the Lake Erie docks reduced the unloading charge, which is paid by the vessels, from 20 to 15 cents, so that the vessels are in 10 cents a ton better position than last year.

Other transport charges on Lake Superior ore are unchanged. The rate from the Mesabi range to docks at the head of the lakes remains at 80 cents, this including the dockage, and this with the 70 cent vessel rate makes a total cost of $1.50 delivered through Lake Erie docks. The rail freight from Lake Erie is 56 cents to the Mahoning and Shenango valleys and 98 cents to Pittsburgh, making the total transport charge from Mesabi mine to Pittsburgh furnace $2.48, covering a distance of more than a thousand miles. The traffic, nevertheless, is quite profitable. The statement is generally credited that there is more than 40 cents profit to the Minnesota ore roads, while Lake transportation must be fairly profitable or ore interests would not have been building lake vessels in the past few years, while the rail rates at the lower end are admittedly quite satisfactory to the railroads.

Pig Iron.

Southern iron has experienced a fairly good movement in January, prices undergoing no change. In a limited way iron can be bought on the basis of $13.75, Birmingham, for No. 2 foundry, if for immediate shipment, while $14 is given for extended delivery, to July 1 and probably through third quarter. The Philadelphia market has shown a fair turnover in basic, at $18.75, delivered eastern Pennsylvania. The valley market is practically unchanged as to prices, although the tone became easier during the month. Basic and No. 2 foundry are available at $17, valley furnaces, for first quarter, and at $17.25 for extended delivery. Bessemer remains firm at $19, valley, although quiet. Buffalo has shown the lowest prices at northern furnace, $17 being shaded rather freely.

Steel.

The tone of the unfinished steel market has been slightly easier. Sheet bars are available at $29, maker's mill, Pittsburgh or Youngstown, for extended delivery, and occasionally odd lots are offered at a slight concession. Billets may be roundly quoted at $27.50 for Bessemer and $28 for open-hearth, Pittsburgh. Wire rods are very quiet at $33, Pittsburgh. A comparatively easy market for unfinished steel is to be expected, as the pressure for delivery of finished material is not equally distributed among the different finished lines.

Finished Materials.

There has been no definite trend in prices of finished steel products. Merchant steel bars, which were available in December in a limited way at $1.45, Pittsburgh, although the general market was more fairly quotable at $1.50, have been firmly on the basis of $1.50 since the first of the year. Hoops are likewise slightly higher, being quotable at $1.60 to $1.65. Other finished steel products are unchanged. The market is quotable as follows, at Pittsburgh unless otherwise noted:

Standard steel rails, $28 for Bessemer, $30 for open-hearth, f.o.b. mill, except Colorado; light rails, 40 and 45-pound, $27.

Plates, $1.55 for tank quality, ¼-in. and heavier.

Shapes, $1.55 for beams and channels, 15-in. and under, zees and angles 2 x 3 and larger.

Steel bars, $1.50, base.

Iron bars, $1.65, Pittsburgh; $1.60, Philadelphia; $1.55 to $1.60, Chicago.

Wire nails, $1.85, base; plain wire, $1,.65; galvanized barb wire, $2.15.

Black sheets, 28 gage, $2.40; galvanized, $3.50; blue annealed, 10 gage, $1.90; painted corrugated roofing, $1.70 per square; galvanized, $3.

Tin plates, $3.60 for 100-lb. cokes.

CORRESPONDENCE.

Silundum.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—In your issue of December, 1909, in which is contained the report of the meeting of the American Electrochemical Society, I find the paper on "Silundum," by Prof. S. A. Tucker, and I regret to say that I am incorrectly reported as follows: "Dr. Acheson remarked that silundum was nothing new; it was the same product that he had called siloxicon." So near as I now remember what occurred was as follows: The president invited me to open the discussion following the reading of Professor Tucker's paper and I made these remarks: "Mr. President, you will kindly excuse me from entering into a discussion of this matter. From what I had formerly heard regarding silundum I was under the impression that it was a compound of carbon, silicon and oxygen, and was, therefore, the compound which I have designated as siloxicon. Since hearing the reading of Professor Tucker's paper, I now appreciate the fact that there is no oxygen present."

My reason for asking to be excused from entering into a discussion of the product as produced by Mr. Bölling was due to the fact that Mr. Bölling was formerly my assistant, he having entered my employ, I believe, in the latter part of the year 1893, while I was still at Monongahela City, and he went with me to Europe as assistant engineer in 1894, and I returned to this country leaving him abroad.

In the active zone of the furnace, as I understand it is operated by Mr. Bölling, there are three elements which may react on each other, viz., carbon, silicon and oxygen. In analyzing the possible reactions and the products that might be obtained from these three elements we find that the furnace content may be so proportioned and the furnace so operated as to produce either graphite, metallic silicon, carborundum, or amorphous carbide of silicon (as made by P. Schützenberger and described by him in a communication to the Academy of Science of France, May 16, 1892), or we may have all the three elements so associated as to produce siloxicon. As to the possibility of any other product being made, I am not advised, but the foregoing were old before the world was informed of Mr. Bölling's experiments.

NIAGARA FALLS, N. Y. E. G. ACHESON.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—In the December issue, 1909, I find an account of an American Electrochemical Society paper on silundum by Prof. S. A. Tucker, H. F. Kudlich, and E. M. Heumann, and in the discussion of this paper a statement of Dr. Acheson that silundum was nothing new; that it was the same product which he had called silixicon.

I must contradict Dr. Acheson and I claim the priority of the discovery of this new material and of the fact that silicon vapors penetrate into carbon, when both are in a highly heated state, converting the carbon into a refractory, electrically conducting material, of a composition with varying proportions of

silicon and carbon. Silundum, therefore, as described in my paper, published in this journal, January, 1909, in form of rods, tubes, crucibles, or any other shape made of carbon, is new. The date of my first German patent is April 24, 1904, and I started experimenting October 31, 1903.

Silundum is used now in Europe for various purposes, especially for electric heating purposes. It is manufactured by the Chemisch Elektrische Fabrik Prometheus in Frankfurt.

Oberwesel, near Frankfort, Germany. F. Bölling.

Iron-Copper Alloys.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—I have read with considerable interest the abstract of Charles F. Burgess' and James Aston's paper on "Iron-Copper Alloys," published in your December issue (Vol. VII, page 527), and believe that the following points are of sufficient importance to bring to the attention of your readers. In the preface the following statement is made:

"The influence which copper has in iron and steel has for long been a subject of controversy and contradiction. It is only comparatively recently that the effect has been investigated systematically and some of the doubt eliminated.

"The prevalence of appreciable quantities of copper in the ores of certain districts makes important a knowledge of its effect, detrimental or otherwise, when present in small amounts. Also the intimate relation of copper, nickel and iron, and the beneficial effect of the addition of nickel to iron and steel, creates an interest in the influence of larger percentages.

"The older opinion was that copper was deleterious; that its chief effect was analogous to that of sulphur in that it rendered iron red short and destroyed its welding power. This view was taken by many eminent metallurgists, some even contending that 0.5 per cent makes steel worthless. Others claimed that the influence was greatly exaggerated, and instances were cited where rolled sections of 0.50 per cent to 0.75 per cent of copper did not display red shortness."

With this statement, the authors proceed to manufacture the alloys, using pure iron and pure electrolytic copper, and while the abstract of the paper does not so state, I judge that the alloys were made from these materials, using the crucible to melt the same. In the manufacture of crucible steel the base materials are melting bar which is obtained by puddling, rolling and rerolling; blister bar obtained by carbonizing the melting bar, and "medicine." This latter consists of black oxide of manganese, clay and any of the ferro-alloys which may be required to produce the kind of steel.

The base material in the crucible manufacture is, therefore, practically wrought iron, which would be free from any copper that might have been contained in the ore from which it came. The puddling and reworking operations would have completely eliminated the copper with the slag, so that it is quite safe to say that no crucible steel contains copper except that intentionally added or introduced with the alloys. Furthermore, crucible steel is the highest grade of steel and is made from selected bar iron, which is usually made from the highest grade of ores and these are free from copper.

These base materials are charged into a pot and protected from the air by a cover and the slag, so that the steel is made in practically a closed receptacle away from all oxidizing influences. Therefore, the resulting product is quite free from all oxides and therein to a large measure lies the superiority of the crucible steel. It would follow that the alloys manufactured by the authors would be the freest from oxides, and have been produced by a process which is never used in the manufacture of steels in which copper from the ores is likely to be found.

The writer believes that the oxide or suboxide of copper is the real cause of the red shortness which is characteristic of the steels containing copper, and all of such steels are the product of the open-hearth or Bessemer processes. In both these processes there is every chance for the formation of an oxide of copper since these steels contain, in comparison to the crucible steel, considerable quantities of the oxide of iron, silicates, etc.

The quantity of the oxide is largely dependent upon the composition of the steel; for instance, that steel low in carbon and manganese is usually found to contain more oxides since the oxidizing action has been carried to a greater degree. Furthermore, the steel made in the basic open-hearth furnace usually contains more oxide than acid open-hearth steel. In the latter process the basic oxides combine with the acid slag and are more readily eliminated from the steel.

Hence, it follows that the amount of copper permissible in open-hearth or Bessemer steel, without the production of red shortness, is of variable amount depending upon the composition of the steel, the degree to which it has been subjected to an oxidizing reaction, and the degree to which the oxides formed by the oxidizing reaction have been eliminated from the steel.

In other words, the degree of red shortness produced by the presence of the copper is dependent upon the conditions of the

HOT ROLLED STEEL.

manufacture and the other elements in the steel. My experience has shown that when the carbon and manganese are low (carbon less than 0.10 per cent, manganese less than 0.12 per cent) the presence of 0.5 per cent of copper produced a decided red shortness. The photographs of some hot rolled steel, herewith reproduced, speak for themselves, showing how the material acted under the rolls. That on the right is higher in manganese by 0.06 per cent and contains less copper. It rolled out very well, showing no red shortness. That on the left is higher in copper and lower in manganese. It showed decided red shortness, as the photograph indicates.

The paper by Messrs. Burgess and Aston is a very good one and quite interesting, but the preface does not seem to me to be quite in harmony with the work which they have done. The two together give an erroneous impression, and the comment made by Dr. J. W. Richards in the discussion (that much of our steel contains copper, from 0.5 to 0.7, that it has been claimed privately that the copper is not injurious, but rather beneficial and that this has now been established by Professor Burgess) would seem to me had been made while under the influence of this wrong impression.

I trust that I have made my remarks sufficiently clear so that the points, which I wish in particular to bring out, will be considered.

Reading, Pa. Geo. W. Sargent,
Metallurgist Carpenter Steel Company.

The Presentation of the Perkin Medal to Dr. E. G. Acheson.

At the meeting of the New York Section of the Society of Chemical Industry, held at the Chemists' Club on Friday, Jan. 21, the Perkin medal was presented to Dr. EDWARD GOODRICH ACHESON, of Niagara Falls. In spite of the inclement weather the meeting was well attended, not only by members of the Society of Chemical Industry, but also by members of the American Chemical Society and of the American Electrochemical Society, Dr. Acheson being a past-president of the latter society.

Mr. MAXIMILIAN TOCH, as chairman of the New York Section of the Society of Chemical Industry, referred in his introductory remarks to the presence of so many who had come from a distance to attend this meeting, and said that "it is seldom given to any man that, when his work is successfully crowned, it be done in the presence of so many prominent scientists. We are assembled to-night to do honor to our fellow-member, Dr. Edward Goodrich Acheson, to present to him the Perkin medal, which originated practically in this hall four years ago. It will always be a credit to the Society of Chemical Industry that the first Perkin medal was presented to Sir William Henry Perkin himself, and many a brilliant gathering has taken place here, but none more brilliant than that at which we are assembled."

Mr. Toch also said that "although Dr. Acheson's name has been linked for many years with the invention of artificial graphite and carborundum, still those of us who have watched his scientific career more closely have, like myself, been most interested in the work he has done in colloid chemistry."

Mr. Toch then called upon Dr. Charles F. Chandler to present the Perkin medal to Dr. Acheson.

E. G. Acheson

Address of Presentation.

Dr. C. F. CHANDLER spoke as follows:

"It is my pleasant duty, as the senior past-president of the Society of Chemical Industry in this country, to present to Dr. Edward Goodrich Acheson the fourth impression of the Perkin medal.

"Dr. Acheson possesses the rare merit of combining the inventive, the constructive and the organizing faculties. He has been most successful in discovering entirely new materials suitable for a great variety of purposes which have become indispensable to the world. He has also been able to devise the most perfect methods and appliances for producing his new products on a large scale, and he has been able to organize and finance great companies to put his inventions into successful operation."

Dr. Chandler outlined briefly the early life of Dr. Acheson (see our Vol. I, page 90), and then spoke at greater length on the chief inventions of Dr. Acheson, as follows:

"Carborundum."—In 1886, while making some experiments, he passed a quantity of hydro-carbon gas over highly heated clay and observed that the clay became impregnated with carbon. In the early part of the year 1891, having at his command an electric generating plant of considerable capacity and looking about for a line of experiments, he thought to try the impregnating of clay with carbon under electric heat. The mass resulting from the experiment exhibited a few small crystalline specks. These he carefully examined and found that they not only scratched, but actually cut glass. They appeared like minute gems. Grasping the possible value of his discovery, he undertook further experiments and after two months' labor succeeded in obtaining a few grams of crystals. In order to test their value as an abrasive, he mounted an iron disk in a lathe, charged its surface with his new unknown product and found that this revolving disk cut the polished face off a diamond.

"Having satisfied himself of the abrasive quality of his new product he took it to a diamond cutter in New York, who tested it, and was glad to buy it at 30 cents per karat; and with the proceeds of this first sale Acheson purchased a microscope to assist in the further study of the material. Having thus obtained this material from clay and carbon, and not yet knowing its chemical composition, he gave it the new name 'carborundum,' but he soon learned that it was the silicon of the clay, and not the alumina, which united with the carbon, producing a carbide of silicon, but the name 'carborundum' has been retained for the substance ever since.

"In 1894 he established the Carborundum Company at Monongahela City, Pa., but subsequently moved to Niagara Falls, where he finally erected works and equipped a plant with 5000 electric horse-power and where he has manufactured more than 7,000,000 lb. of carborundum.

"Time will not permit me to enumerate the various uses to which this new abrasive substance was applied. Suffice it to say that nearly 600 men are employed in the works at Niagara Falls in preparing carborundum and fashioning it into wheels, sharpening stones, hones, bricks, knife sharpeners, paper, cloth, etc., fitting it for every kind of abrasive application. I should not omit to say that carborundum has proved a special substitute for ferro-silicon in the manufacture of steel.

"Artificial Graphite."—In the carborundum furnace Acheson noticed that in the hottest zone, next to the carbon core, there was produced between the core and the carborundum crystals a thin layer of graphite, which he decided must be produced by the dissociation of a portion of the carborundum and the expulsion of its silicon in vapor. The discovery led Acheson to make a long series of experiments, the result of which was the discovery of a process for converting amorphous carbon, such as coke or anthracite, into beautiful artificial graphite. As a result the International Acheson Graphite Company was organized and large works were erected for the manufacture of artificial graphite. This company turns out several million pounds of what is termed bulk graphite made principally from anthracite coal.

"Graphitized Electrodes."—In addition to the bulk graphite, shapes suitable for electrodes are prepared, and by subsequent exposure to the proper heat in the electric furnace, they are converted into graphite, hard and compact. These graphitized electrodes have proved of the utmost value in various electrolytic operations, as they are found to be many times more suitable than the carbon electrodes previously used. The company manufactures several million pounds of these graphitized electrodes annually.

"Soft Graphite."—The artificial graphite produced by the International Acheson Graphite Company was hard and well suited for use as electrodes for electric furnaces and electrolytic cells and for many other purposes. In 1906 Mr. Acheson devised a process for producing a soft graphite. This graphite is especially valuable for use as a lubricant, a stove polisher, coating electrotype molds, etc. This graphite is not only pure and soft, but unctuous, and does not coalesce when compressed.

"Deflocculated Graphite."—In investigating different clays in connection with the manufacture of artificial graphite crucibles Mr. Acheson's attention was called to the great difference in the plasticity of various clays, and especially to the fact that clays far removed from their place of origin were most plastic,

and the idea came to him that this might be due to the fact that in their transportation by water they had become impregnated with vegetable extracts which might increase their plasticity. With this idea in mind, he conducted a series of experiments with various clays and discovered that by treatment with tannin the soft graphite is so modified that it remains suspended indefinitely both in water and in oil, and he immediately realized the importance of the discovery, as enabling him to supply the world with a new lubricating agent invaluable in all purposes where lubricants are required. The value of this discovery is best shown by the fact that a shaft can be run from 50 per cent to 100 per cent faster with oil containing deflocculated graphite than with the plain oil.

"Siloxicon.—In the latter part of the summer of 1893 Acheson became interested in the direct reduction of silicon, and among other experiments he mixed graphite and silica, and exposed the same to the electric current with a view of producing silicon. He produced in this way a peculiar product which he found to be highly refractory. Analysis showed it to be composed of silicon, carbon, and oxygen, hence the name siloxicon. A special company was organized for the manufacture of this interesting product and for producing from it such articles as crucibles, muffles, bricks, etc.

"It is impossible in the short time allotted to me to give anything like an adequate account of all of Dr. Acheson's achievements. Nor it is even possible to give a full account of any one of them. But I have said enough to make it clear that Dr. Acheson is an inventor of the first order—a thorough electrician and a thorough chemist. His efforts have not been directed merely to improving processes and products previously known. He has made entirely new departures and created new industries, supplied previously unknown materials for use in the arts and industries, and he has discovered uses for his new materials and processes for their application. He is a representative electrochemical engineer, and it is eminently proper that he should have been selected by the great chemical societies of this country to receive the Perkin medal.

"It gives me great pleasure, Dr. Acheson, as the representative of the Society of Chemical Industry and the affiliated chemical and electrochemical societies, to place in your hands this beautiful token of the appreciation and affection of your fellow-chemists."

The Dawn of an Electrochemical Epoch.

Dr. E. G. Acheson replied as follows:

"It was a coincidence of more than ordinary importance to me that I first saw the light of this world in the spring of 1856—the very year made memorable by the great discovery of the dye stuff 'mauve' by Sir William Perkin. During my childhood and early manhood occurred the development of most of the great chemical and metallurgical industries with which we are now so familiar.

"In the last quarter of the nineteenth century the world was ready for another step forward in the chemical arts. It was ripe for the birth of a new industrial chemistry, until then unknown—industrial electrochemistry. I believe this birth occurred at the time of Hall's work on aluminium. The great movement for the conservation of our natural resources calls into action a vast amount of work that properly will fall to the chemists of all kind of industries, especially the electrochemists. Their field of research and development will be practically limitless and will extend beyond what the world is now considering the conservation of natural resources, for they will create valuable products from matter now classed by the title of refuse—a field largely devoted to synthetic work.

"I wish to thank you and the societies which you represent, Mr. ——, for the great honor you have conferred upon me, and I have no greater or more sincere desire than that my work may, in a small way, stimulate the rapid expansion and development of synthetic electrochemistry which will result in a great electrochemical epoch, following the chemical era brought to life by the illustrious scientist in whose honor this medal was created."

Dr. Acheson then entered into a description of some of his more recent work on deflocculated graphite, aquadag and oildag. He spoke in an informal way and illustrated his remarks by exhibits and various experiments. It was exceedingly interesting to follow in detail the various steps of discovery, speculation, and invention by which Dr. Acheson finally realized what he prefers to call "deflocculated graphite," or graphite in the colloidal state.

A Classic for Children and an Inspiration to Chemists.

Dr. Wilder D. Bancroft, of Cornell University, then spoke as president of the American Chemical Society, as follows:

"On an occasion like this it is an honor to represent anything. It is a great honor to represent the American Chemical Society. There is a personal reason why it is an especial pleasure to me to be one of the speakers on the particular evening on which the Perkin medal is awarded to Mr. Acheson.

Most of you can remember the time when physical chemistry was looked upon as a fad, interesting enough, but purely theoretical. Some of us preached in season and out of season that a proper training in physical chemistry was the ideal training for an industrial chemist. We were looked upon as harmless maniacs and consequently were not ill-treated. It did not do any good for us to talk like that unless we could prove it—and there was always the painful possibility that we might be wrong.

"I do not wish to claim for Mr. Acheson that he was the first manufacturer to employ a physical chemist at a good salary. I am always hazy on the subject of dates, but I imagine that that honor belongs to the General Electric Company. Mr. Acheson was the first manufacturer, however, who was rash enough to try the Cornell product.

"Once the ice was broken everything went well. So far as I know, the properly trained physical chemist has always made good in industrial work. You remember the French proverb 'Ce n'est que le premier pas qui coute,' which may be rendered into the vernacular as 'It is only the first year that you don't earn your salary.'

"A man who receives the Perkin medal is expected to have made several great inventions. That has been the case so far, and, from what I know of American chemists, the supply is likely to hold out for many years to come. There is a great difference in the work of different inventors when considered as an artistic whole. From a pictorial point of view their inventions may often be said to be spotty; they are determined by business exigencies, or they are like lightning, which is said never to strike twice in the same place. I have never tried the experiment.

"Take our friend Backeland, for instance. He is after me tonight, but I am after him now. He has made two great inventions. Velox is one and bakelite is the other. There is no visible connection between them, and there is no fundamental reason why they should have been made by the same man. They are individual outbursts of genius.

"Take the first man to whom the Perkin medal was awarded, Mr. Herreshoff. I remember that Dr. Chandler cited as two of his inventions improvements in the series system for refining copper electrolytically and improvements in the contact process for making sulphuric acid. Copper sulphate is the electrolyte used in the electrolytic refining of copper and it is made from sulphuric acid, but these are really separate inventions. So far as I can see the only thing which they have in common or which holds them in common is the General Chemical Company.

"If we do not limit ourselves to the field of chemistry everybody will admit that Mr. Edison is the greatest of American inventors. He converted electrical energy into light in the case of the incandescent lamp, he converted electrical energy into sound in the case of the phonograph or gramophone, whichever it was that he invented, he converted chemical energy into

electricity in the case of his storage cell. These inventions hang together in a very satisfactory way; but how are you going to fit in his invention of monolithic cement houses for the poor?

"In the case of Mr. Acheson there is a logical sequence about his inventions which delights the soul of any one like myself who lectures on applied electrochemistry. Mr. Acheson invented carborundum. The carbon in the cone of the furnace changed partially to graphite and he worked out a method for preparing graphite on a commercial scale. On the outside of the carborundum layer there was a ring of 'white stuff' as it was called, and Mr. Acheson organized a company to manufacture siloxicon. There were lumps of silicon throughout the carborundum and a former assistant of Mr. Acheson, Mr. Tone, worked out a process for making silicon, thereby giving the commercial world a new element. Everything in the carborundum furnace has been worked into an industry with the exception of the unchanged charge and that they use over again, so that it is not wasted.

"If Mr. Acheson will forgive me for slurring over the distinction between discovery and invention, I should call him the Swiss Family Robinson of chemistry. You remember the book. The shipwrecked family went out to search for food, and they discovered a bread-fruit tree with sandwiches growing on it. They came to a stream and they invented a cantilever bridge which they constructed with the aid of a bow and arrow and a dog. They discovered or invented a palm tree with cocoanuts growing at the top. In order to get the cocoanuts they invented or discovered a shark stranded in the shallows; they made gaiters of his skin and climbed the tree. Now nature had intended the cocoanuts to be blown off the tree, and when they were brought down by hand it at once became a problem how to open them. Our friends thereupon discovered or invented a crab which inserted the points of its claws in the holes of the cocoanut and spun a yarn which broke the shell and enabled them to get the milk in the cocoanut.

"The analogy with the work of Mr. Acheson is obvious. He began by trying to make diamonds. He denies this, but the myth persists in spite of him. When the crucible was opened there were no diamonds but instead a mass of multi-colored jewels. When he discovered that they were not real gems it was a crushing blow and so hard that he decided to use them as an abrasive. To make carborundum he needed electrodes, so he invented graphite. A suitable refractory was needed to keep the heat in the furnace, and Mr. Acheson thereupon prepared siloxicon. The furnace had to have brick walls at first, and they were promptly forthcoming in the form of Egyptianized clay. In order to keep everything running smoothly, including the three automobiles which he had acquired in the meantime, Mr. Acheson prepared the lubricating graphite which he has shown you to-night as aquadag and oildag.

"When you stop to think of him, Mr. Acheson's work is really a classic for children as well as an inspiration to chemists. He has revolutionized many branches of chemistry and he has revolutionized the whole of electrochemistry. I should like to say a good deal more about this last, but I should play havoc with Baekeland's typewritten speech if I did.

"You may consider Acheson's discoveries as scientific inventions or as dividend payers; you may even look at them in a flippant way. It makes no difference. They stand all tests and they stamp him as one of the great inventors of the world.

"I am speaking to-night for the members of the American Chemical Society, forty-five hundred strong at Christmas, and probably five thousand now unless Parsons has been taking a vacation. In the name of the American Chemical Society I congratulate you, Mr. Acheson, upon the honor which you have received to-night and which you have so richly deserved."

Pathfinder, Conductor, Switchman—Everything.

The last address of the evening was made by Dr. L. H. Baekeland, as president of the American Electrochemical Society. Dr. Baekeland said:

"What is more to the credit of Dr. Acheson than anything else is the fact that he undertook experiments which probably had been attempted several times before without success by people who had more theoretical preparation than he. But his unusual keenness of observation, his commonsense logic, and more than anything, his appreciation of what was of preponderant importance and of secondary interest, brought him results where others had failed and by which he opened a new field of chemical investigation and industrial applications.

"Dr. Acheson stands as a living example to many a chemist, loaded with theoretical knowledge and paper wisdom and light in judgment or common sense. He entered into a new road with very little knowledge to go by, but he had an open mind, a fertile brain, a constructive purpose, and all the talents of the tireless, intelligent experimenter; his theoretical knowledge he made himself, or picked it up by and by along the road he was pursuing, while he himself was building that road. Whatever he took up, he started from the very beginning and developed it to the very end.

"His very insuccesses he turned into practical channels, to change them into victories. He started his work on carborundum by an experiment so simple and so meager in results that few people, if any, would, like himself, have found in this a spur to further investigation. Later on, when in the manufacture of carborundum, he had to battle with irregularities due to a partial dissociation of the silicon carbide, which thus spoiled his product, he tried by a more powerful action of heat to carry this dissociation to the extreme limit; he so created a new industry, the manufacture of artificial graphite. In the same way he turned irregularities due to oxidation or partial reduction into a new field of study and research and gave us siloxicon, a product which seems to have a considerable future in store.

"I shall not attempt to describe here the history of the work of Dr. Acheson. He himself, in a memorable address delivered in Boston before the American Academy of Arts and Sciences, when the Rumford Medal was awarded to him, has given us a most fascinating account of this subject, and to those of my fellow chemists who have not read his 'Seventeen Years of Experimental Research,' I can recommend it as an inspiration.

"Never did the University of Pittsburgh award an 'honoris causa' degree to a more worthy man than when the doctor's degree was conferred upon our distinguished friend.

"In the making of the great road of industrial progress, very few of us are path-finders; some are surveyors, other ones constructors, others, again, are merely switchmen, brakemen and conductors. Dr. Acheson not only was a path-finder, but he has been constructor, conductor, switchman—everything. By his manifold abilities and good judgment he has been able to develop his discoveries to the point where they became a commercial success. He has shown us that an inventor or a scientist does not have, necessarily, to be a theoretical dreamer, unfit for executive or practical work.

'Dr. Acheson has another great claim of recognition upon us; successful as he has been in his enterprises built on the result of his inventive genius, he has never ceased to make us feel that he was near to every one of us. He always taken an active part in the work of our chemical societies, where his presence and his papers have stimulated many of us younger in experience or less successful in our efforts. In many instances he has allowed our chemical societies to inspect the plants where his industrial processes are carried out, and he has never hesitated to publish the results of his research work. He has thus set a praiseworthy example to such of our manufacturers who try to insure by uncompromising secrecy and seclusion the money-making end of their enterprise.

"Dr. Acheson, in the name of the American Electrochemical Society, I congratulate you on this recognition of your splendid work, and I congratulate the Perkin Medal Committee just as much in their choice of you."

All speakers were applauded with considerable enthusiasm.

Treatment of Ore Slime.*

By Andrew F. Crosse.

The chief difficulty in treating slimy material with a cyanide solution is the difficulty of removing and obtaining the dissolved gold, or silver, from the slime. Many methods have been tried, and the methods chiefly in use to-day are the well-known decantation process and various filter-press processes. The decantation is very slow, and, therefore, requires an enormous plant, and a very large quantity of solution has to be handled in proportion to the weight of slime treated.

The filter-press method has many advantages, but it requires much handling and constant renewals. In my process the slime, having been collected in a suitable manner, is washed with a weak cyanide solution into a conical treatment vat. This vat is provided with an interior conical baffle, open at the bottom, and reaching about half way down the treatment vat. Inside the vat is a pipe extending, from nearly the bottom of the cone, to a foot or two above the surface of the pulp. This pipe has a small pipe introduced at the lower end, through which air is forced, this causes a current of concentrated or thickened pulp to be forced up the tube. At the same time the clear liquid, in the outer portion of the conical vat, is overflowing into the zinc box where the gold and silver are deposited, and then this clear solution, deprived of its gold and silver, is pumped by a small pump, which can be regulated as to speed, back into the top of the inside cone, where it mixes with the underflow brought up by the air-lift pump.

By this means the whole mass is kept in continuous circulation under the most advantageous condition for the solution of the gold or silver. As the air circulates and aerates the pulp, the gold and silver are being continually extracted from the solution. The same solution acts as a wash, practically all the gold and silver having been extracted and deposited in the zinc box. During the working of the process it is very easy to control the strength of the cyanide solution or the alkalinity, and should either be found too low more cyanide or alkali can be added.

Clear solution containing gold is passing through zinc boxes and depositing its gold in every cyanide plant. The circulation of cyanide solution and slime by means of an airlift pump is quite the ideal method of solution, as I have proved in hundreds of trials. Most gold ores produce on crushing a certain proportion of real slime, which is usually a hydrated silicate of alumina with an excess of silica, or a very finely divided hydrate of ferric oxide and finely ground pyrites.

Metallurgists all recognize to-day that no hydro-metallurgical gold treatment can be successful unless the ore has been sufficiently finely crushed in order to render every particle of gold or silver accessible to the solvent action of the cyanide solution. In order to carry this out tube mills, pans and other devices have been introduced, so that a large proportion of the crushed ore cannot be treated by percolation.

In my scheme I can handle any fine material, and I can treat successfully either ordinary argillaceous slime, fine sand, or finely ground pyrites. I have treated a highly ferruginous slime and obtained an extraction of 91 per cent of the gold contents from slime containing 7 dwt. fine gold per short ton. This means that the gold so obtained was precipitated in the zinc. The same slime was treated by a filter press plant, and only gave extraction of 61 per cent of the gold; this was owing to the nature of the slime as it contained so much ferric hydrate, and it was impossible to wash out the dissolved gold from the pressed slime.

The cost of erecting a plant according to my plan would be very much less than any other that I know of, as it merely requires the ordinary conical vats, pipes for air-lift pumps, zinc boxes, and an air compressor. The mechanical efficiency of an air-lift pump is quite equal to other pumps for slime pulp as the proportion submerged is very large in proportion to the actual lift required. Very little labor is required to run the plant and one attendant could look after several vats.

As regards the strength of the cyanide solution and consumption of chemicals, that would depend on the nature of the ore and would be neither more nor less than that required by other methods.

My plant (see Fig. 1) consists essentially of a conical vat *A* having within a cone-shaped baffle *B* and a pipe *C*, up which a jet of air is directed from the small pipe *D*; a further pipe *E* leads from near the top of the vat *A* into the zinc precipitating box *F*, which is provided with a pipe *G*, through which a small air-lift pump *H* returns the liquor from the zinc box to the inside of the baffle *B*.

The method of operating the plant is as follows: The slime, having been collected by any suitable method and the excess of

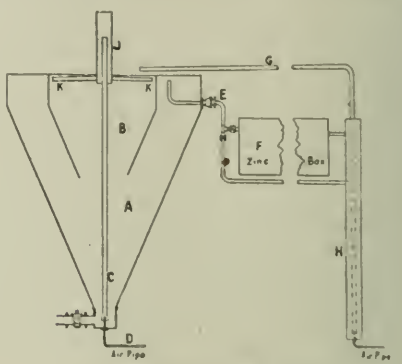


FIG. 1.—SLIME FILTRATION PLANT.

water decanted off, is washed by means of a weak cyanide solution into the treatment vat *A* through the baffle *B*. Further weak cyanide solution is then added and air turned on through the pipe *D*. This forces up through the pipe *C* into the receiver *J* a mixture of slime, dilute cyanide, and air to a point a foot or two above the level of the liquor in the vat *A*, to which it returns through the pipes *K*, which are so arranged as to give their discharge a circular motion within the baffle cone *B*.

The slime then settles down continuously, reaching the bottom of the cone in the form of a thickened pulp, and is again lifted through the pipe *C*. The pulp occupying the space between the baffle cone and the wall of the cone *A* is in a state of comparative quiescence, the only movement being a gentle flow upward of the enriched solution to the pipe *E* on its way to the zinc box. The conical shape of the vat *A* reduces the rate of upward flow and insures that the top few inches of liquor are clear and free from slime, and thus are in a fit state for precipitation. The solution deposits its gold content in the zinc box and is then returned by the pump *H* to the inside of the baffle, where it continues its work, and the sequence above described is allowed to proceed until a large proportion of the gold has been precipitated.

Having described my slime process, I would now like to go into further details.

I have made a considerable number of experiments as to replacement of the solution when the pulp is being agitated in the conical vat as described. The method I used was to have a weak chloride of sodium solution and no cyanide. I found that with the vat constructed as shown in the diagram and having a pulp containing 40 per cent of dry slime, or 1/5, I could obtain a clear overflow of 18 per cent of the contents of the conical vat per hour, and that when a volume of solution equal to the solution in the vat had been replaced by water running into the

* A paper presented at the annual general meeting held in November, 1910, of Johannesburg Transvaal Institute of the Chemical, Metallurgical and Mining Society of South Africa.

inner cone *B* that 75 per cent of the chloride of sodium had been removed. I obtained this result several times with different slimes, both in London and in the Transvaal. The same thing happens with gold and silver in solution, and as the solution, when it is returning from the zinc box, pours into the inner cone the gold-bearing solution is taken off by the decanting arm as described.

I want to call attention to one point. In beginning the operation with a solution not containing gold, the upper portion of the solution which begins to overflow would not obtain any gold for some time and it would be useless to pass it through the zinc box; therefore, in working this process I would recommend circulating for about two hours before allowing any overflow to take place. This is very easily arranged, as the pulp in the inner cone *B* is concentrated and has a much higher specific gravity than in the outer portion, the surface of the liquid is several inches below the level of the outer portion. By means of four or five 2-in. holes just below the surface of the outer portion the clear solution flows into *B*.

When this circulation has proceeded long enough the holes are plugged with wooden plugs and the clear overflow is allowed to pass through the zinc box and return to the inner cone free, or nearly free, of gold and silver. After this circulation has been started it will be apparent that at any given time the upper portion of the solution in the outer vat, between the inner and outer cone, is the richer in gold. Therefore, if the pulp be allowed to settle in the vat, the supernatant liquid is much richer than that portion in the settled pulp.

One can, if one likes, replace a strong cyanide solution by a very weak one. I treated some slime in London (from Nundydroog Mine, Mysore, India) by my process, and after seven hours' treatment the unwashed residue assayed 0.07 dwt. fine gold per ton. The original slime assayed 2.5 dwt. fine gold. This slime was slightly pyritic. There is even in slime an insoluble minimum of gold, and it would be an interesting problem for someone to find out in what condition this very finely divided gold is, and to explain why it is not soluble in cyanide solution.

It is quite evident that the capital outlay required for a plant which requires so short a time for treatment (the time for settlement must, of course, be added) is very much less than for the present plans in use on these fields. This plant is both easy and inexpensive to work. If the overflow is not absolutely clear, it will have to be passed through a clarifier of some kind; this method is usual to-day in most slime plants.

A plant has been erected on the above principle and has been working for 14 months on the Crown Mines, giving excellent results, which will be described in the paper which follows, so that it will be evident that my process has now passed the experimental stage, as a proof of which are appended some notes by Mr. Fraser Alexander on the "Treatment of Black Sands."

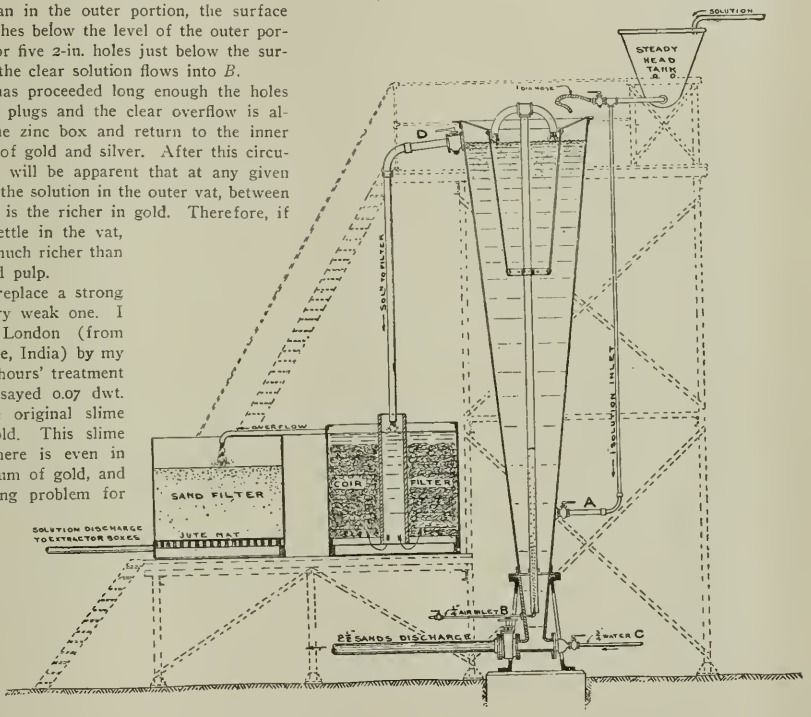


FIG. 2.—BLACK SANDS TREATMENT PLANT.

Appendum on the Treatment of Black Sands.

By Fraser Alexander.

I have been asked to explain the method of treatment and plant adopted at the Crown Mines and worked by me with considerable success since October, 1908, for the quick treatment with cyanide of our battery by-product commonly called "black sand."

After working the plant described by Mr. Burnett for a considerable period, I decided that the sampling of residues could not be relied on, and I often found amalgam accumulated immediately below the stirring gear in sufficiently large quantities to upset all calculations of actual extraction, and this was my chief objection to the paddle stirring process. Seeing a plan of Mr. Andrew F. Crosse's design for the treatment of concentrates, I decided to try the experiment on a working scale, and used for my purpose an old conical-shaped water purifier, 18 ft. deep, by 4 ft. 3 in. at the top, tapering to 2½ ft., to which I fitted air, cyanide solution, and water connections, as shown in the accompanying sectional diagram. This apparatus, which is simple in construction and costs practically nothing for maintenance, is extremely reliable and efficient for the treatment of refractory ores with cyanide.

Since installing this plant I have treated 61,340 lb. of black sands, having an average value of 23.0 oz. per ton, from which 683 oz. fine gold = 96.88 per cent and about 150 oz. of mercury were recovered. In many instances I checked the assay values of charges by sampling the solutions, from which I obtained most satisfactory results in measured quantities.

Appended is a fair specimen treatment sheet: Dry weight of charge = 500 lb. charged into the cone with ordinary working cyanide solution to which is added 15 lb. of cyanide and 10 lb. lime. Charged, 2 p. m.; circulated, 8 p. m.; started washing, 9 p. m.; rate of wash = 75 tons per hour, 3 a. m.; KCy = 0.13, CaO per cent = 0.015. Average value solution = 23.35 dwt. After further washing with weak solution (= 0.05 KCy) for six hours, the average value was reduced to 4.3 dwt. 11 a. m., solution KCy = 0.01, CaO = 0.008, Au = 0.30 dwt. Twelve noon to 1 p. m., water wash.

Original value of charge, 25.0 oz. per ton.
Value of residue = 0.51 oz. per ton.
Theoretical extraction = 97.96 per cent.

The process is practically continuous and requires very little attention.

I have no doubt that this method of treating refractory ores with cyanide is capable of considerable extension as a quick method of extracting gold and silver from their ores.

The partial analysis of a sample of treated black sands from the Crown Mines is as follows:

	Per cent.
Silica (SiO_2)	51.94
Iron, calculated as Fe_2O_3	32.62
Sulphur (S)	16.98

Assuming the sulphur to be contained as iron pyrites having a formula of FeS_2, calculation gives the following data:

	Per cent.
Silica (SiO_2)	51.94
Iron disulphide (FeS_2)	31.84
Ferric oxide (Fe_2O_3)	11.39

A qualitative investigation showed the presence of a very small quantity of arsenic, and traces of copper, titanium, cobalt, and nickel. A small amount of lime was also noted, this being a subsequent addition for treatment purposes. A negligible quantity of magnetic iron is present.

* * *

In the brief discussion which followed the above two papers and which is printed in the November issue of the *Journal of the Chemical, Metallurgical and Mining Society* of South Africa, Mr E. H Johnson said he had seen the plant in operation at the Robinson mine. The system of agitation is perfect in the central cone and gives a very clear overflow from the outer cone, but he thought that "the point in question is the amount of solution which will be required to complete the washing. In Mr Alexander's case, I do not think it is a matter of importance, owing to the high value of the original slimes."

The Late Dr. Charles B. Dudley.

It is with great regret that we record the death of Dr. Charles B. Dudley, chief chemist of the Pennsylvania Railroad, on Dec. 21, 1909, at Altoona, Pa. His loss will be keenly felt by all his colleagues in the scientific world. Not only was he very eminent in his chosen field of testing materials, but by his genial disposition, attractive personality, kindness and fairness in dealing with those associated with him, he had endeared himself to his fellow workers and members of the chemical and engineering professions. His death will be especially deplored by members of the International Association for Testing Materials, of which he was the president, his term extending until 1912.

Dr Dudley was born in Chenango County, New York, on July 14, 1842, and was educated at the local school and academy. The War of the Rebellion having broken out, he enlisted, in 1862, in the 114th New York Volunteers and took part in a number of battles, receiving a severe wound at Winchester in September, 1864. The bullet he received at this time he carried throughout his life and the effects of it may have contributed to some extent to the causes leading up to his death. He left the army in 1865, and went back to his studies, first at Oxford Academy and Institute, in Chenango County, then, from 1867 to 1871, at Yale College. On leaving the latter institution he was compelled to take up newspaper and other editorial work in order to acquire the necessary funds for continuing his education. He then spent two years at the Sheffield Scientific School of Yale University, graduating in 1874 with the degree of Ph.D. The greater part of his time during this stay at Yale University was devoted to the study of chemistry.

On leaving Yale University he acted for one year as the assistant to the professor of physics at the University of Pennsylvania, Philadelphia. He then took up the position of professor of sciences at the Riverview Military Academy, Poughkeepsie, a position which he held for a few weeks only, for, in the autumn of 1875, he received an invitation from the Pennsylvania Railroad to act as their chemist. He accepted the offer, and held this position up to the time of his death.

At the last meeting of the International Association for Testing Materials at Copenhagen, in returning thanks for his nomination as president of the association, Dr. Dudley stated that he had spent his whole life in testing, his thoughts no doubt carrying him back to the day, just 34 years previously, when he commenced the installation of his laboratory at Altoona, in which he was to carry out such useful work, not only for his company, but also for a large number of other concerns and institutions.

His work for the greater part was pioneer work; it covered in the course of a very short time the whole of the large number of materials of different kinds used in the building and the working of a railway, including both the track and the rolling stock; it led to the drawing up of complete series of standard specifications for all the classes of goods required. From the beginning he started minute investigations in the matter of quality and life of steel rails; he carried out these investigations systematically by testing for their physical and their chemical properties rails which had been in service for varying periods of time. His researches in this field, and the conclusions he arrived at, led to his introducing numerous variations in the steel-rail specifications which so far had been accepted.

Dr. Dudley was a pioneer in the investigation of the value of alloys for frictional bearings. He also made an exhaustive study of the most efficient size of bearings for railroad cars, and his experiments led to the adoption by his railroad of wide journals for the heavy cars. Dr. Dudley was elected a member of the Iron and Steel Institute in 1886, and he frequently took part in its discussions. His contributions to the proceedings of scientific associations and to the technical press, quite apart from his specifications for the Pennsylvania Railroad Company, are very numerous, and may be said to cover the whole field of railway construction and working, from the laying of the track to the adequate ventilation of passenger carriages.

Dr. Dudley was at one time president of the American Chemical Society, president of the American Society for Testing Materials and at various times he has occupied a prominent position in other organizations. He was also a member of numerous foreign technical institutions, and, as already stated, he had been four months ago elected president of the International Association for Testing Materials, a position in which by his wide experience and the universal respect in which he was held, he would have greatly forwarded the cause of the association—the unification of methods of testing. Dr Dudley's contributions in the application of chemistry to the industries will not soon be set aside nor quickly forgotten.

American Institute of Mining Engineers.

The following new members have accepted election to the American Institute of Mining Engineers:

Messrs. Donald [illegible] Rossland, B. C., Canada; [illegible] Marshall Cairns, Glasgow, Scotland; Samuel W. Cohen, Cobalt, Ontario, Canada; Raoul Gray Dufourcq, Arizpe, Sonora, Mexico; James Haskell Hoge, Salt Lake City, Utah; Frederick Charles Holmes, Santiago de Cuba, Cuba; Clarence T. Hopkins, Jerome, Ariz.; Archabald Stewart Hummel, High Bridge, N. J.; Percy Kenyon, Concepcion del Oro, Zacatecas, Mexico; Warren Lawson Klutty, Thomas, Ala.; S. Arthur Krom, Plainfield, N. J.; Henry M. Lancaster, Wallace, Idaho; Charles N. Lindley, New York City; Robert Livingstone Livingstone, Alberta, Canada; Eugene McAuliffe, Chicago, Ill.; James P. M'Carthy, Wallace, Idaho; Atpell Francis McEwen, Cobalt, Ontario, Canada; Henry A. Morris, Gowganda, Ontario, Canada; Duncan M. Munter, Huntingdonville, Jalisco, Mexico; James Negus, Bowen Mountain, New South Wales, Australia; John O. Nichols, Berkeley, Cal.; Horace L. Parsons, Garfield, Utah; John G. Petry, Denver, Colo.; Guy Hall Ruggles, Santa Rita, N. M.; Chard Olive Sanford, Globe, Ariz.; Charles H. Schlacks, San Francisco, Cal.; Gustave Schrader, Silver Creek, Cal.; Tock Kai Tse, Wickenburg, Ariz.; Rush J. Wire, Wallace, Idaho; James F. Callbreath, Jr., of Denver, Colo., was elected an associate member of the Institute.

Electrical Drive for Rolling Mills.

By Dr. A. S. McAllister.

In considering the driving of rolling mills by means of electric motors one should not lose sight of the fact that the motors are merely elements in a system the whole duty of which is to transmit energy economically from the point where it can be produced most conveniently to the places where it must be utilized to obtain the desired result. To only a very minor extent can electric motors, when substituted for other means for driving rolling mills, affect the sum total of energy actually utilized in the rolling processes or the rate at which this energy is employed. However, the electrical system, of which the motors form one part, when properly arranged for the purpose does affect in a pronounced manner the instantaneous rate at which the energy is produced, and to a limited extent the total amount of energy which must be produced. That is to say, the electrical system can conveniently make use of means for storing energy between the time when it is produced and when utilized, and it is highly efficient in the performance of its duty with reference to the transmission of energy.

The simplest method of adapting electric motors to rolling-mill service is to substitute for each steam engine previously used for the purpose a motor of equivalent maximum horsepower and install a generating equipment suitable for supplying energy to the motors. An electrical installation of this kind would possess certain advantages over the steam-engine equipment, such as efficiency in transmission, cleanliness in operation, and flexibility in control, but viewed from the electrical standpoint it is unnecessarily expensive. Each motor must be capable of exerting the maximum torque demanded of the shaft which it drives, which torque is reached for only a very small part of the cycle of operation, and hence it must be designed for a load which it will be called upon to carry only intermittently for very short periods. The significance of this statement is that each motor must be built of a size many times (from four to eight times) as large as the size of motor that would be required for carrying its average load. Moreover, the generating equipment must be designed for a large maximum load, while it will operate for a major portion of the time at only a fraction of the rated output, and hence will be excessive in first cost and not too highly efficient in operation.

A simple method for decreasing the cost of the generating equipment is to install a storage battery with regulating devices for causing the generators to charge the battery when the load on the system is light, and for allowing the battery to discharge into the system when the load is heavy. By the use of such a battery installation the generators can be designed for the average load on the system, and they will operate continuously at a high efficiency by carrying a load which fluctuates only slightly.

Direct-Current Motors.

When the older steam-engine practice is followed the rolling-mill motors must be of the reversing type and must be capable of accelerating quickly from rest to full speed. This requirement dictates the use of motors of the direct-current type, which are much superior to alternating-current motors for variable-speed and reverse-motion services. Although direct-current motors without special means for suppressing sparking have given excellent results in rolling-mill service, yet there can be no doubt that the severe duty imposed by the fluctuating torque of the rollers can best be performed by direct-current motors when they are provided with interspaced commutating poles. In fact, reversing rolling-mill service furnishes ideal conditions for the application of interpole direct-current motors.

Although the use of a flywheel directly with a reversible motor would be even detrimental to its performance, yet by supplying energy to the reversible motor through a flywheel-motor-generator set considerable advantages are obtained. The motor of the set is so designed that its speed decreases with increase of load; it could be of either the direct-current shunt-wound type or alternating-current induction type. The generator of the set is a direct-current machine, the voltage of which may be varied at will according to the demands of the reversible rolling-mill motor. The flywheel performs a useful service in storing energy from the supply system when the rolling-mill motor is idle and giving off energy to the generator of the set when the demand is greatest. Thus it permits the motor of the set to operate at a load which varies only slightly, although the load on the generator fluctuates enormously. By making the generator of the set a variable-voltage machine much of the energy which would otherwise be wasted in rheostats is saved and hence the efficiency is improved, but the size of the generator is not appreciably decreased thereby. For the purpose of preventing sparking when operating at low voltage with weakened fields and full value of armature current, the generator could be provided with interspaced commutating poles. The use of such interpoles permits the active material of either a generator or a motor to be proportioned for the mean effective load on the the machine rather than for the maximum load, as in the case when interpoles are not used.

An example of a flywheel-motor-generator installation is the equipment of the Illinois Steel Company, where use is made of a 1300-hp motor-generator set connected to a 100-ton flywheel for supplying energy to two 2000-hp reversing rolling-mill motors.

Alternating-Current Synchronous Motors.

In comparison with other motors which may be used for driving rolling mills the direct-current machine possesses the advantage of flexibility in speed control and the ability to receive energy directly from a storage battery. It is disadvantageous in possessing a commutator which requires much attention, and is limited in voltage to a value much below the e.m.f. at which energy can be transmitted economically over any considerable distance. So far as concerns transmission the synchronous alternating-current motor might be arranged to give satisfactory service, but its absolute constancy in speed ratio with the generator eliminates it from consideration where flywheels are to be used for energy storage.

Polyphase Induction Motors.

For the direct driving of non-reversing rolling mills no motor is better adapted than the polyphase induction machine. This motor has no commutator and hence cannot spark. Its torque speed characteristic is identical with that of the direct-current shunt-wound motor, for which it can be directly substituted. In comparison with this motor it is advantageous in being free from commutator troubles and not restricted to low voltages, and disadvantageous in that it cannot receive energy from a storage battery, and in its simplest form is limited to operation at practically a single speed. The disadvantages are of relatively minor importance and have been minimized by the methods outlined below.

When built with a constant secondary resistance—that is, with a squirrel-cage rotor—the polyphase induction motor operates in all respects like a shunt-wound direct-current motor with a constant resistance in its armature circuit. The machine tends at all times to accelerate to a certain definite speed, and under load runs below this speed by an amount depending directly upon the torque which it must exert. Like the shunt-wound motor it can exert its full-load torque at any speed below the normal full-load value, provided the resistance of the rotor circuit is correspondingly increased. Moreover, a combination of two induction motors in series arrangement can be used for obtaining lower speeds just as the armatures of the direct-current motors can be joined in series for half-speed operation. When thus employed the motors are said to be connected in concatenation. An example of the concatenated method of speed control of two induction motors is found at the rolling mill of the Edgar Thomson Steel Company.

So far as the operation of the motor itself is concerned best results are obtained when a polyphase induction machine is

arranged for operation at a practically constant speed without variation in its secondary resistance. Under these conditions the motor could be built with a squirrel-cage rotor, which may well be considered immune from all circuit troubles. Such a motor provided with a flywheel of such a size as to store and give off sufficient energy during each rolling cycle forms an ideally simple machine for the direct driving of a non-reversing rolling mill. Within all operating limits, the lower the resistance of the squirrel cage secondary the smaller may be the flywheel and the more efficient the operation of the motor, but the greater the fluctuations in the load on the supply system. The least load fluctuations are obtained with the heaviest flywheel and with a high value of secondary resistance—not, however, the highest possible value.

In any event the power demanded by the motor from the supply system must fluctuate, because the motor must vary its speed as it accelerates the flywheel in storing energy or allows the flywheel to decrease its speed in giving off energy to the rolls. The mean speed is such that the motor takes from the supply system that average value of power which, multiplied by the time of each cycle, represents the cyclic rolling energy plus the various losses; the fluctuation in speed is such that the flywheel delivers to the rolls the energy to represent all instantaneous excess of demand for power above the value given by the motor and absorbs energy when the motor power is greater than that being used by the rolls.

A flywheel designed to deliver to the rolls the total of the energy represented by all power above the mean value, while decreasing its speed from the mean value to the minimum permissible value, would more than fill the requirements by reason of the inevitable fluctuations in the power of the motor. By sacrificing some of the simplicity of the squirrel-cage rotor, it is possible to reduce materially the fluctuations in the power taken by the induction motor from the supply system. For this purpose the motor is provided with a coil-wound rotor equipped with slip rings so that the resistance in the secondary circuit may be varied as desired. The power taken by the motor could be kept absolutely constant by increasing the secondary resistance in direct proportion to the "slip" of the rotor from its no-load (synchronous) speed, the fluctuations noted above could even be reversed by subjecting the secondary resistance to variation greater than here assumed. The writer is not aware that the constant load scheme has ever been employed in rolling mills. However, the variable-resistance scheme is used at the Gary, Ind., plant of the Illinois Steel Company, where a storage battery serves to minimize the fluctuations in the load on the generators.

Electrochemical Energy Storage.

Mention has already been made of the fact that in comparison with the alternating current motor, the direct-current motor possesses an advantage in being capable of utilizing energy directly from a storage battery. At the Gary plant any disadvantage under which the induction motor would labor in this respect has been compensated for by the installation of special rotary converters as connecting links between the alternating-current supply leads and the storage battery. By means of these converters, which are of the split-pole variable-ratio type, the load on the alternating current generators is kept as nearly constant as desired in spite of the violent fluctuations in the load on the alternating current system. One of the most interesting features of the storage battery installations is a variable-voltage generator used for changing the excitation of the converters in accordance with the fluctuation in the load on the alternating-current system. This exciter can be adjusted to maintain the load on the generators at any desired mean value with the fluctuation minimized to any desired degree.

Relative Advantages of Electrochemical and Mechanical Energy Storage.

It will be observed that between the rolls at which the energy is utilized and the generators where it is produced, use is made of two devices for permitting the energy to be generated at a constant rate although utilized with extreme fluctuations in rate, namely, the flywheel and the storage battery. Although performing similar services, and thus assisting each other in producing the desired result, the flywheel and the storage battery possess distinctive characteristics with reference to the transfer of energy which preclude the one being substituted for the other or used exclusively.

The chief advantage of the flywheel resides in its ability to deliver its stored energy at practically any rate demanded. For example, a 50-hp motor-flywheel set could readily deliver energy at the instantaneous rate of 50,000 hp for a limited period, while a storage battery capable of supplying energy at this rate would have to be designed for about 12,500 kw hours; however the battery could deliver this same amount of power for a number of minutes. The significance of this statement is that energy stored in a flywheel is available for instantaneous use, while considerable time is required for extracting energy from a storage battery.

Where a limited amount of energy is to be used for a short time at a very high rate the flywheel is much less expensive and more satisfactory than the storage battery, but where a considerable amount of energy is to be stored for use at a relatively low rate (represented by cyclic intervals of several seconds) the battery is far preferable to the flywheel.

It is to be noted in this connection that the storage battery assists only the generating equipment while the flywheel is of assistance to the whole electrical installation; by its use the motors, transmission circuits, generators, and prime movers may be built for the average rather than the maximum load and operated at the best efficiency. In electrically driven rolling mills the flywheel may well be considered a necessary device. For the best service it should be used on the motor driving the rolls rather than on a generator set supplying energy to the rolling mill motor.

New Uses of Liquid Air and Oxygen.

Some interesting industrial applications of liquid air and of oxygen made from liquid air were noticed in a recent paper read by Mr. George Claude on November 10, 1909, before the Society of Civil Engineers of France.

Commercial plants for the production of liquid air by the Claude process or by the Linde process exist in various European countries (there is also quite a large one in this country using the Linde process, in Buffalo, N. Y.) Their chief product is compressed oxygen, and the principal applications of the oxygen are autogenous welding with the oxy-acetylene flame, cutting of metals with the oxy-oxy-acetylene flame, and the manufacture of artificial precious stones.

But the price of oxygen thus produced is still comparatively high, as Mr. Claude points out, on account of the large expense for compressing and bottling the oxygen gas in cylinders and the shipping expenses. The cost of the oxygen gas would be greatly reduced if it was required in large quantities on the spot. An essential item in the cost is represented by the pressure to which the air under treatment must be compressed. This pressure is given by Claude as 18 atmospheres for apparatus producing 100 cu m (3530 cu ft) of oxygen per hour and is estimated as 10 atmospheres for 1000 cu m (35,300 cu ft).

At present 1 cu m (35.3 cu ft) of oxygen is obtained per hp-hour in apparatus producing 50 cu m (1765 cu ft) per hour. The figure of 1.25 cu m (44 cu ft.) of oxygen per hp-hour is guaranteed by the Société de l'Air Liquide for apparatus producing 200 cu m (7060 cu ft) per hour, and the figure of 1.75 cu m (62 cu ft) of oxygen per hp-hour represents the hopes of the manufacturers for apparatus producing 1000 cu m (35,300 cu ft) per hour.

A plant for the yearly production of 9,000,000 cu m (1,765,000,000 cu ft.) of pure oxygen requires not more than 20 m by 30 m (65 ft by 98 ft) floor space and 15 attendants.

For a plant, to be erected with apparatus producing 200 cu. m (7060 cu. ft.) of oxygen per hour, Mr. Claude's company has guaranteed that the cost of producing 1 cu. m (35.3 cu. ft.) of oxygen, including amortization, will be slightly more than 0.6 cent; that is, figured on the basis of the cost of 0.4 cent per hp-hour (an exceedingly low price which is thought to be justified because blast-furnace gases are used in driving the gas engines for power production at that place).

A very important and interesting experiment on a large scale is soon to be made by a Belgian iron and steel plant, the Société d'Ougrée Marihaye, with respect to the enrichment of the air with oxygen for the blast in one of their blast furnaces and converters. (We comment on this development in our editorial pages.)

At the conclusion of his lecture Mr. Claude showed a number of experiments, like the liquefaction of oxygen in contact with liquid nitrogen; the condensation of atmospheric air on the outside of an iron bottle containing liquid nitrogen; further the fact that coke maintained at the temperature of liquid air absorbs gases with such energy that it is possible to realize rapidly a vacuum of less than one-ten-thousandth of a millimeter. Mr. Claude also subjected a current of illuminating gas fed to a burner, to the action of liquid air; the result was a modification of the flame which assumed the characteristic appearance of the hydrogen flame. This experiment indicates the possibility of a cheap process for making hydrogen by partial liquefaction of water-gas. Other experiments illustrated the ease of recovery of volatile substances in chemical industries by refrigeration, such as ether and alcohol in the manufacture of artificial silk, and the facility of extracting neon and helium from the atmosphere.

* * *

In this connection it is interesting to report a lecture delivered November 18, 1909, before the Polytechnic Association of Berlin, by Dr. H. ERDMANN on the technical application of liquid oxygen. (*Chemiker Zeitung.*)

Dr. Erdmann first discussed briefly the chemical properties of the liquid gas and also described a modified arrangement for investigation of its purity by means of the aerometer. It is possible to utilize liquid oxygen at the same time for three purposes, namely, to utilize its cooling effect, the pressure, and its chemical energy.

The consumption of liquid oxygen is much greater than is generally supposed, although we have no accurate statistics regarding it. Three methods of manufacture are in use. By the Brin method barium oxide (BaO) is heated in a current of atmospheric air to form barium peroxide (BaO_2). The air current is then stopped, and the barium peroxide reheated in a partial vacuum, whereby oxygen is given off. By the second, the electrolytic method, the gas is obtained from water, but this is economical only when there is use at the same time for the hydrogen generated simultaneously with the oxygen in equivalent proportion. Electrolytic cells for this purpose have been devised by Schuckert, Garuti, Schmidt and others. The third method, based on the fractional distillation of liquid air, is now very largely employed in Europe. For operation on a large scale the cascade principle, which was first used by Pictet in 1877, is used to advantage. The lecturer mentioned briefly the processes of Cailletet, Kamerlingh-Onnes, Claude, Hildebrand, and Linde.

Altogether there is annually produced in Germany 8000 cu. m. (282,500 cu. ft.) of oxygen (meaning evidently oxygen gas at atmospheric pressure) by the Brin method, 342,000 cu. m. (12,000,000 cu. ft.) by the electrolytic method, and 2,200,000 cu. m (77,700,000 cu. ft.) by the fractional distillation of liquid air. The average cost for 1 kg of fluid oxygen amounts to about 10 pfennigs.

The oldest use found for oxygen is in explosives, oxygen cartridges not being dangerous. In recent years oxygen has found use in the production of light. In this connection was mentioned the Nürnberg light, which it has been found consumes very little illuminating gas and produces less heat than the ordinary gas lamp. It is also more hygienic since the oxygen is supplied to the flame and is not, therefore, taken from the air, thus contaminating the latter. In metallurgy oxygen finds many uses; for instance, in the oxy-hydrogen and oxy-acetylene flame for autogenous welding and cutting of metals. Oxygen is also used in the production of fused quartz (silica) vessels. Oxygen used in medicine must have the highest purity, this accounts for the higher price. According to Michaelis, oxygen, which is 96-97 per cent pure, costs 6 marks ($1.50) per cubic meter, while 99 per cent oxygen costs 10 marks ($2.50).

Platinum Thermometry at High Temperatures.

The Bureau of Standards has already produced quite a number of excellent researches on pyrometry, which are not only of a high scientific standard, but of great practical importance. Recently a long investigation carried out by Dr. C. W. WAIDNER and Dr. G. K. BURGESS, at the Bureau of Standards, on platinum resistance thermometers at high temperatures has been concluded. The general conclusions are as follows:

First with respect to temperature scales. Temperatures defined by the resistance thermometer of pure platinum calibrated by Callendar's formulæ at 0°, 100° and 444.7° (the boiling point of sulphur), agree with the temperatures on the generally accepted gas scale in the range 0° C. to 1100° C. to within the degree of reproducibility of the latter.

Thermometers made of impure platinum, and calibrated in the same way, do not define the same temperature scale as that of pure platinum, the divergence increasing with the impurity of the metal.

If thermometers of impure platinum are calibrated at a fourth temperature, such as the silver freezing point, and the difference coefficient of the Callendar formula is written $\delta = a + bt$, then the scale so defined by these thermometers is brought into very close agreement with that of the thermometers of pure platinum.

The Dickson formula $(R + a)^2 = p(t + b)$ is not applicable to either platinum or palladium thermometers at high temperatures.

The palladium thermometer, calibrated by the Callendar method, gives temperatures agreeing to 1° with the scale of the platinum thermometer up to 600°. It requires a fourth degree equation to express its resistance-temperature relation from 0° to 1100°.

The following table of freezing points represents the scale of the resistance thermometer of pure platinum as determined from the investigation, using only the purest metals:

	Metal									
	Sn.	Cd.	Pb.	Zn.	Sb.	Al.	Ag_2Cu_2.	Ag.	Cu.	$Cu\text{-}Cu_2O$.
Freezing point	231.92°	321.01	327.43	419.37	630.71	658.0	779.20	960.9	1083.0	1063.2
Average deviation for purest metal	.02	.02	.10	.02	.30	.5	.25	.20	.5	.3
Reproducibility factor	.06	.26	.10	.06	2.3	1.2	...	...	1.0	...

The average deviation of a single observation is a measure of the precision attained when two or more thermometers were used in a single metal sample. The reproducibility factor is the average deviation of the metals, designated as c.p., or better, from their mean F.P.

Thermocouples of Pt, 90Pt — 10Rh and Pt, 90Pt — 10Ir, calibrated at three temperatures in terms of the formula $E = a + bt + ct^2$, define a temperature scale in the interval 200° to 1100°, which is in agreement with the platinum-resistance scale at intermediate temperatures to within 0.3°.

As to the behavior of platinum thermometers the authors

conclude that when used at high temperatures the constants of platinum thermometers undergo gradual changes, which necessitate frequent recalibration in work of high precision. The use of thermocouples for work above the F. P. of silver is, therefore, much less laborious.

These changes in thermometric constants are greatly reduced, but not completely eliminated, by annealing the thermometers for at least several hours at a temperature higher than that at which they are subsequently to be used.

For thermometers of pure platinum, the changes in their constants after the thermometers have been annealed, are very much less than for those of impure platinum.

These changes are least for pure platinum wires of large diameter and suspended free from strains.

For impure platinum wire the effect of high temperatures is such as if the wire became purer, possibly because of the evaporation of impurities, for example, iridium. If the platinum is pure, the slight changes indicate a contamination of the wire and the effect of strains. The total change observed is a resultant of the effects of strains of annealing and of contamination and purification.

Calibration Temperatures.

As to calibration temperatures the authors state that the freezing points of the pure metals are the most satisfactory calibration temperatures, being superior to boiling points in constancy, ease of manipulation, and reproducibility. Unless a very slow rate of heating is used, melting points are less satisfactorily fixed and apparently slightly higher than freezing points.

The temperature of the separation of the silver-copper eutectic is an entirely satisfactory fixed point at 779.2°, when the alloy is of the eutectic composition, or about 72 Ag — 28 Cu. The eutectic point of copper-cuprous oxide is also a satisfactory calibration temperature at 1063.2°, when proper experimental precautions are taken to maintain the eutectic composition.

Boiling Point of Sulphur.

The authors also made an extended investigation of the boiling point of sulphur. They find the temperature defined by the vapor of sulphur boiling at known pressure and under carefully specified conditions is a most satisfactory fixed point, reproducible to a few hundredths of a degree. These conditions are, briefly:

(a) Sulphur to be boiled in a glass tube 4 cm to 5 cm diameter and of sufficient length to insure ample depth of immersion in the vapor (tube about 40 cm satisfactory).

(b) Sufficient depth of liquid sulphur to avoid superheating.

A criterion for the above conditions is absence of change in the observed resistance when the thermometer with its attached radiation shields is moved some cm up and down the tube.

(c) Surrounding coil of thermometer with a suitable radiation shield to avoid the cooling effects of the condensed sulphur running down the stem of the thermometer and of radiation to the cooler parts of the apparatus.

Both gas and electrically heated S.B.P. tubes were used; the former gave a temperature 0.03° or 0.04° higher than the latter.

The determination of the S.B.P. by different observers is discussed at length. The determinations of these observers are in satisfactory agreement, the values found ranging, with one exception, from 444.5° to 444.7°. The bearing of some recent work, with the gas thermometer on the value of the S.B.P. is considered, which shows that the above value may be too high. The tendency, however, is not sufficient as yet to warrant any change in the generally accepted and widely used value of the S.B.P.

For details of this interesting investigation the reader is referred to the very long and full report of Messrs. Waidner and Burgess in No. 2 of Vol. VI (1909) of the *Bulletin* of the Bureau of Standards.

Natural Gas Production.

According to a chapter of "Mineral Resources, 1908," by Mr. B. Hill, just issued by the Geological Survey, the approximate value of the natural gas produced from wells and consumed in the United States in 1908 was $54,640,374, being greater than that of any previous year. There was a slight decline in 1908 in the Eastern States, as compared with 1907, but this loss was more than made up by substantial gains in the States of the Middle West and South. A further examination of the tables for the last three years shows at least three interesting tendencies: An increasing quantity of gas used for domestic purposes, a decreasing quantity consumed for industrial purposes, and an increased price for the gas.

The production of crude petroleum in 1908 is valued at $120,106,749. Taken with the natural gas this made a combined annual valuation of $174,329,148 for the two products. An examination of the table of production covering the period from 1882 to 1908 shows that the output of natural gas rose rapidly from a value of $215,000 in 1892 to $22,620,873 in 1888, fell gradually in the next eight years to $13,002,512 in 1896, a little more than doubled in five years to $27,066,077 in 1901, a little more than doubled again in the next six years to $54,222,309 in 1907, and, in spite of the financial difficulties beginning in 1907, increased slightly in 1908. This remarkable record seems to indicate that the producton of natural gas has become one of the great established industries of the country, an indication borne out by the extension of the gas districts in the Appalachian field, by the development of the mid-Continental and Gulf fields, which has practically only begun, and by the systematic efforts being made to conserve the consumption of this most useful, convenient and luxurious fuel.

The total number of natural-gas wells productive December 31, 1908, was 21,375. During the year there were 1241 wells abandoned and 2894 drilled. This latter number of wells is itemized as 2148 gas and 746 dry.

During 1908 natural gas to the value of $22,003 was imported for consumption. No exports were recorded. The report shows that Pennsylvania not only leads all States in the value of its gas production in 1908, but also in the combined value of its natural gas and petroleum production. The statistics of the natural gas industry, as given in the table of distribution, show that the value of the gas consumed in this State in 1908 was $20,678,161, as compared with $22,917,547 in 1907, a decrease of $2,239,386. This decrease was due to a falling off in consumption, the quantity decreasing from 164,541,170,000 cu. ft. in 1907 to 147,790,097,000 cu. ft. in 1908. The average price per 1000 cu. ft. was 13.99 cents in 1908, as against 13.9 cents in 1907, a slight advance. The greatest decline was in gas utilized for industrial purposes, the quantity and value decreasing from 119,700,431,000 cu. ft., valued at $12,070,625, in 1907, to 105,587,229,000 cu. ft. valued at $10,536,644, in 1908, this decline being partly due to the lessened demand for gas as a result of the general depression in business which so largely affected the manufacturing districts of this State in 1908. In some instances wells owned by manufacturing establishments which were idle were closed all the year. In the domestic consumption of gas in 1908 there was a decline of 2,037,886,000 cu. ft. in quantity and of $705,405 in value as compared with 1907.

The preliminary report of the Canadian Department of Mines places the value of Canada's natural gas production at $1,012,060. In 1907 it was worth $815,032.

Iron and Steel Institute.—The annual general meeting of the British Iron and Steel Institute will be held at the Institution of Civil Engineers, Great George Street, London, S. W., England on Wednesday and Thursday, May 4 and 5, 1910. The annual dinner will be held, under the presidency of the Duke of Devonshire, in the Grand Hall of the Hotel Cecil on Wednesday, May 4. The autumn meeting will be held at Buxton on Sept. 27, 28 and 29, 1910.

The Magnetic and Electrical Properties of the Iron Copper Alloys.

BY CHARLES F. BURGESS AND JAMES ASTON.

In connection with an extended investigation of iron alloys, carried out at the Chemical Engineering Laboratories of the University of Wisconsin, under a grant of the Carnegie Institution of Washington, a series of tests were made upon the iron-copper group. The results of various physical tests, including hardness, forgeability, and tensile strength, were described at the October, 1909, meeting of the American Electrochemical Society,[1] and were reprinted in the *Electrochemical and Metallurgical Industry* of December, 1909.

This paper has to deal with the magnetic and electrical properties of the same iron-copper series and is, therefore, supplementary to the one mentioned above.

The details of the preparation of the alloys and of the test samples were given in the previous article, and need not be dwelt upon to any extent here. Electrolytic iron formed the basis of the alloys, together with alloying materials of a corresponding degree of purity; therefore, any properties noted in the tests may be considered as due to the effect of the added elements alone.

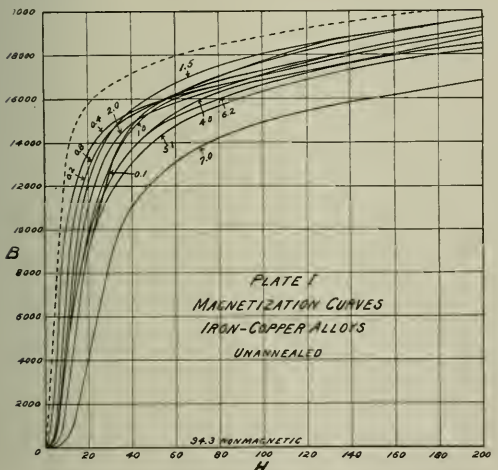


FIG. 1.—MAGNETIZATION CURVES, UNANNEALED.

Tests were made upon 24 binary alloys of iron and copper, and upon 21 alloys with copper-silicon, copper-aluminium, copper-tin and copper-arsenic.

Of the binary alloys the following have been analyzed, and form the basis of the discussion of this series.

	Per Cent Copper.	
Bar.	Added.	Analysis.
147B	0.1	0.089
158A	0.2	0.202
158B	0.4	0.422
158D	0.8	0.804
147H	1.0	1.006
147J	1.5	1.510
158G	2.0	2.005
158I	4.0	3.990
158J	5.0	5.070
158K	6.0	6.160
147U	7.0	7.050
86F	95.0	94.340

The metals mixed in precisely the added proportions. Ingots with all percentages of copper between 7 and 95 per cent were made, but it was impossible to forge this intermediate series until the copper content reached 90 per cent.

[1]Physical Properties of Iron-Copper Alloys, by C. F. Burgess and James Aston. Trans. Am. Elec. Soc. Vol. XVI, 1909.

Magnetic Tests.

The magnetic tests were made upon an Esterline permeameter under four different heat treatments—first, as forged; second, after heating to 675° C. and slowly cooling; third, after heating to 1000° C. and slowly cooling; fourth, after heating to 900° C. and quenching in water. The results are noted in Tables I and II and in Plates I, II, III and IV. All tests are compared to standard bar of electrolytic iron of high quality; the curve for this bar is indicated in the plates by a dotted line. In the unannealed condition the materials fall into two groups,

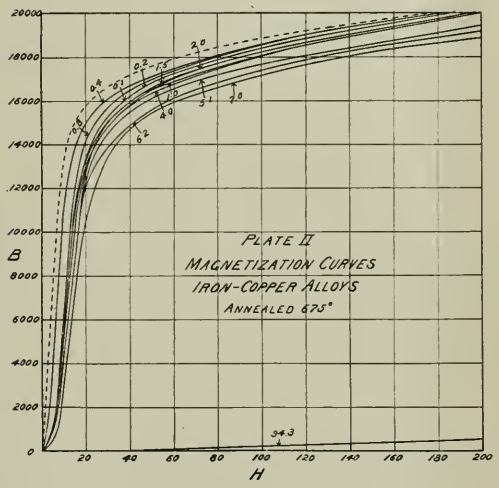


FIG. 2.—MAGNETIZATION CURVES, ANNEALED 675°.

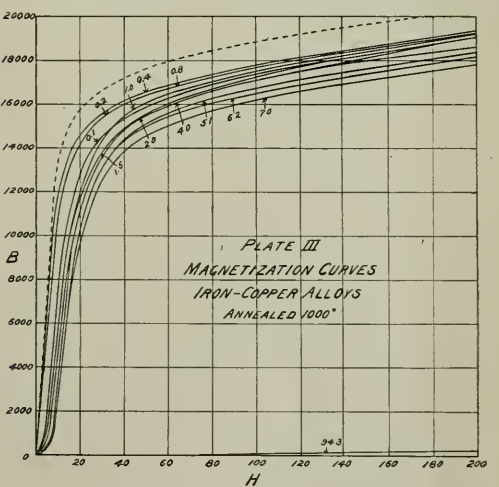


FIG. 3.—MAGNETIZATION CURVES, ANNEALED 1000°.

with those of 1.5 per cent and less irregularly mixed and of the higher magnetic quality; above 2 per cent the permeability falls off rather quickly as the copper content is increased. This is in line with the hardness tests noted in the previous paper. None of the entire series, however, is of a quality worthy of mention; on the other hand, only bars of 4 per cent and up-

warps if copper be conspicuously poor. Bar 86F, with 94.34 per cent of copper, is non-magnetic.

Annealing at 675° has greatly improved the quality of all of the bars, and the curves now fall with considerable regularity in positions indicating a decrease of permeability with increase of copper content. The two best materials are bars 158B with 0.422 per cent Cu, and 158D, with 0.804 per cent copper, the former is but little inferior to the standard electrolytic curve.

Annealing at 1000° has not changed the relative positions of the curves, but has had the effect very generally observed in all of our work of decreasing the densities reached in the upper ranges. For the lower values of H (below 10) there has been an increase of permeability for those alloys of a copper content of less than 1 per cent, but for a percentage of one and upwards there is a deterioration for all values of H.

Quenching at 900° shows practically no change in quality from that of the previous treatment, the annealing at 1000°.

In all conditions subsequent to the unannealed state it has been possible to induce a small magnetic flux in bar 86F, with 94.34 per cent Cu; but the amount is too slight to warrant one in classing this material as other than non-magnetic.

No specific deductions can be drawn from the retentivity which were not analyzed give figures somewhat at variance with the results discussed above, in that three of the bars were of very high magnetic permeability

Table II.—Coercive Force and Retentivity.

Bar	Per Cent Copper	Coercive Force H (Max.) = 200 Unann.	Ann. 675°	Ann. 1000°	Quench 900°	Retentivity H (Max.) = 200 Unann.	Ann. 675°	Ann. 1000°	Quench 900°
117A	....	5.5	6.2	3.1	2.5	12,300	13,800	10,000	8,000
147B	0.09	13.0	8.7	12.1	4.3	12,700	12,000	10,100	8,400
158A	0.20	7.0	9.3	4.0	3.5	11,100	13,300	8,400	8,400
158B	0.42	7.0	6.0	3.7	3.6	9,900	13,300	8,600	9,300
158D	0.80	9.5	9.5	4.1	4.3	11,500	14,900	8,700	9,100
147H	1.01	14.0	9.5	8.8	5.4	11,100	14,500	10,400	8,700
147J	1.51	12.8	10.5	11.3	7.3	11,100	14,300	12,400	12,500
158G	2.00	10.7	10.0	9.0	8.3	10,200	12,800	10,200	9,900
158I	3.99	6.0	9.8	9.3	9.7	9,200	13,200	11,100	11,300
158J	5.07	5.7	10.9	11.5	11.2	8,300	13,200	11,500	11,700
158K	6.16	12.5	10.5	11.8	11.8	9,300	11,600	11,800	12,100
147U	7.05	12.5	13.2	12.5	12.6	7,700	12,500	13,100	11,200
86F	94.34	Non.	200+	200+	200+	Non. Mag.	500	300	400

Among the Cu-Si, Cu-Al, Cu-Sn and Cu-As alloys nothing of high quality was indicated, except for four bars where the percentage of added elements is low.

Table I.—Data of Magnetic Tests.

Bar	Per Cent Copper	H = 10 Unann.	Ann. 675°	Ann. 1000°	Quench 900°	H = 20 Unann.	Ann. 675°	Ann. 1000°	Quench 900°	H = 50 Unann.	Ann. 675°	Ann. 1000°	Quench 900°	H = 100 Unann.	Ann. 675°	Ann. 1000°	Quench 900°
117A	C	13,100	10,000	13,350	11,800	15,700	15,950	15,450	14,600	17,600	17,700	17,050	16,950	18,850	18,850	18,200	18,150
147B	0.09	2,900	5,700	7,600	9,200	8,400	13,250	12,800	12,600	15,650	16,800	15,850	16,200	17,600	18,350	17,400	17,600
158A	0.20	7,700	5,700	10,700	10,300	12,800	13,250	14,050	13,650	16,150	16,800	16,350	16,350	18,100	18,350	17,850	17,700
158B	0.42	10,150	10,950	11,700	11,500	13,500	14,950	14,550	14,300	15,800	17,100	16,550	16,650	17,100	18,500	17,700	17,800
158D	0.80	7,700	6,300	10,700	11,000	12,800	14,000	14,150	14,400	15,700	16,950	16,400	16,550	17,150	18,...	17,850	18,800
147H	1.01	2,900	6,300	5,500	9,200	9,600	13,250	11,950	13,450	15,450	16,500	16,000	16,350	17,100	18,050	17,450	17,650
147J	1.51	5,600	4,300	3,500	8,200	12,100	12,650	11,050	13,850	16,150	16,600	15,550	16,200	18,100	18,200	17,250	17,400
158G	2.00	4,500	5,000	5,600	5,600	10,900	12,650	11,600	10,750	15,400	16,450	15,450	15,250	17,050	18,250	17,100	17,200
158I	3.99	4,500	5,700	4,900	4,100	9,600	12,900	11,050	10,750	14,550	16,350	15,450	15,650	16,800	18,050	17,050	17,400
158J	5.07	2,900	4,300	3,000	2,950	8,900	12,350	10,150	10,300	14,000	16,100	15,050	15,300	16,250	17,650	16,700	17,000
158K	6.16	4,500	4,300	2,200	2,400	9,600	12,000	9,600	9,600	14,550	15,550	14,550	14,600	16,550	17,400	16,200	16,350
147U	7.05	400	3,000	2,400	2,950	4,200	10,400	10,150	8,400	12,150	15,450	14,950	14,200	14,900	17,200	16,500	16,150
86F	94.34	0	0	0	0	0	0	0	0	0	200	100	100	0	50	0	0

values of Table II. The values of coercive force follow a general tendency to diminish with the successive treatments, although in a few minutes the reverse is conspicuously evident. The values are at no time low, except for those alloys

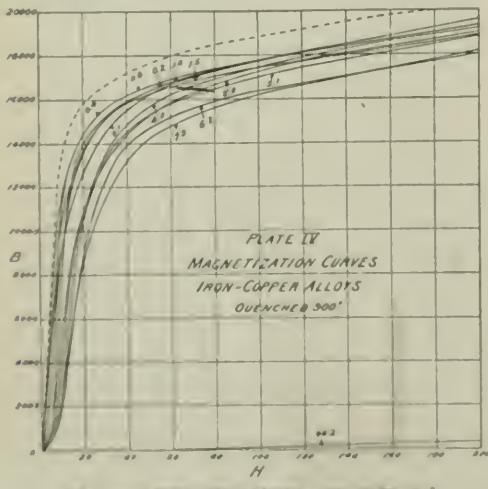


Fig. 4.—Magnetization Curves Quenched 900°

of less than 1 per cent copper after heating to the higher temperatures.

The results of the latter tests on the 12 binary copper bars

In the light of previous deductions, and because of the absence of accurate analyses, no conclusions will be drawn regarding these high values. They may be the result of high purity of the electrolytic iron and the absence of the intended content of the alloying elements.

Summarizing, it may be said there is no advantage to be gained in adding copper to iron if a good magnetic permeability is wanted. The quality deteriorates almost in proportion to the added copper content. On the other hand, it may be said that there is nothing to be feared from the effect upon the magnetic quality of the small percentages of copper ordinarily found in commercial materials.

In our previous discussions of the physical properties of the iron-copper alloys, stress was laid upon the fact that the alloys

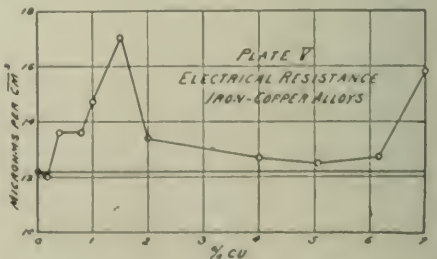


Fig. 5.—Electrical Resistance

with copper content of from 1 to 2 per cent were entirely comparable with the iron-nickel alloys with percentages of nickel from three to four, which is the range ordinarily adopted in

commercial practice. Direct comparison of the magnetic properties of the above-mentioned ranges is not possible, since we do not have the necessary data for the nickel series. However, it is worthy of mention that the magnetic data of the iron-copper alloys of the above-noted percentages is very similar to that for iron-nickel alloys of 1.93 and 7.05 per cent nickel, and they may, therefore, be taken as equal in magnetic quality to the nickel steels. Again, inspection of the magnetization curves, particularly after annealing at 675° C., shows permeabilities for the alloys of 1 to 2 per cent copper sufficiently high for commercial purposes, should their use be desired because of the increased tensile strength.

Electrical Conductivity.

Results of measurements of electrical resistance are given in the following table and in Plate V. The data are given in microhms per cubic centimeter; also in terms of the relative resistance compared to the same standard bar of electrolytic iron used in the magnetic tests. The bars had all been quenched at 900° C.:

ELECTRICAL RESISTANCE.

Bar.	Per Cent Cu.	Microhms Per Cu. CM.	Relative Resistance.
147B	0.09	12.2	1.01
158A	0.20	12.0	0.99
158B	0.42	13.6	1.12
158D	0.80	13.6	1.12
147H	1.01	14.7	1.22
147J	1.51	17.0	1.41
158G	2.01	13.4	1.11
158I	3.99	12.7	1.05
158J	5.07	12.5	1.03
158K	6.16	12.7	1.05
147U	7.05	15.8	1.31
86F	94.34	3.9	0.24

The conductivity is in no case above that of the standard iron bar, except for bar 86F, with 94.34 per cent Cu. Here the resistance is only one-fourth that of the iron; however, this alloy really belongs at the copper end of the series.

The resistance rises rapidly to a maximum of 17 microhms per cubic centimeter, at a copper content of 1.51 per cent. Between 2 and 6 per cent copper the conductivity is but little inferior to that of the iron, while at 7 per cent the resistance has again risen to 1.31 times that of the standard.

Later measurements upon numerous unanalyzed iron-copper alloys give results so irregularly scattered in relation to those here given that they should be accepted with caution.

CHEMICAL ENGINEERING LABORATORIES,
UNIVERSITY OF WISCONSIN.

The Failure of a Plant for Making Cyanide by the Fixation of Atmospheric Nitrogen.

Records of failures are as interesting and instructive as records of successes, but they remain usually unwritten. An example from the rule is an article by Mr. WILLIAM BETTEL in the *South African Mining Journal*, August 7, 1909, page 640, and reprinted in the *Journal* of the Chemical, Metallurgical and Mining Society of South Africa, November, 1909, page 187. Mr. Bettel writes as follows:

Spasmodic attempts at the manufacture of cyanide, based upon reactions, discovered over 60 years ago, for the fixation of atmospheric nitrogen as an alkaline—or alkaline earthy—cyanide have from time to time been made with more or less success, but I do not think that a description of a plant which was in use in the early nineties, but subsequently closed down owing to its unpayable nature, has hitherto appeared in print.

The works I refer to were in Scotland. A coal-gas works was obtained, suitable additions were made, the process being carried out as follows: Charcoal, selected for its low ash contents and even porosity, was soaked in a saturated solution of potassium carbonate. The charcoal was drained and dried, charged into gas retorts and dry nitrogen (from the air), deprived of oxygen as completely as possible, was passed over the heated charcoal ($+ K_2CO_3$) until the absorption of nitrogen slackened. The charges were then "drawn" out of contact with air, being cooled by the incoming dry nitrogen, and fresh charges introduced. On the average about 25 per cent of the potassium salt was converted into cyanide by the following reaction: $K_2CO_3 + 2\,C_2 + N_2 = 2\,KCN + 3\,CO$. At the same time a certain amount of metallic potassium was formed by the reduction of the carbonate with charcoal: $K_2CO_3 + C_2 + 3\,CO$. The carbon monoxide under certain undefined conditions formed a carbonyl (potassium carbonyl) $K_4(CO)_4$, which has explosive properties.

The crude cyanide so produced was subjected to systematic lixiviation and the saturated solution of $K_2CO_3 + KCN$ decomposed by the CO_2 present in gases from a slow-combustion furnace fed with gas-coke. The operation consisted in passing the gases containing nitrogen and carbon dioxide (with some impurities), through a tower containing coke over which the stream of cyanide solution trickled. The decomposition was as follows: $2\,KCN + 2\,H_2O + 2\,CO_2 = 2\,HCN + 2\,KHCO_3$.

The exhausted liquor, after suitable concentration, was reused for saturating the leached and dried charcoal, with suitable additions of fresh material. The saturated charcoal was then dried as before and went through the above-mentioned cycle of operations.

The hydrocyanic acid gas swept out of the "scrubber" by the nitrogen was absorbed by a cold-water spray in another tower, also filled with coke, the gas being dissolved in the water as hydrocyanic acid. This liquid was collected in stills (actually large steam boilers) and heated with steam coils. The HCN, with some water vapor, passed through a coil cooled to 25° C. to 28° C., which allowed the water to condense and return to the still, while the HCN passed forward. This gas was then condensed in another coil, kept as cold as possible, the liquefied HCN dropped into caustic soda, the compound NaCN, H_2O being formed. The crystals were "fished out" and "centriffed." The trace of caustic soda adhering to the cyanide became converted into sodium carbonate, and the crystals were sold as 80 per cent (equivalent of KCy) strength. By long-continued drying the combined water could be removed to a considerable extent, the "cyanide" content being similarly increased.

When the works were started it was expected that the total cost of manufacturing cyanide (based on 98/100 KCy) would be about 3 d. (6 cents) per pound, but the actual cost was about 1 s. (25 cents) per pound. Among the causes which contributed to this unexpected increase of cost were the following:

(1) Loss of carbonate of potassium by formation of silicate of potassium from ashes in wood and impurites in coke.

(2) Loss of carbonate of potassium by formation of sulphide, sulphate and sulphite of potassium from sulphur compounds in furnace gases and from coke.

(3) Waste of charcoal from oxidation due to porosity of retorts.

(4) Excessive fuel consumption (over that calculated as necessary) for power and heating.

(5) Excessive wear and tear through damages to plant from explosions, etc.

I have in my possession some of the sodium cyanide made by this method, but it has lost most of its water of crystallization. Modern methods in vogue in Europe and America for the manufacture of cyanide of sodium and the mixed (98/100 per cent) cyanides of potassium and sodium are now brought to such a state of perfection that it seems useless to try and make cyanide here by the fixation of atmospheric nitrogen. Recent experiments have convinced me that—in this country at least—such processes are foredoomed to failure from a financial standpoint.

Gold Refining.

The Wohlwill process of gold refining with the recovery of platinum has been the subject of quite a number of articles in this journal. (See especially the article by D. K. Tuttle, Vol. I, page 157, and the articles by the inventor himself, Dr. Emil Wohlwill, in our vol. II, pages 221 and 261, and vol. VI, page 450.) A recent patent of the Norddeutsche Affinerie A. G., with which Dr. Wohlwill is connected (German patent 207,555, Sept. 22, 1908), is exceedingly interesting since it refers to a simple modification which appears to broaden greatly the scope and applicability of the Wohlwill process.

If gold, rich in silver, is refined in a solution of gold chloride containing pure hydrochloric acid or other chlorides, it becomes necessary to remove mechanically from time to time the layer of silver chloride which forms on the anode; in practice it has been found that this is necessary whenever the silver content in the anode is more than 6 per cent. If the layer of silver chloride is not removed from the anode, chlorine is set free and this will take place at an earlier stage of the process, the higher the current density used and the higher the silver content of the anode.

It then becomes necessary to reduce the current density, but this increases considerably the expense due to the smaller output and the increased capital charges per unit weight of output.

A novel arrangement by which these difficulties are overcome is the use of a pulsating current instead of a constant direct current. This is best obtained by connecting a direct-current dynamo in series with an alternating-current dynamo. As long as the voltage of the latter is lower than that of the former, the current obtained by the superposition of the two currents will always have the same direction, but will change periodically from a minimum to a maximum. While such a current produces some good effect in electrolytic gold refining, it is preferable for practical purposes to use an alternator with a higher (maximum) voltage than the direct-current dynamo in series

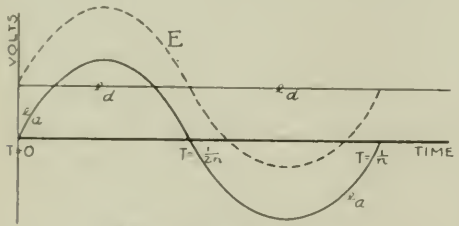


DIAGRAM OF PULSATING CURRENT.

with it. In this case we get a current which has not always the same direction, but is really an alternating current symmetrical with respect to the voltage curve of the direct current dynamo. The adjoining diagram illustrates the voltage curves. The straight line ca is the curve of the direct-current dynamo voltage, the curve c_a is the curve of the alternating current dynamo. The dotted curve E is the resulting voltage obtained by the superposition of the alternating current upon the direct current. It is symmetrical with respect to the straight line ca.

The electrolytic effect is, according to Faraday's law, equal to the effect of the direct current alone. In other words, the total gold deposited on the cathode can be found from Faraday's law, on the supposition that only the direct current is passing.

The effect of the alternator in series with the direct-current dynamo manifests itself in various ways. The voltage at the terminals of the refining cell measured with a direct-current voltmeter is less than if an absolutely constant current is used. Further, the anodic reaction is essentially influenced by the alternating current. It is possible to increase considerably the anodic current density without causing the development of chlorine gas and without scratching off the layer of silver chloride.

A limit is set to the increase of anodic current density in so far as a relatively weak development of oxygen gas sets in when by increasing the current density the voltage reaches that amount for which chlorine would be set free with a purely direct current. The evolution of oxygen is first so small that the resulting decrease of ampere-hour efficiency is comparatively unimportant. It has even a certain advantage, since it facilitates the dropping off of the silver chloride layer. With increasing current density the evolution of gas becomes stronger. The limit of the current density above which one should not go without decreasing unduly the ampere-hour efficiency, depends on the content of silver in the anodes and on the value of the alternating current strength.

The permissible limit of the direct-current density (amperes per square inch of anode) is the higher, the less the content of silver and the higher the ratio of alternating-current strength to direct-current strength. As soon as a strong gas development takes place at the anodes (more than a few small bubbles) it becomes necessary to decrease the direct current density while the alternating current is not changed, or to increase the alternating current while the direct current remains unchanged.

As an example it is mentioned that if raw gold containing about 10 per cent of silver is treated according to the original Wohlwill process, an anodic current density of not more than 750 amp per square meter must be used, and in this case it is necessary to scratch off the silver chloride from the anode every 45 minutes. When using an alternating current, the strength of which is about 1.1 times that of the direct current, it is possible to maintain an anodic direct-current density of 1,250 amp per square meter without any necessity of scraping off the silver chloride.

The method also enables one to treat by the electrolytic refining process alloys much richer in silver than is possible with direct current alone. For instance, gold containing 20 per cent of silver can be refined if the ratio of the alternating current to the direct current is 1.7 to 1 and it is possible to use in this case an anodic direct-current density of 1200 amp per square meter. With lower direct-current densities the process is applicable also for gold still richer in silver, and appears to cover now all cases which occur in practice.

The use of a pulsating current and the resulting possibility of employing high-current densities involves another important advantage. In the original Wohlwill process employing direct current alone, about 10 per cent of the gold of the anodes passes as a finely subdivided dust into the slime and must be recovered by chemical means. If, however, a pulsating current is used, gold passes into the slime only at the beginning of the process as long as the anode is not completely covered with silver chloride. Later on practically no gold whatever passes into the slime.

The slimes, therefore, consist essentially of silver chloride and its spongy structure causes it to drop off by itself. Since at the beginning of electrolysis some little gold always passes into the slime and mechanical losses from the anode cannot be entirely avoided, especially near the end of the refining process, the slime will always finally contain some gold, but generally not more than 1 per cent of the gold content of the anodes. The saving of interest due to this considerable reduction of the gold remaining in the slimes represents a very essential advantage.

On account of the small amount of gold lost from the anode into the slime it is preferable to use a pulsating current instead of a direct current, even in such cases in which gold poor in silver is to be refined.

Whenever an especially high current density of more than 1000 amp per square meter is not of importance, the process has the advantage that the amount of pure hydrochloric acid or of chlorides added to the cell may be much smaller than with

direct current alone. With gold which contains little silver, electrolysis with a current density of 1000 amp per square meter with a purely direct current requires the addition of about 3 per cent hydrochloric acid and the electrolyte must be heated to a temperature of 60° C. or 70° C. The superposition of alternating current upon the direct current permits the use of only one-fourth of the amount of hydrochloric acid (which gives a solution containing about 1 liter of hydrochloric acid, 1.19 s.g., for 60 liters solution—that is, about 7 or 8 grams of hydrochloric acid per liter). On the other hand, if a solution containing 3 per cent HCl or more is used and if the current density is not above 800 amp or 1000 amp per square meter, it is no longer necessary to raise the temperature of the electrolyte to 60° C. or 70° C.

When carrying out the process with a purely direct current the heating of the electrolyte is absolutely necessary, since otherwise the gold is precipitated in form of a dark brown or black powder. This is true even if a current density of less than 500 amp per square meter is used, which seems absolutely impractical in view of the increase of interest charges. So much about the effect on the anodic reaction.

The application of a pulsating current, however, permits also an increase of the cathodic current density in a hot solution; or if the cathodic current density is below 1000 amp per square meter it is possible to obtain perfectly dense coherent deposits even from a cold solution.

The process is carried out in practice as follows: A direct-current and an alternating-current dynamo are connected in series and the voltage of the direct-current dynamo is regulated so that a direct-current ammeter (which does not indicate alternating current) indicates the desired direct-current strength in amperes. If this direct-curent strength is id amp and if it is intended to use an alternating current with an effective* value of ia amp, the effective value of the pulsating current obtained by the superposition of the direct current upon the alternating current is $I = \sqrt{id^2 + ia^2}$. This is the total effective current I as measured by a hot-wire ammeter (the action of which depends on the Joulean heat).

A hot-wire ammeter is, therefore, used in series with the direct-current ammeter and the voltage of the alternating-current machine is now regulated until the hot-wire ammeter shows that the alternating current has reached the correct value. If, for instance, it is intended that the "effective" value of the alternating current should equal the direct current, both being equal to 200 amp, the hot-wire instrument must indicate $\sqrt{80{,}000} = 283$ amp.

The total voltage which determines the power consumption is also measured by a hot-wire instrument. Its value depends on the strength of the total current. If, for instance, raw gold is refined which contains about 10 per cent silver, it is recommended to use a direct-current density of about 1250 amp per square meter and an "effective" alternating-current density of the same value.† the total current density, as found with the hot-wire instrument, is $\sqrt{1250^2 + 1250^2} = 1767$ amp per square meter.

In this case the direct-current voltage at the terminals of the cell, measured by a direct-current instrument, would be about 1 volt and the total voltage about 1.4 volts. This gives an alternating-current voltage of 0.9 volt.

In the design of the dynamos it is best to figure on 1.3 volts as the maximum direct-current voltage, and on 1 volt as the maximum alternating-current voltage per cell. Since both dynamos are connected in series their armatures must be dimensioned so that they can stand the heating effect of the maximum total combined current. Within the usual limits of frequency (that is, for frequencies below 50 periods per second) the value of the frequency has practically no effect on the results obtained with the alternating current.

The most suitable value of the current density and voltage depends, of course, on the composition of the gold or gold alloy. If the gold contains less silver than 10 per cent, it is possible to increase the direct-current density somewhat and to decrease the ratio of alternating current to direct current slightly. If the gold contains much more silver, either the direct-current density should be diminished or the alternating-current density increased.

Otherwise, the process is the same as formerly carried out. In general, gold chloride solutions are used containing at least 3 per cent of hydrochloric acid and heated to 60° C. or 70° C. If, however, time is not of great importance, so that a direct-curreut density between 500 amp and 1000 amp per square meter can be employed, it is possible to use a gold chloride solution containing only 0.7 per cent to 0.8 per cent of hydrochloric acid and heated to 60° C. to 70° C., or it is not necessary to heat the solution at all if 3 per cent of hydrochloric acid is added.

Instead of hydrochloric acid, or part of it, chlorides which form double salts with gold chloride, for instance, sodium chloride, can be used. As previously described, thin rolled gold sheets are used as cathodes and gold is supplied to the electrolyte periodically in form of gold chloride to make up for the loss of gold from it. If gold rich in lead is refined, sulphuric acid is added to the electrolyte in quantities equivalent to the free hydrochloric acid present.

Rutgers Chemistry Building.—The contract for the new chemistry building of Rutgers College has been awarded and the work is under way. The building will be of brick with terra cotta trim, and will harmonize architecturally with the engineering building which was completed slightly over a year ago. On the first floor will be a large lecture room and three classrooms. On the second floor will be a qualitative and a quantitative laboratory, a laboratory for work in electrochemistry and one for water analyses, professors' private offices and laboratories, and a supply room. In the basement will be the laboratory for the elementary work of the freshman year, a stock room and the necessary heating and ventilating apparatus for the building.

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*The direct current i_d does not change in time, while the alternating current does change periodically. The simplest kind of wave form is a sine wave; we may, therefore, write the instantaneous value of the alternating current $= i \sin(2\pi nt)$ where i is the maximum value of the current, t is the time in seconds and $1/n$ is the period in seconds. It will be seen that the current is zero at the moments $t = 0$, $1/2n$, $1/n$, while it reaches the maximum $+ i$ at the time $t = 1/4n$, and the minimum $- i$ at the time $3/4n$. The arithmetic mean value over a whole period is zero. As this determines the electrolytic action (since according to Faraday's law the deposit is proportional to the product of time and current), the total electrolytic action caused by an alternating current during one full period is zero; in other words, all the electrolytic deposit obtained during the first half of the period (while the current has always the same direction), is being undone during the next half period (while the current has the opposite direction). It is different, however, with the Joulean heat produced by an alternating current because this is proportional to the square of the current; its value during one half period is the same as during the next half period and the two add together. It is, therefore, customary to define the "effective value" of an alternating current (such as indicated by a hot-wire ammeter) as the square root of the mean square of the current. That is, the mean is determined of the squares of the instantaneous values of the current at all moments during a whole period; this gives the "mean square" of the current; by taking the square root, we get the "effective current." It can be easily shown by mathematical calculation, that if i is the maximum value of the current as above the effective value is $i/\sqrt{2}$; this effective value is designated in the text with i_a. With a pulsating current, obtained by the superposition of a direct current i_d over an alternating current $i_a\sqrt{2} \sin(2\pi nt)$, hence of effective value i_a, we can also form the two mean values over a full period. The arithmetic mean, which determines electrolytic action, is simply i_d (as though the alternating current was absent). The "effective" mean value, which is found by a hot-wave ammeter, is $I = \sqrt{i_d^2 + i_a^2}$ as stated above in the text.—Editor.

†If the direct-current density is 1250 amp. per square meter and the effective alternating-current density is also 1250 amp. per square meter, the maximum alternating-current density is $1250\sqrt{2} = 1767$ amp. During that half period of the alternating current, during which the latter has the same direction as the direct current, a certain electrolytic work will be done by the sum of the two currents. During the other half period, some more electrolytic work of the same nature will be done as long as the instantaneous value of the alternating current is below 1250 amps. But during that portion of time, during which the alternating current rises from 1250 amps. to its maximum (1767) and falls again to 1250 amp., the electrolytic action is opposite: what was anode is now cathode and inversely; part of the electrolytic work formerly performed will now be undone. The total electrolytic work performed by the pulsating current (weight of gold refined and deposited) during one full period will be just the same as though we had only a direct current of 1250 amp. But the electric power consumption is, of course, that of both the direct current and the alternating current together.—Editor.

The Use of Carbonaceous Filters in the Smelting of Zinc.*

BY C. O. BANNISTER.

The process described in this paper is the outcome of experiments commenced by Evan H. Hopkins some ten years ago.

These experiments were begun with the object of obtaining zinc free from lead as a product when smelting zinc materials containing lead.

In the first experiments a mixture of zinc oxide, lead oxide and carbon was placed in a cavity in a carbon block fitted with a carbon lid, the carbon block was then introduced into a muffle, and the zinc was distilled over into a receiver. As the result of the experiment, zinc practically free from lead was obtained, and the lead was found in a lump inside the carbon block.

The first idea as to the method of applying this principle on the large scale was to use a modified retort such as that

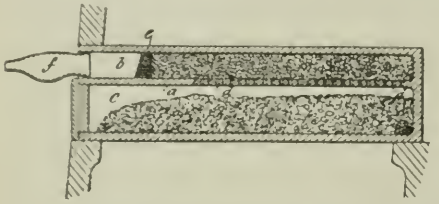


FIG. 1.—FIRST FORM OF RETORT, WITH CARBONACEOUS FILTER.

shown in Fig. 1, and to pass the products of distillation through an incandescent filter of carbon. In the figure given the retort is shown to be divided into two portions by means of a partition, *a*, made of clay, and perforated with small holes, *d*.

The upper chamber, *b*, was filled with fine lumps of charcoal or coke, well pressed together, and kept from falling apart by means of a movable and perforated stopper, *c*. After the ore mixture had been placed in the chamber, *c*, this portion was closed and heated. During the process the zinc distilled through the carbonaceous filter, and was condensed and collected in the receiver, *f*, in the ordinary way.

From the beginning it was evident that in order to make this process a commercial success it would require considerable modification and simplification, and experiments were carried out with a view of making the condenser also take the part of a

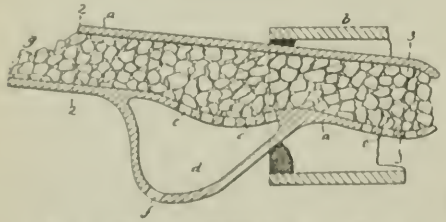


FIG. 2.—MODIFIED CONSTRUCTION OF CONDENSER.

filter. After a large number of experiments the form shown in Fig. 2 was adopted. This consisted of a large condensing chamber divided by means of a perforated partition into a condenser proper and a receiver, *d*.

The upper portion or condenser was filled with small pieces of carbon in the form of charcoal or coke, and the whole was placed sufficiently far into the zinc retorts to insure that the carbon filter should attain a sufficient temperature to prevent choking. This form of condenser was modified slightly from time to time and was used in actual practice, the results obtained being compared with those obtained from retorts fitted with the ordinary condenser. The experiments were made with much care, particular attention being paid to weight, charge, analyses, and like details, and the whole operation was watched for 24 hours without intermission.

During these experiments, which were made in furnaces treating galvanizer's dross, the following points were clearly brought out:

1. The zinc obtained from the Hopkins condensers was invariably purer and freer from lead.

2. No oxide or zinc fumes emanated from the Hopkins condensers.

3. The zinc flame, always observable at the mouth of ordinary condensers, was absent when the Hopkins condensers were used.

4. By the use of the Hopkins condensers a larger proportion of the spelter contained in the charge was obtained as metal.

5. It was established that the hotter the furnace the greater

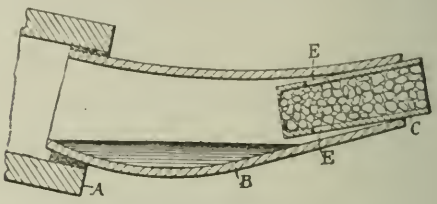


FIG. 3.—CONDENSER CONSTRUCTION.

was the yield of metal by the Hopkins condensers, whereas with the ordinary condensers higher temperature means increased zinc flame, fumes and loss.

It will thus be seen that the above series of trials was most fruitful in indicating the practicability of preventing the zinc fumes which are always a cause of annoyance in the neighborhood of zinc works, and at the same time the prospect of obtaining thereby a greater yield of zinc.

It was, however, considered still possible to simplify the construction of the condenser so as to make it much cheaper. To this end the whole of the central portion of the condenser (Fig. 2) was eliminated, but exactly the same principle was retained in the new arrangement. In this case the condenser, *B*, as ordinarily used in spelter works, or a slightly modified form thereof, may be used, into which is inserted, at the inner end, a carbonaceous filter to keep back the lead, and, at the outer end a tube, *C*, packed with carbon. This combined form is to be used when the aim is to obtain a high quality of zinc as well as the subsequently mentioned advantages, but when it is merely desired to eliminate the objectionable fumes

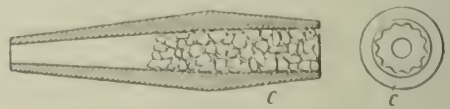


FIG. 4.—CARBONACEOUS FILTER

and to obtain a higher extraction, the outer tube only need be used as indicated in Fig. 3.

Various forms of fire-clay tubes have been employed to contain the carbonaceous filter; to illustrate one of these Fig. 4 is given, which shows a double conical tube, having one end, namely, that which enters the condenser, made with longitudinal corrugations internally. By reason of these corrugations the choking of the tube is prevented during work, but the escaping gases, etc., come into contact with the hot carbon. The tube at present in use consists of a cylinder about 8 inches long, of 3½ inches external diameter, and 2 inches internal diameter, and has flanges to keep in the coke; it also

*A paper presented before the Institute of Metals at the annual general meeting held in London on January 10, 1912.

has a small hole running through its wall for the insertion of a pricker when found necessary during the process.

It will be seen from the foregoing that the process, beginning with the object of making lead-free spelter only, has been gradually modified and simplified, and in its present form embraces the three following objects:

(a) Manufacture of pure spelter.
(b) Prevention or reduction of zinc fume.
(c) Obtaining of higher yields.

(a) Manufacture of Pure Spelter.

Since Mr. Hopkins first demonstrated the possibility of obtaining pure zinc by passing the vapors from zinc retorts through incandescent carbon, this process has received considerable attention from zinc smelters, and it is now an accepted fact that the operation may be carried out economically and effectively.

To what extent the process will be applied to ordinary zinc smelting is at present unknown, as the demand for zinc of a very high standard of purity is somewhat limited; but there is no doubt that it will be adopted in the treatment of materials containing too much lead for the production of commercial spelter direct, that is, without a subsequent refining operation.

The fact that a carbonaceous filter behaves as a strong reducing agent puts it far in front of filters composed of materials which do not possess this property, for its presence in an incandescent state insures that the whole of the metallic vapors shall be and remain reduced, and at the same time any carbon dioxide passing out with the gases will be reduced to carbon monoxide.

As the amount of reduction that takes place in the filter requires only a little carbon, the consumption of the carbon of the filter is small; but this consumption is advantageous, as it tends to keep open the filter and lessen pressure due to the deposition of lead, etc.

In the actual working, coke has proved more suitable than coal for the filter, and at the same time it is more economical. In fact, experiments have demonstrated that coke is the best material for this purpose, and it should be of a light, porous variety, not dense, and should contain only a small quantity of volatile matter.

As to the results obtained, the following illustrates the purity of the zinc:

From galvanizer's dross the spelter contained 99.86 per cent zinc from the Hopkins filter as compared with 98.75 per cent from an ordinary condenser working on the same charge.

As to whether the process will be used to any extent for the treatment of zinc-lead concentrates, time alone will decide, for details other than the production of pure zinc have here to be taken into account, such as the manner in which the charge attacks the retorts and other parts of the apparatus. For rich zinc concentrates containing smaller quantities of lead, however, there is no doubt that the process is most suitable, as not only can pure zinc be obtained but a considerable quantity of the lead can be recovered on treatment of the filters.

Some comparative tests have been carried out on the treatment of Broken Hill middlings, and not only was a purer zinc obtained from the Hopkins filters, containing 99.6 per cent zinc as compared with 96.5 per cent from ordinary condensers, but the yield of metal averaged over 20 per cent more when such filters were used.

(b) Prevention of Zinc Fume.

In view of the necessity for some cheap, simple and effective method for preventing the formation or reducing the amount of fume during zinc smelting, the Hopkins outer cork is of very great importance, for it is found to be most efficient as a fume preventer.

The presence of this cork or nozzle, packed with carbon, acts in almost a magical manner, keeps back practically all trace of zinc fume, and gives to the gas burning at the ends of the nozzle a flame similar to that obtained during the burning of carbon monoxide gas. The inspection of a zinc furnace fitted partly with ordinary condensers and partly with Hopkins condensers is most surprising and instructive, and the absolute efficiency of the nozzles as now used is at once clearly demonstrated. The presence of the hot carbon at the outside end of the condenser or pipe seems to affect the whole condition of the condenser, for even when put in so loosely that part of the flame escapes between the carbon-containing nozzle and the mouth of the condenser, this flame is also practically free from zinc. From the time the gas in the coal finishes until the last tap, the whole condition of the furnaces supplied with nozzles is improved.

During the periods of tapping the zinc the fumes are very much less, for only two or three condensers are uncovered at a time, and as tapping proceeds, the nozzles as removed are put into the condensers already tapped. The workmen also notice further advantages, for the metal in the condensers fitted with nozzles is hotter than the metal in ordinary condensers, and will more readily melt up "drippings" of spelter; also skulls of metals are never present in the condensers.

The Local Government Board have taken great interest in all methods suggested for the prevention of the escape of zinc fume, and four Government inspectors have examined the Hopkins process in operation in Swansea, and were much impressed by what they saw. The following are extracts from the Annual Report on Alkali, ..., Works, 1908:

Page 90: "In a previous report (42d, page 64) I have mentioned that trials were being given to the Hopkins nozzle, one of the variants of this type of appliance, and with some amount of success. The results of trial have led to simplification in design and construction without, I am informed, any sacrifice of efficiency. It is now believed that not only can spelter of greater purity be obtained, but also in substantially increased yield as compared with other retorts working under the same conditions in the ordinary way without the nozzle. If working for quality, *i. e.*, freedom from lead, the vapors from the retort pass through an additional incandescent chemical filter suitably formed in the furnace end of the condenser. At the outer end of this condenser another appliance aims at reducing any zinc oxide formed in the condenser from access of air to the vapor of zinc. From recent observation of two contiguous benches of thirteen retorts each and comparison of escaping gases near the time of first tapping, *i. e.*, at the period of the greatest liability to evolution of fume, I am convinced of the possibility of success in preventing this in very great measure. It was easy to compare the fume escaping from similar benches not provided with the nozzle, and in the same condition, as to charge and equipment."

Page 180: "Several experimental efforts are at present being made to reduce the impalpable white cloud of oxide of zinc which escapes from the retorts during the smelting operation.

"The 'Hopkins Condenser' is being tried at Swansea with encouraging results. The condenser, which is provided with what may be called a chemical cork, consisting of coke, is adapted to the retort or pot commonly in use.

"By this means the inventor endeavors to prevent the formation of zinc oxide during distillation, thereby obviating the escape of the metal as an oxide seen in the familiar white fumes, inseparably associated with the manufacture of spelter or commercial zinc. At the works referred to, these condensers showed a remarkable diminution in the escape of fumes, and we are informed that there is a substantially increased yield of metal as compared with that obtained by the ordinary process of smelting."

(c) Higher Yields.

The absence of fumes which is noticed when the Hopkins nozzles are in use on zinc retorts should naturally lead to less loss by oxidation and therefore to higher yields of metal.

This has proved to be the case when the process is worked under proper conditions, and the results of continued comparative tests have always shown the Hopkins nozzles to yield more metal. It must be remembered, however, that in the extraction of zinc many factors must be considered which are likely to upset results, such as the breaking of retorts, unequal heating of the furnace, etc. The first comparative tests for yield were carried out on contiguous bays of thirteen retorts, in a two-tiered, gas-fired zinc furnace; further bays of thirteen retorts were added until the whole of one side of the furnace was worked against the whole of the other side.

As the charges in all cases are absolutely similar, only the weight of metal actually obtained need be taken into account.

Table I shows the results of eighteen weeks' continuous work, and it will be noticed that during this run only one week showed a decrease in the amount of spelter yielded by the Hopkins nozzles.

TABLE I.—RESULTS OF FIRST EXTENDED SERIES OF TESTS.

Date. Week ending	Ordinary: Average number of pipes used each day.	Ordinary: Average yield per pipe per day. Lb.	Hopkins: Average number of pipes used each day.	Hopkins: Average yield per pipe per day Lb.	Increase by Hopkins Per cent
Oct. 12	143	20.88	13	21.90	4.9
Oct. 19	143	20.43	13	22.90	8.9
Oct. 26	143	20.00	13	21.80	9.0
Nov. 2	130	20.20	11	22.60	12.0
Nov. 9	141	21.26	13	22.94	7.9
Nov. 16	141	22.10	13	23.30	5.4
Nov. 23	135	20.00	18	22.60	13.0
Nov. 30	129	20.20	26	19.00	—6.3
Dec. 7	132	19.20	23	19.70	2.6
Dec. 14	130	19.20	25	20.40	6.2
Dec. 21	129	19.13	25	20.60	7.7
Dec. 28	125	20.90	24	21.16	1.2
Jan. 4	125	20.80	24	22.40	7.7
Jan. 11	129	18.20	26	19.20	5.5
Jan. 18	129	16.10	26	18.20	13.0
Jan. 25	128	15.91	25	17.70	11.2
Feb. 1	127	16.87	25	18.68	10.7
Feb. 8	116	17.54	38	18.85	7.5

Table II shows the results of the first week's work on a recently repaired furnace, one side being worked against the other.

It must be remembered that in the ordinary method of working the zinc furnace the yields vary very considerably from day to day in different furnaces and even in the two sides of one furnace, and therefore it is not to be expected that occasionally for one day, or even in the case of the average for a week, the yield should apparently be less when the condensers are used. On examination of the tables given it will be noticed that there is not only a distinct advantage from the point of view of yield, but also that the percentage variation and the greatest deviation from the mean result are less with the Hopkins nozzles, points which are both in favor of this method.

TABLE II.—DETAILS OF ONE WEEK'S WORK.

Date	Ordinary: No. of pipes	Ordinary: Yield Cwts.	Ordinary: Yield Qrs.	Ordinary: Yield Lb.	Ordinary: Average yield per pipe lb.	Hopkins: No. of Pipes.	Hopkins: Yield Cwts.	Hopkins: Yield Qrs.	Hopkins: Yield Lb.	Hopkins: Average yield per pipe.	Increase by Hopkins Per cent
Oct. 7	75	12	1	5	17.51	13	2	0	21	18.[illegible]	3.0
Oct. 8	75	11	0	14	15.49	75	11	1	21	17.45	12.6
Oct. 9	78	11	1	16	16.36	75	12	3	6	19.12	16.9
Oct. 10	78	10	3	29	15.27	78	12	0	7	17.45	12.6
Oct. 11	78	13	2	25	18.08	78	13	2	1	19.15	7.4
Oct. 12	78	11	2	1	16.62	74	13	1	6	19.62	18.[illegible]
1923	75	12	1	10	18.43	—4.1	Oct. 13			28	11	1	10		

In examining the tables given it will be noticed that there is not only a distinct advantage from the point of view of yield, but also that the percentage variation and the greatest deviation from the mean result are less with the Hopkins nozzles, points which are both in favor of this method.

It is estimated that the corks or nozzles will be made at a cost of 4s. to 5s. ($1.00 to $1.25) per 100, and that filling for a furnace of 150 retorts will cost 2s. to 2s. 6d. (50 to 62 cents) per day.

Summary.

1. By means of the Hopkins carbonaceous filters and corks or nozzles the following three advantages may be obtained in zinc smelting: (a) Pure spelter. (b) no zinc fume. (c) higher yield.

2. The process may be applied to any already existing retort zinc furnace, no constructive alterations being necessary.

3. Only a slight modification of the condenser is necessary, this consisting in the widening of its mouth.

4. The process is absolutely simple to work.

5. The metal is hotter in the condenser, and no skulls of solid zinc are formed.

6. Not only is a higher yield obtained, but also a more steady yield, less fluctuation taking place when the nozzles are in use.

Kirchhoff Luncheon.

Some 130 friends of Mr. Charles Kirchhoff assembled at a luncheon at the Engineers' Club, New York City, on Sunday, Jan. 16, to express their admiration for him as editor and man, on the occasion of his recent retirement from the editorship of *Iron Age* which had extended over 30 years.

CHARLES KIRCHHOFF.

Mr. Philip T. Dodge, president of the Engineers' Club, acted as toastmaster.

Mr. T. C. Martin read letters from many who were unable to be present, among those who had sent expressions of their high esteem for Mr. Kirchhoff were Andrew Carnegie, James Douglas, John Hays Hammond, James H. McGraw, J. Langeloth, John Birkinbine, and W. H. Wiley.

Dr. R. W. Raymond, secretary of the American Institute of Mining Engineers, made one of his delightful characteristic speeches. He referred to the contest for the secretaryship of the Institute in 1885, when both Raymond and Kirchhoff were candidates. This contest became the foundation of a long, firm, and undisturbed friendship.

Mr. George W. Cope, editor of *Iron Age*, spoke eloquently as the representative of the old associates of Mr. Kirchhoff in David Williams Company and presented to him a French bronze by Picault, "La Source du Pactole."

Mr. Kirchhoff's reply, which showed clearly how much he was moved, was as modest and sincere as the man himself.

The final speakers were President Henry S. Drinker, of Lehigh University, and Mr. John Fritz, the veteran iron master, who had come, in spite of his 87 years, from Bethlehem, Pa., to express his regards for his friend of many years.

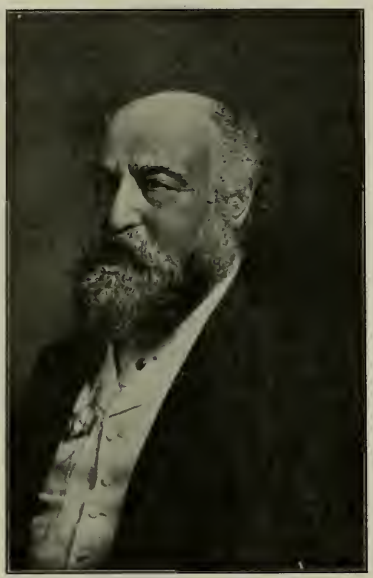
"Auld Lang Syne," played by Mr. Geo. H. Guy, marked the conclusion of the luncheon, which was a modest yet most expressive tribute to one of the greatest, ablest, and most influential editors which the engineering press of any country has ever had.

Mr. Kirchhoff has sailed on a trip to the West Indies.

The Late Dr. Ludwig Mond.

Dr. Ludwig Mond, who died in London before the sun rose on Saturday, December 11, was one of the most successful industrial chemists of his time—a man who owed his success to a rare combination of business ability, capacity for research and energy. The public knew him as one of the founders of the alkali firm of Messrs. Brunner, Mond & Company, as the inventor of the Mond gas and of a nickel process, and as a generous donor to many scientific institutions. We take this account of his life from the London *Engineering*.

Ludwig Mond was born nearly 71 years ago at Cassel, then the capital of the Electorate of Hessen, which since 1866 has formed part of the Prussian kingdom. His father, a merchant, had him trained at the Polytechnic Institute of Cassel, and later at the universities of Marburg and Heidelberg. Recommendations from so distinguished professors as Hermann Kolbe and Robert Bunsen stood young Mond in good stead when, after having held appointments in two German chemical works, he came to England in 1862. He found employment at Widnes in the chemical works of Messrs. Hutchinson and Earle, and there he was able to show that a method patented by him for recovering the sulphur, which had so far been wasted, in the black ash of the Leblanc process was technically applicable; this method was adopted in Newcastle and Glasgow.



DR. LUDWIG MOND.

Returning to the Continent, he joined Ernest Solvay, of Charleroi, near Brussels, the inventor of the ammonia-soda process. Having founded Solvay works at Utrecht, he returned to England in 1867, acquired the Solvay patents for England and started, in 1872, works of his own at Winnington, near Northwich, over the Cheshire salt deposits, with Mr. (now Sir) John Brunner, whose acquaintance he had made in the Hutchinson family, as partner.

It was a risky undertaking to compete in England against the Leblanc process, where it had been elaborated into one of the most perfect cyclic processes known. The raw materials of the Leblanc process are salt, sulphuric acid, coal and lime—all relatively cheap. In the ammonia-soda process the brine is saturated with ammonia—an expensive material—and then with carbonic acid, the crystallizing bicarbonate of soda being afterwards converted, by boiling or by calcination, into ordinary carbonate of soda. The Leblanc process yields hydrochloric acid as a by-product; in the Solvay process the chlorine combines with calcium to form chloride of calcium, and the enormous quantities of this compound remain practically waste heaps up to the present time. A good deal of the success of the new firm was due to the improvements which they effected in the recovery of the ammonia (by means of quicklime). When they started they knew that Dyar and Hemming had tried an ammonia-soda process, likewise in Cheshire in 1838, and had failed, and that the struggle against the giant Leblanc was considered hopeless. After 1875, however, their success was assured. In recent years new and formidable rivals have arisen in the electrolytic alkali processes. To meet that rivalry new efforts were made to utilize the calcium chloride; some of it is now treated with zinc oxide and transformed into zinc chloride, which on electrolysis yields a very pure zinc, and the calcium chloride in itself makes a fair road dressing.

The sight of the enormous waste of gases in his Winnington boilers, it is said, suggested to Dr. Mond the study of the producer-gas problem. He succeeded in burning cheap bituminous slack by the aid of an air-blast and steam and in absorbing the ammonia evolved by dilute sulphuric acid, thus obtaining a fair producer gas and, in addition, ammonium sulphate as a by-product, whose value, unfortunately, decreased, owing to this and other new sources of production. Mond gas is now chiefly made at Dudley Port, in Staffordshire.

The same researches led him to another discovery. The producer gas contains hydrogen, and when, 20 and more years ago, the problem of galvanic gas cells for the direct generation of electricity from coal—not by way of boilers, steam engines and dynamos—excited general attention, Dr. Mond attempted to isolate the hydrogen from the other constituents of the producer gas. For this purpose he conducted the gas over various compounds and metals, and observed that nickel was taken up by the gas; it was the carbon monoxide which formed nickel carbonyl under certain conditions of temperature and pressure, and this carbonyl is again decomposed at 200° C. and deposits a mirror of pure metal. The new compound was first described by Dr. Mond in 1890, and the Mond Nickel Works at Clydach, near Swansea, are now converting part of the nickel ores from the famous Canadian mine at Sudbury, or the matter smelted there, into a very pure nickel.

Years were, of course, required to work out this process, and Dr. Mond had the assistance of many able chemists and engineers in his researches and undertakings. It would be wrong, however, to regard Dr. Mond mainly as the managing director and commercial chief. Dr. Mond was himself a remarkably capable and painstaking research chemist. During the last five years his health caused his friends much anxiety. About a year ago he recovered wonderfully, and he ought to have spared himself. Always busy and hard at work with the general management, however, he took the study of the carbonyls up again, and succeeded in preparing carbonyls of iron, cobalt, molybdenum and ruthenium, which are stable only at very high pressures—450 atmospheres in the last-mentioned case. He announced these discoveries at the International Chemical Congress, which met in London in May and June last. When he fell ill again this autumn grave fears were entertained.

Since 1857 Dr. Mond has been domiciled in England. The year previous he married his cousin, Miss Frieda Loewenthal. His winters he liked to spend in Italy, and his house, the Poplars, in Avenue Road, near Regent's Park, in which he died, is well stocked with masterpieces of the early Italian school. His friendship with the great Italian chemist, Stanislao Cannizzaro, induced him to found a Cannizzaro prize.

Having long supported low-temperature research in the Royal Institution of London, at whose lectures he was a familiar figure, he established the Davy-Faraday Research Laboratory in the house adjoining the Royal Institution in Albemarle Street, and endowed it liberally. The laboratory was opened by the King in 1896. His donations to the Universities of London, Cambridge, Manchester and Liverpool, the Lister Institute of Preventive Medicine, and the association which aims at creating a Chemische Reichsanstalt, a sister institute to the Physikalisch-Technische Reichsanstalt in Char-

lottenburg, were on a grand scale; and it was due to his munificence that the Royal Society, which elected him a Fellow in 1891, was enabled to start its international catalogue of scientific papers. Dr. Mond held honorary degrees from the Universities of Padua, Heidelberg, Manchester and Oxford. One of the honors he valued most highly was his election, in 1898, to the Athenæum Club under the rule which empowers the Council to admit distinguished men of science.

Dr. Mond leaves two sons—Dr. Robert Mond, a Cambridge scientist, and Arthur M. Mond, M. P. for Chester.

* * *

By Dr. Mond's will he leaves $300,000, payable free of legacy duty, on the death of his wife, to the Royal Society of London. The income from this bequest is to be devoted to the assistance of research in the natural sciences along various lines. A like sum for like purposes is also bequeathed to the University of Heidelberg.

In addition to other bequests, Dr. Mond left a fund in the care of six trustees to be chosen by the directors of Brunner, Mond and Company, three trustees to be taken from among the workmen of that firm. The income of the fund is to be used to assist or pension aged or disabled employees of the company. In the event of the dissolution of the company the money will go to the poor of Northwich, where the chief works are located. Another $100,000 is also bequeathed to the city of Cassel, Germany, for charitable purposes. The will then disposes of the collection which Dr. Mond had got together with the advice of Dr. Richter. The collection, which is not large in point of number, was formed slowly, Dr. Mond hardly ever buying more than two or three pictures a year since he made his first acquisition, in 1892.

A schedule annexed to the will gives the names of fifty-six pictures, the list being headed "Pictures for the National Gallery." On the death of Mrs. Mond the trustees of the National Gallery may select at least three-fourths of the works named in this list, or may make up the number, forty-two, from his remaining works of art. These pictures are to be kept together in a room or rooms in the National Gallery under the name of the Mond Collection. He leaves his wife power to bequeath not more than twelve of his pictures to any person she may think fit, but they may not be taken from those selected by the National Gallery.

All the remainder of the pictures are to be given, either singly or in lots, to museums or institutions in Europe or Canada, each one to be marked as the gift of Dr. Mond, but without any restriction as to their being kept together unless at least forty go to one institution. Dr. Mond hoped that the proper authorities will consider the pictures given to the National Gallery as of national interest, and remit all duties that might be levied on the bequest.

The Late Ottokar Hofmann.

We regret with sorrow the death of Mr. Ottokar Hofmann, the distinguished metallurgist, which occurred on Dec. 24, 1909, at Kansas City, Mo. Mr. Hofmann's last work was done in Mexico from 1907 to 1908, and during the course of it he encountered hardships which seriously injured his health. Over six months ago he suffered from a complete breakdown, and from then up to the time of his death he was confined to the house.

Mr. Hofmann was born in 1843 at Rusberg, in Austria-Hungary, and educated at the Realschule and Polytechnic Institute in Vienna, and the Bergakademie at Freiberg, graduating from Freiberg in 1867.

In 1867 Mr. Hofmann came to the United States, entering the assay office of Curtis Kunzel, in San Francisco, as a partner. This was his first venture.

Mr. Hofmann has largely been identified with leaching processes for the extraction of gold and silver from ores. In 1868 he erected at La Dura, Sonora, Mexico, the first lixiviation works operated in North America, following up this success by the construction of other works at La Trinidad and San Marcial, Sonora, and Trinufo, Lower California. He has served as metallurgist for various amalgamation and chlorination mills in Nevada and California, and since 1880 was engaged in the practice of hydrometallurgy.

In 1895 Mr. Hofmann took charge of the leaching department of the Consolidated Smelting & Refining Company, at Argentine, Kan., and in 1897 he displaced the Hunt and Douglas process then in use by a process which freed copper sulphate solutions in one operation from iron, arsenic, antimony and other impurities. Mr. Hofmann further became consulting chemist for the American Smelting and Refining Company, and consulting metallurgist for the Avino mines of Mexico and chemical director of the United Zinc and Chemical Company, of Kansas. Mr. Hofmann has written a valuable treatise on the hydrometallurgy of silver.

Nitrates in Norway.

The Norwegian industry of manufacturing air nitrates for fertilizer is undergoing rapid extension involving the expenditure of nearly $15,000,000, according to the British consul at Christiania.

Two processes for the manufacture of this product are in use in Norway. The older of the two is the Birkeland-Eyde process, previously described (Vol. VII, page 304), and the newer is that of the Badische Company, also dealt with in these columns. (Vol. VII, page 245.)

Regarding the details of the Norwegian activity we take the following summary from the report of the British consul:

Up till now about $6,000,000 has been expended on the works at Notodden and Svaelgfos and the power stations under construction at Rjukan and Vamma. When all the works are completed, at the end of 1910, $14,600,000 will have been spent. A great point in connection with the development of this industry is that the opportunity has now arisen of opening up several industries in connection with the manufacture of nitrates, such as nitric acid, nitrate of ammonia, nitrate of potash, also sodium nitrate, which last is already being manufactured.

The Nobel syndicate, in conjunction with the Birkeland and Eyde Company, is now concentrating the weak acids, with the assistance of the gas furnaces, to an acid of such percentage as to become an article of transport, and further opportunities have thus been opened for export trade, especially from works with water-power that are situated near the seaboard.

It is of interest to note that no coal is used in the production of saltpeter or other products here referred to. It is stated authoritatively that there is no probability for many years to come that the sale of saltpeter produced by the method practiced at the Notodden and Rjukanfos works will be disturbed by the competition with Chile saltpeter on the question of price.

When the Rjukan Falls works are fully completed, they and the Notodden works combined will represent 240,000 hp. with a production of saltpeter representing an export value of $6,164,000.

At some not very far distant time it is not improbable that the waterways and loughs between Skien and Notodden may be increased to such size as to allow seagoing vessels to load up at Notodden. Plans have already been worked out and are under consideration in connection therewith. At present everything has to be lightered to and from Skien.

The value of the output of nitrates in Norway in 1908 was about $536,000, and the total expenses amounted to $402,000.

The following are the companies that are producing, or will shortly produce, saltpeter from the air:

(1) The Norsk Hydro-Elektrisk Kvaelstof Aktieselskab, Christiania, who are the owners of the Notodden Saltpeter-

fabrikker, the power of which (35,000 hp) is supplied from the Svaelgfos. The capital of the company, which is French, is $7,890,000.

(2) The Vammafos (Vamma Falls) Company, whose saltpeter works are now under construction. This company is a separate company, but half of its shares are said to be owned by the Norsk Hydro-Elektrisk Kvaelstof Aktieselskab; some of the preference shares are in the hands of Norwegians. The amount of the capital cannot be stated.

(3) The Rjukanfos (Rjukan Falls) Company, whose saltpeter works are also in course of erection. The capital of the company, which is one-half French and the other half Scandinavian and German, is $3,376,800. This capital has been furnished by some of the shareholders of the Norsk Hydro-Elektrisk Kvaelstof Aktieselskab and of the Kraft Aktieselskab, but the company is an independent one.

(4) The Matre and Tyin waterfalls are owned by the Norsk Hydro-Elektrisk Kvaelstof Aktieselskab and by the Kraft Aktieselskab, but the companies formed in connection with these falls are two separate companies with separate administrations. Electrotechnical works will probably be erected at both falls.

(5) Christianssands Elektrokemiske Aktieselskab, which is the property of the Badische Anilin und Soda Fabrik, Ludwigshafen, and has a capital of $53,600.

The Chemist's Place in Power Economy.

Industrial progress depends no longer on *"increased* production" only, but on "increased *efficiency* of production." Nowhere more than in the production and transmission of power are there greater opportunities for improvement for the saving of time, and labor, and money. That the chemist ought to take the foremost part in solving the problems of power efficiency is the argument convincingly set forth in a paper read at the recent convention of the American Chemical Society by Mr. ARTHUR D. LITTLE, of Boston.

Mr. Little made a vigorous presentation of the chemist's special qualification for dealing with problems of power, which should no longer be monopolized by the mechanical engineer. It is sometimes forgotten that chemical science deals with energy no less than with matter. In fact, chemistry as a science, began with the recognition of the principles, materials, and products of combustion. But chemists have allowed the mechanical engineer to usurp many things which properly come much more directly within their own province.

The combustion of coal is a typical chemical process. The selection of the most efficient coal and the determination of the conditions necessary for its most efficient combustion are essentially chemical problems. Chemical problems also are those arising in the manufacture of producer and illuminating gas, their utilization in gas engines, the development of power from the waste gases of the blast furnace, the adaptation of conditions to the proper handling and burning of peat, lignite, and waste coal, the thermometric exploration of coal piles to forestall spontaneous combustion, smoke abatement, the control and improvement of fireroom conditions by draft regulation, flue gas analysis, temperature measurements and even the placing of firemen on the bonus basis. Taking power-plant practice and the conditions of coal purchases as they stand, the properly equipped chemist should be able to increase the efficiency of power production from 5 per cent to 30 per cent.

The analysis of boiler compounds as an end in itself presents little to excite enthusiasm, but when such analyses become the means of saving $3,600 a year in the power plants of a single company they take on a new and larger aspect, not only in the mind of the chemist, but in the mind of the chemist's client.

The chemist who attacks the problems of power production will not hesitate to go outside the laboratory and take his property wherever he finds it. He will conduct boiler and engine tests, study the efficiency of grates and stokers, familiarize himself with the marvelous promise of the low-pressure turbine as an agent in efficient power production. While straining every resource of his science to produce steam economically by the combustion of coal, is it common sense for the chemist to stop there in ignorance of the fact that the efficiency of that steam can be increased at once from 25 per cent to 100 per cent by coupling a turbine to the exhaust?

The distribution of power-supplies problems no less directly within the province of the industrial chemist. He may begin with the analysis of lubricating oils. He proves his own inefficiency if he stops there. He must inform himself regarding the market prices of oils used elsewhere for similar service, the adaptability of the oils in question to application to the bearing by soaked waste, sight feeds, or gravity cups. He must be prepared to interpret his analysis in terms of practice and to follow the oil through the plant in order to prescribe conditions which shall keep down waste. There are few plants in which the industrial chemist working along legitimate lines cannot save from 20 per cent to 60 per cent of the entire lubrication account, while the oil analyst has to his credit merely a few figures which his client probably fails to understand.

The efficiency and life of bearing metals varies over an extraordinarily wide range. Some are merely the refuse from type foundries, others are so carefully adapted in their composition to the requirements of particular service as to show an efficiency 15 times or more as great as that of inferior material. Here, again, the mere analysis means little, the practical question is: "Which is the more efficient metal under the conditions imposed by practice?"

Much additional might be said regarding the opportunity before the chemist when any material concerned in power transmission is the subject of his study, whether it be leather, rubber or canvas belting, belt dressings, insulating material, trolley wire, trolley cars or trolley wheels. In every case it is within his power to create a new standard of efficiency.

Notes on Metallurgy and Chemistry in Great Britain.

(From Our Special Correspondent.)

Institution of Mechanical Engineers.

At the meeting on December 17 papers were read on "Mild Steel Tubes in Compression and Under Combined Stress," by Mr. W. Mason, and on "Compound Stress Experiments," by Prof. C. A. M. Smith.

The discussion was opened by Mr. J. J. Guest, who said that he thought the authors had not explained sufficiently fully the aim they had in view. He said so, because, after reading a similar paper himself, he found that many in his audience had been unable to acquire a precise idea of the ground he wished to cover. When a specimen was put in an ordinary testing machine it would break at a certain number of pounds per square inch; and a tension member in a bridge could be strictly compared with the result of such a test. In the case of mild steel in an unstayed boiler there was a hoop-like stress around the boiler, as well as the longitudinal stress, and the material was consequently subjected to tension in the two principal directions at right angles to each other. Experiments in ordinary testing machines did not give any evidence as to the second tension affecting the capacity of the material in respect of the first tension. The experimenters were trying to solve the following questions: (1) Could the effect of tension on the resistance of the material be ascertained? and (2) Would the second stress diminish the material's capacity of resistance? In the case of the crankshaft referred to by Professor Smith the second stress was of an opposite nature to the first; that is, one was a stress of tension and the other a compressive stress, or vice versa.

Experiments which Mr. Guest himself had carried out had satisfied him that the shear stress was the chief factor in the case of ductile mild steel, but the authors had extended their

researches to new ground. He would first criticise Mr. Mason's experiments. He did not think that two of the figures represented a perfectly fair view of matters. Mr. Mason had tested his material in axial tension, and found it gave a certain point, *A*. He had also made experiments corresponding to the boiler plate and had applied equal tension to test pieces both longitudinally and laterally, thus getting another point, *B*, as the yield point for combined axial and hoop tensions. A new tension in a direction at right angles to the first would not lower the capacity of the material to resist the first tension. In dealing with stresses combining tension and compression, Mr. Mason had first subjected the material to compression, and ascertained the yield point. Then he had applied a certain tension in combination with an equal stress of compression, and found that he could not apply much more than half the previous stress. The stress in the material in such cases was purely shear stress, and as a consequence the second stress had a great effect on the resistance of the material to the first stress. Mr. Guest's further remarks on Mr. Mason's paper are not intelligible without the aid of a diagram.

Professor Smith, as well as others, had criticized the use of tubes for these experiments, but while Mr. Guest fully recognized the necessity for using solid test pieces in some cases, he considered that in others tubes would give useful and reliable results. The whole problem of compound stresses in mild steel appeared to have closely approached solution. It was a fortunate circumstance that the problem had been attacked by the authors in totally different directions. He considered that the axis of stress was never coincident with the axis of the specimen, although in his own tests he had endeavored to make them coincide. The ordinary testing machine was merely a weighing machine, not an instrument of precision, and too much should not be expected from it. The chief point raised by Professor Smith was that where tubes were loaded in compression they might fail through secondary flexure. Mr. Mason had also had that kind of trouble, but, of course, when it occurred the tests must be rejected. Professor Smith had stated that engineers seemed to lack confidence in tests in tubes because they knew the probability of flaws existing in cheap tubes, and this naturally made them suspicious even of tests in good tubes.

Mr J. P. E. C. Stromeyer regretted that Mr. Mason had not given the hoop-tension, which would have completed the curve. With regard to Professor Smith's paper, the author seemed to infer that when a ratio of maximum stress to mean stress did not exceed a certain amount the material would not fail. From this it might appear that the stress could be increased to infinity without failure, if only the ratio were not exceeded. [Professor Smith rose and pointed out that stresses on the third plane had to be taken into consideration, and that the speaker's conclusion was erroneous.] With regard to the use of the sphingometer, he had arrived at the conclusion, as long as 20 years ago, that several instruments should be used to ascertain the elastic limit. He had always used two instruments, and for testing in connection with a Blackfriars' Bridge he used four One objection to Professor Smith's instrument was that the coiled metal strip was not of the same material as the metal which was being tested, and the difference in the coefficients of expansion would affect the accuracy of the instrument

The Physical Society.

The fifth annual exhibition of physical apparatus was held at the Imperial College of Science, South Kensington, on December 14. The exhibits fully maintained the standard of excellence of which the experience of previous years warranted the expectation

The Bausch and Lomb Optical Company had some very efficient microscopes of new model, microtomes, electrical centrifuges and their opaque projection apparatus. The Cambridge Scientific Instrument Company showed new patterns of the Féry radiation pyrometer, in use with a Méker furnace, having decided advantages over last year's types; automatic temperature regulators, a newly designed Duddell thermo-galvanometer, and a very sensitive gas-leakage indicator. On the stand of Cassella & Company were Captain Sankey's metal-bending tester, recording rain gages, a gradient telemeter level, Hepworth's electric artificial horizon, and the Hennah & Rissel automatic safety-course recorder and indicator for ships. Harry W. Cox & Company exhibited large induction coils for radiography and wireless telegraphy, a new type of gas interrupter, and wireless telegraphy sets.

Mr. T. Thorne Baker's instruments for telegraphing photographs, shown in operation by the *Daily Mirror*, attracted considerable attention. Elliott Brothers displayed the Wimperis accelerometer and gradient measurer, for which are claimed accurate and rapid measurements of starting acceleration, retardation, comparative braking efficiency, tractive effort at varying speeds, and slopes and gradients. They also had an effective selection of their well-known electrical instruments. Everett, Edgcumbe & Company's exhibit consisted mainly of electrical measurement instruments and photometers. Gambrell Brothers showed some well-designed instruments, including their new "foolproof" pointer and reflecting galvanometer. Mr F. Harrison Glew demonstrated his novel "radium collector" for atmospheric electricity, which can be employed to give time records, operate continuous electric chimes and perform other like operations, also an electric recording inclined plane and spark counter, and an actinium emanation lamp. John J. Griffin & Sons exhibited Sand's apparatus for rapid electro-analysis of metals, a gamma-ray electroscope, and a temperature recorder connected to quartz-glass platinum resistance thermometers.

Notable among Mr. James J. Hicks' instruments were a compensated micro-manometer, indifferent to vibration or changes of level, and a meteorograph for attaching to kites to obtain a record of pressure, temperature, humidity and velocity of wind in the upper air. A. Hilger & Company had a capital display of spectroscopic apparatus, and a 56-plate diffraction grating. The Marconi Wireless Telegraph Company had their improved valve receiver in operation. Nalder Brothers & Company showed a Duddell's patent bifilar vibration galvanometer and Mansbridge's variable standard of self-induction; Nalder Brothers & Thompson, a telethermometer, consisting essentially of a moving-coil ohm-meter combined with a Wheatstone bridge, and a lamp-testing wattmeter. Negrette & Zambra showed meteorological instruments. Newton & Company made an attractive feature of gyroscopes, synchronous springs and pendulums, and the Wilberforce spring.

Mr. Robert W. Paul demonstrated the use of the Sumpner reflecting dynamometer and the Paul single-pivot dynamometer. The Reason Manufacturing Company showed Wright's electrolytic meter, two other forms of meters, and a very ingenious battery-driven clock of very simple and apparently efficient mechanism. Messrs. Ross., Ltd., exhibited excellent cameras and microscopes; Siemens Brothers & Company had an extensive assortment of electrical instruments, including a portable current transformer for accurate laboratory work. Snell & Tinsley had on their stand several instruments embodying new developments, among which were Drysdale's alternating-current potentiometer, the sine flux phase shifter, a phase-shifting transformer, and an improved siphon recorder. The Weston Company made an especial feature of their models 151 and 156 voltmeters and ammeters, for which they claim no inductance error, no discernible working error, no chamber error, dead-beat, and very uniform scale. The mineralogical and metallurgical microscopes shown by Carl Zeiss also deserve mention.

During the afternoon and evening Prof C. V. Boys, F.R.S., gave two demonstrations with soap bubbles, after which Prof S. P. Thompson, F.R.S., showed some combinations of mica and selenite crystals.

The Faraday Society.

The proceedings at the annual general meeting on Nov. 30 were very brief. Dr. Sands' paper on the electrolytic determination of lead as peroxide was read by Dr. F. Mollivo Perkin. The speaker, who opened the discussion, said he quite agreed with Dr. Sands as to the necessity for high density of the peroxide, the employment of high temperature, initial elimination of water by alcohol followed by ether, and drying at 200°. Dr. Seuter briefly pointed out some apparent discrepancies. Dr. Perkin replied that the difficulties which had arisen were due to omission of the working with alcohol and ether. He had not arrived at satisfactory results with alkaline solutions. There was very great difficulty in freeing the peroxide from alkali. With the nitric solution the whole of the lead was deposited in about 20 minutes, but with alkaline solutions a much longer time was required. The other papers evoked practically no discussion.

New Uses for Blast-Furnace Slag.

A new process for the utilization of blast-furnace slag has been worked out by W. P. Thompson Company, of Liverpool. Sand and sodium sulphate, with a little lime if required, are put into the furnace with the result that the slag is converted into a glass, which can be cast into blocks or rolled out, with or without wire reinforcement, into sheets for use as slates, slabs, tiles, building and paving blocks. Where the introduction of sand and sulphate into the blast furnace is not permissible, these materials are added and melted with the slag after this has been run into a reverberatory furnace; but this process demands care to ensure thorough mixing, and entails increased charges for fuel. The inventors claim that they can produce roofing slates of any size, shape or color at less than half the cost of Welsh slates, while this translucency is a great advantage. Building blocks of this material have a polished surface which will not permit adherence of sooty matter and thus always retain a bright appearance, and are, moreover, quite waterproof. The Patent Corporation, of Liverpool, is experimenting with slags from various sources.

LONDON, January, 1910.

Synopsis of Metallurgical and Chemical Literature

Iron and Steel.

Cementation by Carbon has been studied by L. GUILLET and CH. GRIFFITH, their results being published in *Revue de Métallurgie*, October last. Their object was to determine if carbon alone would cement iron by contact in a vacuum where the absence of gases was assured. Special precautions were taken to first remove dissolved gases from the iron and from the carbon used. The carbon was tested simply in contact with the iron and mechanically pressed against the iron. Their conclusions are important.

(a) In a vacuum, pure carbon does not cement iron unless mechanical means are provided to press it into contact with the metal. As the pressure is increased, cementation increases.

(b) Carbon *alone* plays but an insignificant part in industrial cementation processes.

(c) Dissolved gases, from iron or carbon, will promote cementation.

Moisture in Cupola Blast is discussed by Engineer A. MESSERSCHMIDT, of Wiesbaden, in *Stahl und Eisen*, Dec. 1. He calculates that since as much as 15 cu. m (525 cu. ft.) of air is blown in per kilogram of coke burned, that this may carry in wet weather as much as 0.2 kg of moisture, whose decomposition absorbs some 10 per cent of the calorific power of the coke. The damage done cannot be repaired by simply using 10 per cent more coke, because the real factor affected is the combustion temperature before the tuyères, and while this is materially lowered by the decomposition of moisture, it is not regained by increasing the coke. It is, therefore, common experience to find the cupola iron running cold in damp, warm weather, and unable to make sharp, thin castings. Messerschmidt cites a case where the cupola blast was taken from an underground conduit 15 m long, which was often partly full of water, thus giving air to the cupola saturated with moisture. Drying of the cupola blast will save melting coke, and give a more uniformly heated and hotter quality of melted iron.

Corrosion of Iron.—In the very interesting yearly report of the Bureau for Testing Materials of the Institute of Technology of Berlin for 1908-9 an account is given of the results obtained in the researches of E. HEYN and O. BAUER. The authors had formerly studied corrosion of iron in a solution of one special salt in water. They have now extended their investigation to the solution of salt mixtures. One special point is emphasized: In their former investigations the pieces of iron were completely suspended below the surface of the liquid. The speed of corrosion depends in this case on the quickness with which fresh oxygen can come by diffusion to the surface of the iron from the surface of the liquid. If, however, the iron samples are only partly suspended within the liquid, the rust which forms along the line of contact between the iron and the air acts as a transmitting medium for the oxygen downward. It acts like an oxygen sponge. The speed of rusting of the iron below the surface of the liquid depends now also on the nature of the rust; if this is porous or spongy it is a better transmitter of oxygen than if it is dense and compact.

Various cases of corrosion of boiler tubes and other apparatus were investigated. The general public usually blames the steel. But the bureau has repeatedly pointed out that in most cases the cause of strong corrosion is not to be found in the material of the steel, but in the method of operation. For instance, the way in which the feed water is supplied to the boiler may have considerable influence. If the water has an opportunity to absorb considerable quantities of oxygen by saturation with air, the oxygen is set free in form of little bubbles when the air is heated during operation. The oxygen bubbles adhere to the iron surface and produce corrosion. The bureau had formerly recommended the use of charcoal filters in the pipes for the feed water, but no results from practice have yet been reported. In five cases of corrosion of boiler tubes which were investigated the fact was established that the fault was not in the steel; in almost all cases the nature of the water was to be blamed, since very pure water (distilled water) was used. The bureau emphasizes strongly that the distilled water attacks iron in presence of oxygen far more strongly than most ordinary kinds of water. In one case the iron had been painted with a protected paint almost over the whole surface, with the exception of one end which had been left unpainted. This showed strong corrosion and it was found that the corrosion had also progressed from this end further below the protective film of paint.

Rail Steel.—In the same report a comparison of the microstructure of various kinds of steel is reported. Electrically refined rail steel was found free from slag enclosures; all other kinds of rail steel investigated were found to contain slag enclosures.

Ductile Electrolytic Iron.—At the last convention of the German Bunsen Society the Langbein-Pfanhauserwerke A. G. of Leipzig exhibited various samples of electrolytic iron made by the method of F. Fischer, from ordinary "commercial iron" with cathodic current densities up to 20 amperes per square decimeter or more (the composition of the electrolyte being not

stated. This electrolytic iron which has properties quite different from Burgess' electrolytic iron, contains not more than 0.05 per cent impurities (Gumlich gives the following analysis: 0.024 C, 0.004 Si, 0.008 Mn, 0.008 P, 0.001 S.) It is soft, the hardness in the Brinell scale is 92 unannealed, 62 annealed (while silver is 91, aluminium 52). Rolling increases the hardness; when a 3-mm plate was rolled out to 1.2-mm thickness, the hardness increased from 90 to 126; further rolling out to 0.6-mm thickness increased the hardness to 175. After annealing, however, the hardness dropped again at once to 60 or 62. This kind of electrolytic iron is easily worked, cut with the knife, rolled cold, and can be forged. It can be hardened in the whole or in parts by absorption of carbon. The magnetic qualities are excellent. Electrolytic iron sheets in thicknesses from 0.1 mm or less to 8 or 10 mm can be easily made by deposition on a metallic cathode. For steel cuts or electrotypes this electrolytic iron is specially suitable, since it is possible to carry out the process directly in such a way that the front of the cut is hardened, while the back is ductile. Seamless tubes, made by this process, were also exhibited (20 mm diameter, 1 mm thickness), also a lead pipe of 20 mm diameter and 0.5 m in length, armored with an adhesive iron coating of 1 mm thickness. (*Zeit. f. Elektrochemie*, Aug. 15, 1909.)

Electrolytic Steel.—A. Muller (*Métallurgie*, 1909, p. 145) has experimented largely on this subject. As anodes he used various soft steels and soft iron; as cathodes, iron, copper, lead, tin plate. The deposited iron detaches most easily from copper or lead. The best electrolyte was found to be a 30 per cent solution of double iron-ammonium sulphate, worked cold. The best cathodic current density was between 50 amp and 80 amp per square meter (5.5 amp to 7 amp per square foot). It is recommended to work under the following conditions: Use anodes as pure as possible, enclose the anodes in porous cells, replace the electrolyte daily, clean the anodes frequently by brushing in dilute HCl, keep the electrodes quiet and the electrolyte unagitated, protect the surface of the bath from the air by a layer of paraffine, keep the current below the higher limit named, use cathodes carefully cleaned and greased, use an electrolyte free from carbon. A sulphate electrolyte will always give iron deposit with a little sulphur—0.006 per cent when using perfectly pure anodes. The deposits contained also 0.012 to 0.014 per cent of hydrogen and 0.010 to 0.035 per cent of nitrogen. When such specimens were heated they showed numerous points of disengagement of heat, due to the reactions of hydrogen, but these disappeared on repeated heatings. The critical points showed distinctly at 894° and 761°. The fusion point was determined on a specimen containing 0.017 per cent C, 0.089 Si, 0.028 P, 0.025 Mn and 0.037 S, by heating in vacuum in a magnesia crucible, as between 1485° C. and 1525° C.

Recovery of Flue Dust.

Friction System.—Mr. Charles W. Goodale, of Butte, Mont., contributes an interesting note to the discussion of Mr. D. W. Bronton's address on modern progress in mining and metallurgy in the Western United States. It is printed in the January issue of the *Bulletin* of the American Institute of Mining Engineers. Mr Goodale describes the friction system designed for the recovery of flue dust from furnace gases as used at Great Falls. Screening tests on the blast furnace charges of smelting ore, and on the fine concentrate treated in the McDougall furnaces, show a high percentage of fine material, which would naturally be carried into the flues, and at a high velocity of the furnace gases through the flues a considerable loss in dust would inevitably occur.

Before deciding upon the friction system a flue was built 300 ft. long, 4.5 ft high and 4 ft wide, through which the furnace gases could be drawn in measured volume and temperature, and at varying velocities. Two tests were made maintaining a velocity of about 500 ft per minute, and with no obstructions or dust arresters in the flue. The amount of dust was determined per unit volume of gas; then similar determinations were made with baffle plates, Freudenberg plates (of sheet iron, suspended parallel with the direction of the flow of the gases), with numerous contractions and expansions, and with wires (patented by Roesing in the United States in 1890). It was found that the wires gave nearly as high an efficiency of dust recovery as the baffle plates, and with very much less frictional resistance to the passage of the gases. It was then decided to build a dust-chamber of such dimensions that the furnace gases would not have a velocity greater than 500 ft. per minute through it, and to fill this chamber with steel wires.

All the furnace gases from the several departments are assembled through individual flues in an up-take in the furnace building. From the top of this take-up a cross-take leads over the smelter buildings and tracks to the dust chamber. The cross-take is 34 ft. wide and 20 ft. high. The main dust chamber, rectangular in shape, is 367 ft. long, 176 wide, 21 ft. high. Steel-wire netting, 1⅝-in. mesh, is bolted to the I-beams of the roof, and at alternate intersections of the netting wires, steel dust arresting wires, No. 10 gage, weighing about 1 lb. each and reaching nearly to the floor of the chamber, are hooked on. From the entrance to the chamber, and for a distance of about 150 ft., the space is fully occupied by wires; then comes a length of about 50 ft. with no wires, then 150 ft. filled with wires. In the vacant space, air ducts, both from the basement and the roof, are provided, so that the temperature from that point on can be reduced and the condensation of arsenic effected on the wires.

The purpose of this arrangement is to collect, as far as possible, the dust which is carried along mechanically by the furnace gases in the first part of the chamber, which leaves the condensable elements to be recovered in the upper part. Experience has shown that at or below the condensing temperature of arsenic the wires become heavily coated, and it is therefore necessary to shake them. Provision has been made for this. The dust chamber is divided longitudinally by a partition wall, and dampers placed at the lower and upper ends of the chamber make it possible to deflect all the gases through either half if it is so desired. Leading from the dust chamber to the chimney, which is 506 ft. high and 50 ft. in diameter at the top, is a flue 1200 ft. long, 48 ft. wide and 21 ft. high.

In the floor of the dust chamber there are more than 1000 sheet-steel hoppers, arranged in 22 lines, and a complete system of tracks enables the dust to be drawn from any hopper at any time. The cross-take is also provided with hoppers, and a hopper crane draws off the dust and conveys it to chutes leading down into the bottom of the up-take, where there are hoppers from which the material can be drawn into the charging cars for the reverberatory furnaces. There are 1,215,000 wires in the dust chamber, weighing about 608 tons, and nearly 3500 tons of structural steel in the flue system. Recording thermometers and pressure gages have been placed at the entrance and exit of the dust chamber and in the flue near the chimney, so that complete information regarding conditions is available.

Precious Metals.

Assay of Gold Alloys.—At the meeting of the Institute of Metals, Jan. 18, in London, Mr. Ernest A. Smith presented a paper on the assay of industrial gold alloys. In this paper Mr Smith gave a brief description of the methods in general use for the assay of industrial gold alloys, and embodied the results of experimental work carried out some years ago at the Royal School of Mines, and more recently in the Metallurgical Department of the University of Sheffield. The alloys dealt with are those of definite composition or "standard" alloys specially prepared for industrial purposes, and also those of more or less indefinite composition resulting from the treatment of scrap, lemel, sweep, and such substances obtained during the processes of manufacture.

After giving examples of the composition of some of the alloys in general use by goldsmiths and jewellers, which are

mainly ternary alloys of gold-silver-copper, Mr. Smith discusses the various methods of sampling and the precautions to be taken in the case of alloys in which segregation has taken place. Attention is drawn to the importance of removing the "colored" surface resulting from the "pickling" before taking the samples for assay. When this "colored" or enriched surface is included in the sample very serious errors may be introduced into the assay results in the case of sheet metal and wire.

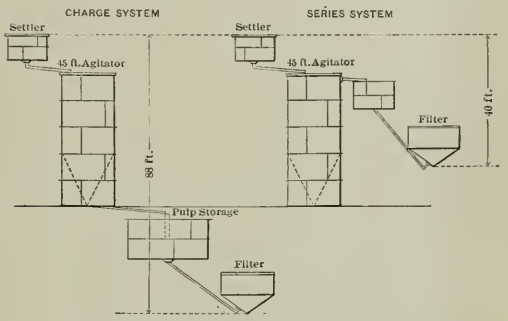
The question of the best proportion of lead to be used for the cupellation of gold alloys is then considered, attention being drawn to the fact that copper has a greater affinity for gold than for silver, consequently a larger proportion of lead is required to ensure its oxidation when combined with gold than when combined with silver. It is also pointed out that the separation of copper from gold is greatly facilitated by the presence of silver.

The quantities of lead used by different assayers experienced in assaying these alloys are given in a table. Brief reference is made to the methods in general use for charging the assays into the furnace, and the results of experiments on adding the lead in two separate charges (viz.: adding one-third after the other two-thirds have worked off).

The chief item of the paper is the comparison of the assay methods used for industrial alloys, based on experimental work, and it is shown that the results for the gold assay are invariably higher when determined by direct-cupellation with parting silver than when determined indirectly by double cupellation. The importance of using check assays in all cases where extreme accuracy is desirable is emphasized by Mr. Smith. The method of double cupellation, or "parting-assay," in which the gold and silver are determined in the same assay piece, is largely used by trade assayers for the valuation of lemel bars, and like substances, and in the hands of experienced assayers the results are sufficiently accurate for commercial purposes. In making assay by this method, it is well to remember that it is the weight of the gold which is the most important, because of its value, and the working conditions should be such that no undue loss of gold is occasioned. The different methods of reporting assay results are referred to. The decimal system is now very generally adopted for reporting gold assays, but the older "carat" system is still in use, as the generality of workers and dealers in gold and silver are more conversant with this system of computation than with the decimal equivalent. (From advance sheets.)

Cyaniding Slime.—In the January issue of the *Bulletin* of the American Institute of Mining Engineers, Mr. MARK R. LAMB discusses the two methods of treating pulp in air-agitation tanks. The usual method is to fill each tank in turn, agitate the mixture the required period and then discharge the treated pulp into a storage tank, from which it is drawn to the filter as required. A second and later method is to run the pulp from the tube-mill, classifier, thickener or settler into the first of a set of tanks, and thence continuously or in series through the remaining tanks, finally drawing off the pulp to a continuous filter, or to a pulp-storage tank if the filter is of the intermittent type. Mr. Lamb emphasizes the advantages of the latter method and mentions that in the design of a recent cyanide plant the estimated cost of agitation tanks, storage tanks and pumps was 30 per cent higher for the charge-agitation arrangement than for the series plan. An objection which might be raised against the series method is that some of the pulp might get through with little or no treatment. The author shows by simple approximate calculation that this is out of the question and that it is physically impossible for any particle of the pulp to go through the series of tanks in less than a certain minimum period. This minimum period gives the series method a big handicap, and one which increases directly with the number of tanks used. In fact, the resulting low circulation velocity makes the operation resemble the treatment of pulp in a horizontal tube of the same diameter as the agitation tanks and of a length equal to the combined lengths, having the agitation at right angles instead of parallel to its length.

Mr. Lamb mentions some interesting results obtained in tests on a small scale, with tanks of a capacity of 100 lb. of dry slime in a 2-to-1 pulp. With pulp containing coarse sand (and even with fine sand in a thin pulp) care must be exercised in drawing the pulp from one tank to the next in series, otherwise either the sand or the slime will accumulate. If the drawing-off pipe is merely flanged to the tank, the outflowing pulp will contain a greater proportion of slime to sand than is



CHARGE AND SERIES SYSTEMS OF CYANIDING SLIME.

contained in the average of the pulp, since during agitation by this method the sand has a tendency to sink through the pulp as soon as discharged through the central air-lift, rather than to work out to the sides of the tank with the flocculent slime. This difference in pulp flow is greater with a thin mixture and less with a thick one, and can be utilized practically in a very simple manner if it is desired to agitate the sand longer than the slime. A concentration of sand in the tank will be attained by drawing the slime from a quiet point in the circumference of the surface of the pulp. To do this, a discharge pipe, pivoted at the side of the tank, is arranged to swing in a horizontal plane and to be placed and held at any desired point in its arc between the central pipe and the side of the tank. After the desired initial concentration is attained, this discharge pipe is then placed so as to draw off a mixture of the same proportions of sand and slime as that being fed to the tank, which will maintain the charge in the tank at a higher ratio of sand to slime. Thus the slime will go through faster and the sand slower than the average pulp flow.

To illustrate this point, imagine a tank containing 100 tons (dry weight) of pulp consisting of 3 of sand to 1 of slime. Admit a stream of pulp carrying 100 tons in 24 hours, which consists of half sand and half slime, drawing off an exactly similar stream. The slime which was originally in the tank (25 tons) will be replaced twice in 24 hours, by the 50 incoming tons, while the original sand (75 tons) will be replaced in only two-thirds time. In other words, the slime will receive a 12-hour treatment and the sand will receive a 36-hour treatment. Mr. Lamb considers this one of the important advantages of the "series" system.

A point of great advantage possessed by the series method in the design of many plants is the saving in difference of level required in the site. The agitation tanks, if used in this manner, need not require more than from 2 to 4 ft. difference in level between the inlet and the outlet, depending to some extent upon the number and size of tanks and the distance apart, but also and mainly upon the margin of capacity required to provide for fluctuations in pulp feed and discharge. This margin depends largely upon the type of filter used. Ordinarily, the pulp will be

drawn from near the top of the last tank, while in the "charge" system the outlet is at the bottom of the tank. The accompanying figure shows this difference.

In the series system no storage tank is needed below the last agitator tank if a continuous filter, such as the Oliver or the Ridgway, is used; only a small tank of a capacity to supply pulp to equal displacement of a loaded basket of leaves if a Moore filter is used, and a slightly larger tank (sufficient to fill the filter box with the unloaded frames in place) with a Butters filter. On the other hand, in the charge system the storage tank, even with continuous filters, must have at least a capacity equal to that of an agitator, and with either the Moore or the Butters filter must have a slightly greater capacity.

Continuous Collection of Sand for Cyaniding.—With respect to Mr. W. A. Caldecott's paper, abstracted in our January issue, page 42, Mr. C. O. Schmitt makes some eulogistic remarks in the *Journal* of the Chemical, Metallurgical and Mining Society of South Africa of November, 1909. He thinks that Caldecott's process represents a real advance in the treatment of sand. The satisfactory collecting of the sand contained in the final pulp leaving the battery, preparatory to treatment, is by no means a simple proposition. It is obviously necessary, in order to facilitate treatment, to collect the sand free from all slime, but this is not by any means always done, the slime contents of collected sand being sometimes as high as 10 per cent. Slime pulp should be treated in the plant provided for the purpose, and since the treatment of slime is, as a rule, less expensive than the treatment of sand, any slime retained in the collected sand not only costs more to treat than it otherwise would, but it also displaces sand which could be handled in its place, quite apart from the fact that the presence of slime in the sand-collecting vats interferes with the treatment. Mr. Schmitt emphasizes the excellent results obtained on the Simmer and Jack plant, using a 15-ft. table, although it is only of an experimental nature. The objection might have been raised to the introduction of further machinery into the already complicated reduction plant, but Mr. Caldecott has disposed of any such objections by pointing out that the continuous running of the filter table presented no difficulties; so much so that the arrangements made for temporarily reverting to the old style of working, in case of temporary failure, have not so far been needed.

The four chief advantages of the Caldecott process are summarized as follows: The sand collected is cleaner and contains less slime than is the case in the ordinary plant. The capital cost of the sand plant has been reduced by nearly 40 per cent. The space occupied by a sand plant has been reduced. The time of treatment is reduced to about two-thirds of that now occupied. The question of capital expenditure will appeal to most, and since the saving of 40 per cent on the sand plants amounts to about 10 per cent on the whole reduction plant the result accomplished is worthy of considerable praise. The saving on capital account may be still further increased by providing only one set of treatment vats, and while this may not be acceptable for very high-grade propositions there will be many instances where it could be justified.

The tendency all over the world is toward all sliming, and in view of the ease with which the average slime is treated, this is, in many cases, quite justified. "In view of the result obtained it is obvious that Mr Caldecott has done quite a good deal toward the desired end, and (in the writer's opinion) more than those who propose to spend a shilling in crushing and grinding, to save sixpence in the cost of the succeeding treatment. We are not by any means at a stage where the sand plant can be discarded," and in this connection the writer refers to an all sliming plant erected by Mr Charles Butters in Mexico which in spite of being intended as an all sliming plant contains a sand plant. The reason is that upon work having been started, it was found that the product from the crushing and grinding plant did contain what is usually termed sand and which did not yield sufficient of its precious metal contents.

Copper.

Cartridge Brass is the subject of a very interesting study by Capt. Grard, published in the *Revue de Métallurgie*, October, 1909. As is probably generally known, this brass must sustain great mechanical strains in being stamped or drawn into shape, and only the best copper and the very finest zinc can be used in it. It is usually made with 33 per cent zinc. Exhaustive mechanical tests were made with the brass as cast, worked, partially annealed and annealed, while an extensive microscopic study was simultaneously made. It was determined that from 0° C. to 250° C. this brass possesses its minimum of malleability; when worked, the structure showed broken octahedral crystals. Heated to from 250° to 400°, the normal properties commence to appear. Between 400° and 700° is a range for semi-annealing; heated between these limits the tensile strength diminishes gradually, but the elongation before rupture increases. Complete annealing is attained by heating to between 750° and 830°, but the upper limit must be approached circumspectly; slight impurities in the metal and possible errors in determining the temperature both make it advisable to keep below 800°. Annealing above this temperature causes a rapid falling off in the quality of the metal.

Failure of Brass Loco-Tubes.—At the meeting of the Institute of Metals, in London, Jan. 19, Mr. T. Vaughan Hughes presented a paper on the failure in practice of non-ferrous metals and alloys, with particular reference to brass loco-tubes. The paper describes an investigation into the cause of a breakdown of brass loco-tubes which led to serious casualties and Government inquiry. The breakdown occurred at the firebox end of the tubes nearest the firebox. The tubes were not corroded externally, but were thus affected internally, mostly along a semi-diameter. The corrosion appears to be the result of the action of volatile sulphur compounds from the fuel used (coke) in presence of hot reducing gases, which may have volatilized the zinc from the internal surfaces of the tube alloy. Condensed water vapor of an acid nature cleared the tubes during the morning stoking and at other times, allowing a repetition of above actions. Thus the tubes wore down internally until too weak to withstand the working pressure in the boiler. The action was by the author's further investigations found to be facilitated by the peculiar nature of the "scale," only 1/20 of a millimeter thick, which offered a resistance of 20 megohms to an e.m.f. of 250 volts when dry, and 150,000 ohms when wet.

This electrical measurement was taken as being indicative of the heat conductance of the "scale," and explains an overheating of the tube at the seat of breakdown and other parts. The "scale" consisted largely of silica, magnesia and ferric oxide, combined with a small amount of sulphuric acid initiated and held together by a soap of iron, aluminia and magnesia. The breakdown only occurred when the boilers were fed with untreated water supplied from a peaty source. Analysis of the "scale" and other data are briefly given. (From advance sheets.)

Aluminium.

Analysis of Aluminium and its Alloys.—A paper by Dr. R. Seligman and Mr. F. J. Willott was presented at the annual meeting of the Institute of Metals, in London, Jan. 18. The rapid development of the aluminium industry makes it desirable that everything appertaining to it should be standardized, among other things the methods of analyzing pure aluminium and its alloys. The technical methods described in this paper have been in use for a number of years, and it is hoped that they may form the basis for a future standard course of analysis. The aluminium of commerce contains from 0.15 to 0.6 of iron, 0.2 to 0.7 of silicon, and 0.02 to 0.06 of sodium, together with traces of carbon, oxygen and nitrogen, while occasionally small quantities of copper are found. The purity of the metal varies from about 98.5 up to more than 99.75 in exceptional cases. Owing to the impossibility of estimating the aluminium itself with sufficient accuracy, it is customary

to determine all the impurities and to calculate the purity of the metal from the figures so obtained. In most cases, only those impurities present in appreciable quantities are estimated, a course which has been fostered by the general ignorance of the influence of the minute quantities of the ingredients thus ignored on the properties of the metal. It is hoped that information will shortly be forthcoming as to the effect of these materials on the metal, so that the methods will have to be modified in this respect. A detailed description is given of methods of estimating silicon in its various forms, iron and sodium, together with figures indicating the accuracy of the various determinations.

The alloys of aluminium in common use contain, in addition to the impurities cited above, copper, zinc, nickel and magnesium, and as an additional impurity, lead. Alloys containing tin are met with in the form of solders or soldered joints. The methods which the authors use for determining these ingredients are described. Occasionally manganese, titanium and tungsten have to be dealt with, and the determination of these elements is briefly referred to, because the authors are of the opinion that the closer study to which these alloys are being submitted will enhance their importance and promote their wider introduction. (From advance sheets.)

Alloys.

Phosphor Bronze.—A paper by Messrs. O. F. HUDSON and E. F. LAW, on phosphor bronze, was presented at the meeting of the Institute of Metals, in London, Jan. 18.

The object of the authors' work is to indicate the relation between the mechanical and other properties of the copper-tin-phosphorus alloys and their constitution and structure. Copper and phosphorus combine with the formation of a definite phosphide of copper, which forms with copper a binary eutectic, and a ternary eutectic with copper and the compound Cu_4Sn. With less than 4.5 per cent of tin the slowly cooled alloys consist of copper holding in solution the tin and, perhaps, a small amount of phosphorus, and of copper phosphide in rather large but fairly uniformly distributed masses, there being no sign of the typical eutectic structure. With more than 4.5 per cent of tin the ternary eutectic, having the composition 81 per cent copper, 14.2 per cent tin, and 4.8 per cent phosphorus, and a melting point of 620° C., makes its appearance.

A diagram was given showing the constitution of all the alloys containing up to about 25 per cent of tin. For practical purposes the phosphor bronzes may be divided into two classes, (1) the malleable phosphor bronzes containing relatively small amounts of tin and phosphorus, not more than about 6 per cent of tin and 0.3 per cent of phosphorus, and (2) the cast phosphor bronzes containing in some cases as much as 12 per cent of tin and 1.5 per cent of phosphorus. The malleable phosphor bronzes consist of a homogeneous solid solution of tin and phosphorus in copper, or they may contain some phosphide of copper in the form of small rounded particles evenly distributed throughout the mass of the alloy. The cast alloys contain the ternary eutectic with some phosphide of copper. The ternary eutectic is exceedingly hard, while at the same time it possesses a good crushing strength, and it is to this constituent that the phosphor bronzes owe their property of resisting wear. The paper includes notes on the examination and analysis of the phosphor bronzes, and is illustrated by a number of photomicrographs illustrating the structure of the alloys. (From advance sheets.)

Copper-Arsenic Alloys.—A paper by Messrs. GUY D. BENGOUGH and B. P. HILL, on the properties and constitution of copper-arsenic alloys was presented before the Institute of Metals in London on Jan. 19. The paper is divided into two parts; the first deals with the mechanical properties of the alloys of industrial importance, the second with their chemical constitution.

In the first part the authors begin by stating the most important purposes for which these alloys are used, namely for locomotive parts and for dynamo brushes. They also give a statement of what are generally considered to be the peculiar characteristics of the alloys as compared with pure copper, namely, greater strength and rigidity, and greater resistance to the action of flames, i. e., of gases at high temperatures. The best proportion of arsenic to be used to insure maxima of the advantages named was shown to be a matter upon which opinions differ widely, and one of the principal objects of the paper was to bring forward data for an authoritative pronouncement on the matter. For this purpose five 1-in. round copper bars containing amounts of arsenic varying from 0.04 per cent to 1.9 per cent were prepared under ordinary industrial conditions by the Muntz Metal Company, and were thoroughly tested by the authors in three conditions, namely: (1) As rolled; (2) after being heated for various lengths of time in an oxidizing atmosphere; (3) after being heated for similar times in a reducing atmosphere. The reducing gases used in the third set of tests were coal gas, hydrogen and carbon monoxide. Since these gases are liable to come in contact with the copper-arsenic alloys used in practice in locomotive firebox construction, it was thought that an interesting light might be obtained on the causes of deterioration and failure of the alloys in practice.

Upwards of a hundred tensile tests were made to ascertain the exact conditions under which the mechanical properties of bars containing 0.04, 0.26, 0.75, 0.94, 1.94 per cent of arsenic respectively were spoiled by contact with gases at high temperatures. Briefly, these conditions were found to be: (a) Alloys containing between 0.04 and 1.0 per cent of arsenic were ruined by the action of reducing gases for three hours at 700° C.; in some cases a temperature as low as 600° C. was sufficient. (b) Similar alloys were not harmed by identical treatment in an oxidizing atmosphere, nor when the temperature was raised to 900° C. (c) Alloys containing 1.9 per cent of arsenic were not harmed by treatment in either an oxidizing or a reducing atmosphere.

With regard to the best percentage of arsenic to use in these alloys, the authors found that the alloying metal when present in quantities less than 2 per cent tends to increase slightly the tensile strength and the elongation of their alloys. They also consider that the homogeneity and the resistance to the action of reducing gases increase with increased percentage of arsenic up to 2 per cent.

The authors describe a series of experiments carried out to elucidate the *modus operandi* of the reducing gases in spoiling the alloys, and put forward tentative suggestions as to the means which might be taken to produce bars which would resist the action of reducing gases.

In the second part of the paper the authors describe the work undertaken to elucidate the constitution of these alloys, and give a diagram illustrating it under certain defined conditions. They confirm the existence of the compounds Cu_3As, and Cu_3As_2, already proposed by Friedrich, and deny the existence of the compound Cu_2As, proposed by Hiorns. (From advance sheets.)

Electrochemistry.

Fixation of Atmospheric Nitrogen.—In *Zeit. f. Elektrochemie*, Jan. 1, a review is given by F. HABER and A. KOENIG of the present status of fixation of atmospheric nitrogen. The following table sums up the results obtained with the three successful processes in commercial operation; the process of the Badische Company (Dr. Schönherr) (see our vol. VII, p. 245), the process of Birkeland and Eyde (see our vol. IV, p. 126 and VII, p. 304) and the process of Salpetersäure-Industrie Company (Pauling) (see our vol. VII, p. 430). The table gives both the yield in grams HNO_3 per kilowatt hour and the con-

	Grams HNO_3 per kw-hour.	Concentration in per cents NO.
Badische Company	75	2.5
Birkeland and Eyde	70	2
Pauling	60	1 to 1.5

centration in per cents of NO, since both items are of fundamental commercial importance.

The comparison with theoretical figures is difficult. The commercial yield of the process of the Badische Company is about 63 per cent of the theoretical value obtainable (without regeneration of heat) on the hypothesis that the oxidation of the nitrogen is a purely thermal process and that the temperature of the arc is 3800 deg. C. The authors give a very useful review of the various scientific investigations made in recent years on the mechanism of the oxidation of atmospheric nitrogen by electric discharges through air.

Calcium Carbide and Acetylene Tetrachloride.—A report by J. H. Vogel on the present situation of the calcium carbide industry in Europe is given in abstract in *La Revue Électrique*, Nov. 15, 1909. While in the past the consumption of calcium carbide in Germany has increased from year to year (30,340 tons in 1906, 34,916 in 1907, 39,180 in 1908), there is a decrease to 38,000 tons in 1909, due to the introduction of electric instead of acetylene lighting on the cars of the Prussian railways. With this single exception the consumption of carbide has further increased. During the past year dissolved acetylene (see our Vol. I, page 274) has been introduced in Germany. The manufacture of acetylene tetrachloride is now carried out on a large scale by the simple addition of chlorine to acetylene, without using any further compounds. It contains about 85 per cent of chlorine and 15 per cent of acetylene, and its manufacture is advisable where cheap chlorine is available. Since 1 kg of commercial carbide yields 1/3 kg of acetylene (285 liters) there are needed 85 kg of chlorine for 45 kg of carbide to produce acetylene tetrachloride. This is an excellent solvent and is used as a substitute for benzine and like materials, its great advantage being its non-inflammability. It is also made in Jaice (Bosnia) and in Runcon (England); at the latter place a plant capable of producing 1000 cu. m of acetylene per day has been erected. At both places, the process of the Consortium f. elektrochem. Ind. is employed. Another process has been worked on an experimental scale in Neustassfurt and is about to be started on a large scale. A derivation of acetylene tetrachloride is trichlorethylene, C_2HCl_3, which is a substitute for carbon tetrachloride. The latter has the disadvantage that it attacks iron at elevated temperatures. Trichlorethylene is made in Jaice in Bosnia and sold in carloads. The long experiments on the utilization of acetylene for the manufacture of lamp black appear to have been successful; in a plant at Friedrichshafen 1000 tons of carbide will be treated per year. This plant has a contract with the Zeppelin Company for the supply of large quantities of hydrogen for airships.

Bleaching Liquor by Electrolysis with Magnesium Cathodes.—The direct production of hypochlorite or bleaching liquor by electrolysis of alkali chlorides, depends on a good mixing of the anodic and cathodic products of electrolysis. The efficiency of this process is reduced by the production of chlorate and anodic oxygen and by the reduction of hypochlorite at the cathode. To prevent the latter the only remedy is to keep the ClO ions away from the cathode. By a well-known process of Imhoff and Müller this is accomplished by an addition of chromate to the electrolyte; this has such an effect that a very thin yet effective diaphragm forms automatically at the cathode surface thus preventing the ClO ions from making contact with the cathode. Betts and Sherry have proposed the use of magnesium cathodes. In the October, November and December issues of 1909 of *Elektrochemische Zeitschrift* M. F. Schmidt gives an account of an investigation in which he studied the reason for the good effect of the magnesium cathode. It is again the automatic formation of a very thin diaphragm right on the cathode. As soon as the magnesium plates are inserted in the solution they are covered with a film of magnesium hydroxide which prevents the ClO ions from making contact with the cathode. From a commercial point of view it is to be considered that on account of this film the internal resistance of the cell is increased. With a current of 1.6 amp. in the author's experiments, the voltage was 1.1 volt higher with the magnesium cathode than with one of platinum. Further the formation of chlorate increases with increasing concentration of ClO. With respect to the efficiency, the use of a magnesium cathode differs from that of a platinum cathode of the same size in so far as the same or a somewhat higher yield of active chlorine may be obtained with magnesium by more watts in a shorter time. But by increasing the magnesium cathode it is possible to decrease the voltage (and therefore the power) and nevertheless obtain a high efficiency.

Ozone.—According to a note taken from *Revue d'Électrochemie et d'Électrométallurgie* and printed in the London *Electrical Engineering* of Dec. 30, ozone prepared by electric discharges through air is sometimes found to contain oxides of nitrogen, and Boujeau has noticed the action of these oxides upon the lead pipes which conducted the ozonized air in water purification works. As a result, nitrate of lead has been found to contaminate the water, which is naturally a serious matter. For this reason it is recommended never to employ metallic pipes for conveying the ozone from the ozonizers to the sterilizing plant. As an indication of further progress made in the industrial sterilization of water by ozone, it is stated that at the town of Chartres, in France, drinking water is now obtained by purifying, with ozone, water from the Eure River. This contains originally from 3000 to 16,000 bacteria per cubic centimeter, but treatment with 2 grams of ozone per cubic meter of water renders it practically sterile. The installation, employing the Otto system, is a double one, each part being capable of sterilizing 6000 cu. m (212,000 cu. ft.) of water in 24 hours. A 40-hp steam engine raises the water a height of 11 m (36 ft.) and drives a 500-period 250-volt alternator. The voltage is then raised by a transformer to 20,000. The ozonizers have glass plates as dielectrics, with tin foil between them, and the air is passed between the plates; 50 watt-hours are required to produce 2 grams of ozone, and as this is sufficient to sterilize 1 cu. m of water, 12 kw are required for the 6000 cu. m in 24 hours. A project is stated to be on foot to sterilize Paris water in a similar manner, when it is hoped it will be possible to set up a plant which will be capable of sterilizing 1 cu. m of water with an energy consumption of 10 watt-hours.

Electric Vacuum Furnace Installation.—In our vol. IV, p. 223 (see also vol. VII, p. 427), an electric vacuum furnace of Arsem for experimental work was described with a vertical graphite helix as heater, in the center of which is the crucible. The "radiation screen" (a device for diminishing the loss of heat by direct radiation from the heater) is an annular graphite box, filled with graphite powder which is a poor conductor. In the January issue of the *Journ. of Ind. & Eng. Chemistry* of the American Chemical Society Mr. W. C. Arsem describes some modifications of this design. A larger size of the vertical type differs only in size and in details from the above. In the horizontal tube furnace the heater is a graphite tube, held in graphite clamps with a tight sliding fit, to permit expansion and contraction. Finally in the "box type" of tube furnace the chamber is a rectangular gun-metal box with a removable cover. The heater is made up of four graphite grids, connected in series. These grids are made by sawing slots in graphite slabs 5 in. x 18 in. x 3/4 in. The heated space enclosed by the grids is large enough to take a box-shaped crucible 4 in. wide, 12 in. long and 8 in. high, or 384 cu. in. The radiation screen is made up of eight box-like sections, each hollow and filled with graphite powder, the top section being removable. The research laboratory of the General Electric Company has two installations of vacuum furnaces each containing four furnaces with the necessary transformers, pumps, etc. A very convenient installation for small scale experimental work would consist of a small vertical-type tube furnace, a 10-kw flux-shunt transformer, a Geryk pump and a 15-kw wattmeter.

Electric Furnace Temperature Regulation.—A contrivance to maintain a regular temperature in an experimental electric

Physique by Mr. L. Kolowrat (also in London *Electrician*, December 31, 1909). In Fig. 1 is a resistance furnace fed by current from the mains *A*.

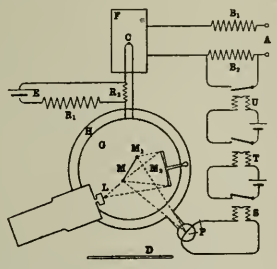


FIG. 1, APPARATUS FOR TEMPERATURE REGULATION.

The thermo-couple *C* is connected to a Le Chatelier galvanometer *G*, which carries on its movable part, in addition to the concave mirror *M* indicating on the scale *D*, a thin, flat glass mirror, 4x5 cm. in size. *L* is the filament of a Nernst lamp placed vertically, M_2 a spherical mirror with a gilt outer surface, and *P* is a Rubens thermoelectric cell, consisting of 12 copper-constantan elements. The position of the mirrors is so arranged that, when the temperature of the furnace reaches a determined point and the arrow of the galvanometer consequently registers at a certain angle, the rays of the lamp are focused on the junctions of the thermo-electric cell *P*. The current generated in the cell acts on a system of three relays, *S*, *T*, and *M*, which operate successively. By this means a resistance is introduced into the circuit *A*, and the heat of the furnace is consequently diminished. When this happens the deflection of the galvanometer decreases, the concentration of rays is removed from the cell *P* and the relay system shunts B_2.

The range within which the temperature of the furnace may be confined may be decreased by increasing the sensitiveness of the galvanometer.

Randall in the *Physical Review* (Vol. XLV, page 142) has described a furnace, the regulation of which can be accomplished by means of a thermo-couple attached to a galvanometer. This latter acts as a relay, but the principle involved is quite different from that of the apparatus just described.

Chemical Engineering.

Concentration of Sulphuric Acid.—In the *Chemiker Zeitung* of Jan. 11, Dr. G. LÜTTGEN remarks that the hopes which have been placed in the contact process for making sulphuric acid have not been completely fulfilled in the course of the last years. It had been wrongly supposed that the lead chamber process was doomed. The numerous difficulties of the contact process have not yet been overcome. The process is still very sensitive. On the other hand, various improvements have been made in concentration apparatus for chamber acid. These apparatus not only serve for increasing the concentration of the chamber acid (in some cases up to 98 per cent H_2SO_4), but they also permit the regeneration of the weak acids formed in some processes.

The first system discussed makes use of two series of 20 porcelain pans, arranged stepwise. The acid passes from one pan into the next. At the lower end a coke fire is provided by which the pans are heated. This system had the disadvantage that the porcelain pans were very liable to break. It has been improved by the use of pans of acid-proof metal or fused quartz; the loss by breakage has been reduced, but it still exists. The disadvantage of these pans is that they permit acid to trickle through, since after a certain time microscopically small cracks are formed. During the course of the last few years a coke filter has been added to this system. The acid obtained by this treatment is clear and may be easily concentrated to 92 or 93 per cent H_2SO_4. The consumption of coke is fairly high, about 150 kg (330 lb.) per ton of 92-93 per cent H_2SO_4. To pass the acid vapors into the coke filter, a small ventilator is used; the power required is small, amounting to ¼ hp for one hour and one ton of concentrated acid.

In the second system described a tower is used into which hot gases enter at the bottom, while at the top the chamber acid (52 to 53 deg. Baumé) is introduced in the form of a spray. Under the effect of the heat the acid mist is concentrated and the acid passes out of the tower at a concentration of 92 to 93 per cent. The steam mixed with acid escapes at the top. The ("volvic" stone) tower may have a height of 7 to 15 meters (23 to 49 ft.). The acid is injected in the form of a spray by means of a lead injector with a circular end-piece of cast iron with a nozzle in the center. This nozzle must be adjusted to the desired degree of concentration of the acid. It is therefore always necessary to change the nozzle whenever the degree of concentration of the acid is to be changed. There are therefore different nozzles used for 78 to 80 per cent, for 92 to 93 per cent and for 96 to 97 per cent acid. The inside of this injector contains a part in the form of a spiral which ends at the front in a point, and at the back in a cylinder. When the chamber acid is forced through this spiral passage, it assumes a rotating motion and leaves the nozzle in form of a rotating mist which fills the whole inside of the tower.

The weak point of the method is that a very pure acid is required which is taken from the second chamber and must be filtered very carefully, since any small impurity would clog the spray nozzle, or, at least, clog it partially. In the latter case the acid is not set into rotary motion, but is thrown in a straight thin stream against the tower walls. The acid becomes turbid and colored. In such a case the apparatus must be stopped and the spray nozzle must be cleaned. But this takes some time, since the attendant will not become aware at once of what has happened and a large quantity of dark acid may have formed meanwhile.

To remedy this the chamber acid is passed through a large sand filter, but this is not always successful. For the condensation of the acid vapors two filters are connected to the apparatus. Both may be filled with coke, but the second is generally filled with quartz. Between the tower and the first coke filter there is inserted a nine-meter lead column to cool the acid vapors and condense them partially. The movement of the gases is obtained by a ventilator at the end of the apparatus. The temperature in the upper part of the tower is 120 deg. to 140 deg. C., while the temperature in the upper part of the lead column is 90 deg. to 100 deg. C. This apparatus with its latest improvements requires 110 to 120 kg of coke per ton of 92-93 per cent commercial acid. The power required for operation is 0.75 hp per hour per ton of concentrated acid. The output of a plant of usual size is 10,000 kg of 92-93 per cent acid in 24 hours. The quantity of acid from the coke filter is also quite considerable. In 24 hours about 2000 kg of acid at 10 deg. Baumé and 500 kg of acid at 14 deg. Baumé are obtained. The price of such a plant when completed is 40,000 to 50,000 marks ($10,000 to $12,500). Since there are only a few plants of this kind in use and these have been in operation only a short time it is too early to give exact figures regarding the cost of maintenance, but the author thinks the repairs will be somewhat expensive. The third system discussed by the author (an improvement of the old Kessler system) will be dealt with in our next issue.

Pyrometry.—In the work on pyrometry, carried out by the Reichsanstalt in 1908, as noticed in London *Engineering* of Dec. 17, 788 thermo-couples were tested; 691 of these were of the Le Chatelier platinum, platinum-rhodium type. The electromotive force of these couples decreases when the couples are exposed to very high temperatures for many hours, and the whole length of wires for a new couple was therefore wound on a magnesia tube, without allowing the two wires to come into contact, and heated to 1400° C. in an electric furnace. Carbon, silicon and iridium, which had been held responsible for the change, were absent in these experiments.

It would appear that some of the rhodium evaporates and settles on the platinum wire, causing a change in the electromotive force, which, at the end of four hours, was equivalent to 35° C. This confirms an observation made by W. P. White in 1906, who found rhodium in the originally pure platinum wire. The effect is local and confined to those portions which are exposed to the highest temperature; that would explain the dependence of the electromotive force upon the depth to which the couple is inserted. The number of constantan-silver couples tested by the Reichsanstalt has diminished and there is a prejudice against their constancy. Repeated tests, during which such couples were heated for 12 days, finally to 600° and 650° C., did not bring out any discrepancy; the indications remained constant within 1 deg. The want of constancy may be due to the arrangements used in engineering practice. The standardization of the Wanner optical pyrometer (35 of these were tested) against black bodies by means of the Koenig and Lummer-Brodhun pyrometer is difficult, on account of the temperature determination in the black body by the aid of thermocouples. In the work on Seger cones, iridium, iridium-ruthenium couples and a tubular iridium furnace are used. Twenty cone temperatures have now been fixed; cone No. 20 has a softening point of 1530° C. But the research had to be interrupted because the iridium tube had to be repaired again.

Alcohol From Sawdust.—Concerning the manufacture of alcohol from sawdust in a plant at St. Marcel, Department of Ardeche, in Southern France, a recent consular report of Frank H. Mason, of Paris, states that this is a new, up-to-date concern and employs a process which consists in exposing sawdust to sulphurous acid gas, whereby the alcohol is generated and extracted without being mixed with or contaminated by the sulphur, as had always happened when sawdust was macerated in a solution of sulphuric acid under the older processes. Consequently it is claimed that alcohol made by the new process can be taken internally or used for any other purpose to which pure alcohol is usually applied.

One metric ton (2204 pounds) of sawdust yields by this process 100 liters (27.47 gallons) of alcohol, 20 kilograms (44 pounds) of acetic acid, and the residue or spent sawdust is pressed into briquettes and used as fuel.

Recent Metallurgical and Chemical Patents

Non-Corrosive Dead-Soft Open-Hearth Steel.—It has been known that in the last years the American Rolling Mill Company, of Middletown, Ohio, has developed a process for making in the open hearth a very pure steel which they have placed on the market under the name of "American ingot iron" and for which the important claim is made that it resists corrosion successfully. Exact information on the process is now available from two patents recently granted to Mr. Robert B. Carnahan, Jr., the superintendent of the American Rolling Mill Company. The company is working under license from the International Metal Products Company, of Newark, N. J., to which both patents are assigned. One (940,785) is a product patent, the only claim referring to "improved rolling mill and bloomery ingots characterized by a deoxidized and degasified slagless body crystalline in structure and containing at least 99.80 per cent of iron, and containing not over 0.14 per cent of sulphur, phosphorus, carbon, manganese, and silicon taken in the aggregate, and containing not over 0.05 per cent of oxygen." The other patent (940,784) refers to the process of making this steel. A basic-open hearth furnace is charged with pig iron or cast iron or molten iron. In practice there has been substituted low-carbon iron or steel scrap to a considerable extent, in some cases as high as 80 per cent of the charge. Lime or limestone is charged to reduce sulphur and phosphorus, these agents being preferably charged prior to charging the metal. The charge is then refined, as in making soft steel, by the employment of ore or agitation, or both, but recarburization at this stage is not necessary in case the charge melts specially low in carbon. Carbon, manganese, sulphur, and phosphorus are thereby reduced. This refining operation is carried out for a much longer time, from one to four hours more, than is usually employed in making high-grade soft steel; naturally, while the carbon is being more and more reduced, the melting point of the iron bath rises and the temperature must be gradually raised, until it is finally at least 2850° Fahr. The refining operation is considered finished when by bath test the bath will contain not over 0.14 per cent of sulphur, phosphorus, carbon, manganese and silicon taken in the aggregate. For the best results it is desirable and practicable to reduce this aggregate to less than one-half the percentage named, and to have neither carbon nor manganese over 0.02 per cent.

The refined metal is poured into molds while still at high temperatures, and while still molten it is deoxidized and degasified to such extent that the resulting product will contain not over 0.05 per cent of oxygen as determined by the method of Ledebur, and preferably considerably below this, and will contain at least 99.80 per cent of iron. "The deoxidation and degasification may be very satisfactorily accomplished by the addition of aluminium in the molten metal, say, 2½ lb. per ton. If done in the furnace it is recommended that aluminium pig iron may be used." Aluminium may be economized by partially deoxidizing the bath by employing pig iron, say, 2 to 8 per cent of the charge. The degasifying and deoxidizing agents must, of course, not be such as to lessen the above-mentioned purity of the ultimate product. In tapping the metal the slag is floated off and the product is thus freed from it. We comment upon this very interesting process on our editorial pages. (940,784 and 940,785, Nov. 23, 1909.)

Annealing and Hardening Furnace.—Mr. W. S. Rockwell patents a furnace for annealing and hardening small pieces of steel, copper, brass, etc. Since the furnace has already been described in this journal, it should suffice to say that it is an internally fired cylindrical furnace, with a spiral lining, the furnace being rotated at a certain speed and charged and discharged automatically, a pyrometer indicating the actual temperature at the point of discharge. The only manual part of the operation is that of firing, but as the fuel is constant, and the temperature is indicated by the pyrometer, it is only necessary to get the speed of the furnace and the temperature correct before starting, after which the entire operation is automatic. The patent is assigned to the W. S. Rockwell Company (940,280, Nov. 16, 1909.)

Mineral Wool.—In making mineral wool the molten slag or scoria as it comes from a cupola is blown by an air or steam blast into a blow tube and from there into a settling chamber. The commingling of the smoke and the hot filamentous slag acts to elongate and toughen the fibers of the mineral wool and completely convert the slag into filaments. In order to render the mineral wool waterproof and specially suitable for insulating boards and blocks, Mr. T. B. Parkinson introduces a heavy crude hydrocarbon oil (crude petroleum) in small streams or drops into the hot slag, as it comes from the cupola and before it enters the blow tube. The oil vaporizes, "the residue or smoke of the vaporized hydrocarbon being thus brought into intimate contact with the exterior surfaces of the flying filaments of molten slag and thoroughly coating the same." (945,583, Jan. 4, 1910. Assigned to Union Fiber Company.)

Roasting Furnace.—A novel construction of the rabbling mechanism for reverberatory roasting furnaces is the subject of a patent of Mr. Benjamin Hall. If the rabbling mechan-

ism travels from one end of the furnace to the other it often carries with it adhering unroasted ore. To avoid this, the present inventor makes the rabbles swing as a pendulum across the furnace. In order to push the material forward, the rabble shoes are tapered; the thicker end is toward the feed end of the furnace, so that the rabble shoes present a surface gently sliding down toward the discharge end, thus feeding the material along. (945,522, Jan. 4, 1910.)

Hot Blast Stove.—Since Dr. WALTER O. AMSLER'S new hot blast stove of the three-pass central-combustion-chamber type was described and illustrated in our December issue, 1909 (Vol. VII, p. 542) reference to the patent covering this construction must suffice here. (945,360, Jan. 4, 1910.)

Copper.

Smelting Sulphur Ores.—Mr. RICHARD FLEMING, of the General Electric Company, proposes to smelt sulphur ores, like copper ores of the pyritic variety, in the electric furnace as follows. The process is also suitable for smelting galena. It is carried out in a totally enclosed electric furnace. When the copper pyrite ore, which usually contains also iron pyrite and to which some silicious material is added as flux, is in a fused state, steam and air are blown up through it. The steam decomposes the melted sulphides and drives out the sulphur at the same time oxidizing the iron to form iron oxide, which combines with the silica to form silicates of iron. The copper settles at the bottom of the furnace as matte. The sulphur is given off as hydrogen sulphide, but at the high temperature of the electric furnace this is broken up into sulphur and hydrogen. If air is blown through the ore together with the steam, a certain amount of sulphur dioxide is formed, which yields the following two reactions:

$$SO_2 + 2H_2 = S + 2H_2O$$
$$SO_2 + 2H_2S = 3S + 2H_2O.$$

Hence the vapor is essentially a mixture of sulphur vapor and steam, together with the vapors of arsenic or zinc, etc., if these are associated with the copper. The vapor is condensed in a separate condenser. When a sufficient amount of copper matte has accumulated in the furnace it may either be run off to another furnace to be finally purified, or it may be separately blown with either steam, or with steam and air, in the same furnace. It is finally drawn off as metallic copper containing a small amount of impurities. (945,926, Jan. 11, 1910.)

Zinc.

Electric Zinc Smelting and Zinc Condenser.—In our Vol. VII, page 468, a full illustrated description was given of the electric zinc process of E. E. CÔTE and P. R. PIERRON, in operation in Southern France. A recent patent of the two inventors refers especially to the zinc condenser. This is a vertical cylinder with a lining of refractory earth and of greater diameter at its upper end than at its lower end. This condenser is kept at the proper temperature for zinc condensation by a vertical heating rod, in the center of the condenser. It is heated by an auxiliary electric current. It is a thin-walled tube of finely ground, strongly pressed refractory earth, filled with fragments of powder of retort carbon. The temperature in the condenser is adjusted by varying the length of this column of carbon, or by varying the size and state of comminution of these particles of carbon or by adjusting the current which passes through this column. There is only a comparatively narrow space between this vertical central core and the slightly inclined inside walls of the condenser and within this space the zinc vapor (which enters at the top) is condensed and the liquid zinc is drawn off from the inclined bottom. (944,774, Dec. 28, 1909.)

Zinc Oxide.—In a patent for a method of preparing zinc oxide for reduction Mr. H. PAPE emphasizes the disadvantages of the very loose condition of ordinary zinc oxide for reduction. He proposes to treat the zinc oxide in a muffle or reverberatory furnace, in which it is exposed for some considerable time, say, two to three hours, to great heat, which may advantageously exceed 1000° C. By this treatment the loose zinc oxide is condensed or solidified in such a manner that it coagulates into solid lumps and pieces, and this solidification can be carried to such an extent that the weight of a given volume of zinc oxide which has been thus treated and then reduced to pieces 3 mm in size is about five times as great as the weight of an equal volume of crude oxide, which is removed from the settling chambers or dust flues. When the zinc oxide has been coagulated and solidified by heat in the manner described it is withdrawn from the muffle or reverberatory furnace and allowed to cool. It can then be transported in the same way as roasted blende and after the larger lumps have been broken up it can be used in the same manner as blende for forming the mixture for reduction to zinc. The instability of zinc oxide, which is prejudicial in the reduction process, is entirely obviated by this means, as is also the coagulation of the oxide in the muffle during the reduction process. (947,399, Jan. 25, 1910.)

Zinc-Calcium.—The production of zinc-calcium and other similar alloys by melting together the two materials involves considerable loss. Messrs. G. O. SEWARD and F. VON KÜGELGEN propose to produce such alloys by the electrolysis of a mixture of salts of the two metals. For instance, for the production of zinc-calcium a mixture of calcium chloride and zinc chloride is used. However, the tendency of the current is to separate first only the zinc. To prevent this the zinc chloride is employed in a relatively small quantity. An electrolyte of fused calcium chloride is used, with a carbon or graphite anode, and an iron cathode of such dimensions relatively to the current used that the calcium is obtained in a molten condition. A current density of about 60 amp per square inch is suitable. Upon commencing the electrolysis a small proportion (from 1 to 5 per cent) of zinc chloride is introduced, and from time to time so much zinc chloride is added as is necessary to keep the concentration of zinc in the electrolyte constant. Calcium chloride is also fed to replace that decomposed, but the electrolyte is always kept so that the proportion between the zinc and the calcium salt remains constant or approximately so. The proportion of the zinc in the resultant alloy may be varied by adjusting the concentration of the zinc in the electrolyte. In similar manner may be produced alloys of calcium and aluminjum, calcium and magnesium, barium and aluminium, barium and magnesium, etc. These alkali earth metal alloys are much more easily obtained than the alkali earth metals alone, for the reason that their melting points are lower and that there is not so much tendency for redissolution as with the alkali earth metals alone. (944,826, Dec. 28, 1909.)

Resistance Alloy—Mr. WILBUR B. DRIVER patents a resistance alloy of nickel and manganese in which the nickel predominates. Pure nickel has a comparatively low resistance, only about 12 microhms per cubic centimeter. A mixture of 95 per cent of nickel with 5 per cent of manganese gives a resistance of about 25 microhms per cubic centimeter, or twice that of pure nickel. Eighty per cent nickel with 20 per cent manganese gives a resistance of about 70 microhms; while 70 per cent nickel and 30 per cent manganese has a resistance of about 100 microhms. The melting point of these alloys is estimated to be in the neighborhood of 2500° Fahr. These alloys all have the advantage of not corroding readily on exposure to the air. They have the further advantage of electric stability, differing in this respect from alloys of copper and manganese alone. (943,066, Dec. 14, 1909.)

Calcium Carbide.

A number of patents have just been granted to Mr. HERMAN L. HARTENSTEIN, whose activity in the calcium carbide field has been repeatedly noticed in this journal.

Method of Manufacture.—The object of one of Mr. Hartenstein's patents is the use of anthracite coal screenings in the manufacture of carbide, while avoiding the disadvantages of excessive smoke generation in the furnace during reduction or the higher voltage required on account of the presence of particles of hard coal in the charge. The lime is taken from the calcining furnace while highly heated and is mixed with a

mixture of 80 or 90 per cent finely comminuted anthracite coal screening and 20 or 10 per cent of powdered coke (breeze) or charcoal. The result may be still further improved and facilitated by the addition to the carbonaceous constituent of from 3 to 20 per cent of bituminous coal rich in tarry compounds, say, about 5 per cent. It is this addition of coke and bituminous coal to the anthracite screenings which overcomes the above-mentioned difficulties. (946,511, Jan. 11, 1910.)

Attaining Uniform Composition.—To secure more uniform composition and quality, Mr. Hartenstein proposes to empty various charges of carbide as they are drawn from the furnaces into a large storage reservoir, allowing them to stand for some time before solidification in final form. During this storage period the different charges become thoroughly mixed merely by the dumping of each new charge, the heat being kept up by the constant accession of carbide direct from the furnaces at high temperature. This not only secures greater uniformity of composition, but allows for the perfect completion of all reactions. When a sufficient quantity of carbide has been collected part is drawn off. A certain amount, however, always remains in the reservoir The advantages claimed for the process over the mechanical admixture of the finished product is that the properties of all the granules are the same, thus obviating the possibility of troublesome secondary generations of gas. (946,510, Jan. 11, 1910.)

Carbide in Globular Form.—For use in acetylene generators of the "carbide feed" type, Mr. Hartenstein makes a carbide in globular form, by subdividing the solid carbide while still molten or plastic. This may be done, for instance, by a method analogous to that of making drop shot, the molten carbide being allowed to fall down a shaft under such thermal conditions that it preserves its fluidity sufficiently long to form spherical or rounded drops prior to solidification. Another method is to deposit molten carbide on a revolving table, from which it is thrown outward by centrifugal force in the form of globules. The globules are allowed to harden completely before packing. The main advantages of turning out the product in this form is that grinding is unnecessary and the fine dust, so troublesome on account of its excessive surface exposure to the atmosphere and its irregular generating qualities, is avoided. (946,433, Jan. 11, 1910.)

Carbide in Aggregated Mass.—In a process very similar to the above the globules are assembled after they have solidified enough to retain their shape, but before they have lost their surface viscosity. The particles thus become autogenously united to form an open-textured honeycombed mass. (946,497, Jan. 11, 1910.)

Utilization of Carbide Fines.—By incorporating finely powdered carbide as it comes from the crushing and sizing apparatus into the molten carbide coming from the furnace, Mr Hartenstein produces a magma which solidifies without the usual piping and segregation, giving a fine grained homogeneous carbide block or body of uniform quality throughout and yielding but a relatively small proportion of fines on comminution (946,432, Jan. 11, 1910.)

Electrode Holders.—Finally two patents of Mr. Hartenstein refer to mechanical details of carbon holders for use with electrodes in large furnaces. (946,434; 946,435, Jan. 11, 1910.)

Electrolysis of Common Salt.

Mercury-Cathode Diaphragm Cell.—The cell of Mr. J. J. Rink, of Copenhagen, Denmark, for the electrolytic production of caustic soda and chlorine from common salt is essentially a mercury cathode cell, but the principal feature is that the mercury cathode is vertical. It consists of a vertical conduit plate, the pure mercury being supplied through a pipe on the top, and flowing downward on both sides of this conduit plate in a thin film. Anodes are placed on both sides opposite to this conduit plate, so that while the mercury film passes downwards, it is changed into a sodium or potassium amalgam. To prevent the chlorine, which is set free at the anodes, from coming into contact with the mercury or amalgam, diaphragms are inserted between anodes and cathode. The amalgam flows off at the bottom into a washing chamber, where the caustic is formed, the pure mercury being pumped up again to the top of the cell. (947,741, Jan. 25, 1910.)

Chlorate.—For the production of caustic soda and chlorine (bleaching powder) from common salt, it is all important to keep the anodic and cathodic products of electrolysis separate from each other, so much so that in the patent of the preceding abstract two means to accomplish this were provided; a diaphragm and the mercury cathode which absorbs the sodium ions and removes them from the field of electrolytic action. Just the reverse is true, if a common salt solution is electrolyzed to produce chlorate, as the chlorate formation depends on the interaction between the anodic and cathodic products of electrolysis. Messrs. Harry H. Bates and Folger Adams, of Joliet, Ill., feed the salt solution to be electrolyzed into a central compartment which is entirely separated from the anodic and cathodic portion of the cell. The central compartment is entirely jacketed by the hot solution in contact with the electrodes so that the liquid in this central compartment becomes heated. From this central compartment the hot sodium chloride solution is let to the outer portion of the cell which contains the movable electrodes. By this motion the electrolyte is kept in violent agitation and the chloride is changed into chlorate. From this compartment the liquid is drawn off into the ordinary cooling or settling chambers. (944,650, Dec. 28, 1909.)

Electric Discharges Through Gases.

Hydrocyanic Acid.—The classical examples of producing gas reactions by means of electric discharges are the production of ozone and nitric acid from air. Ozone is produced by the silent discharge through air; the oxygen is the only constituent of the air which is thereby affected; about the mechanism of the reaction we know little; it has even been suggested that the reaction is simply due to the ultra-violet light which accompanies the silent discharge. The production of nitric acid from air by the electric arc, on the other hand, seems to be, to a large extent at least, a thermal phenomenon; that is, the nitric acid would be formed from air if the latter was heated to arc temperature by other means. An interesting example of the possible extension of such methods is the production of hydrocyanic acid from nitrogen and hydrogen. It is known that by passing a mixture of nitrogen and hydrogen through an electric arc produced between two carbon electrodes, combination of the gases with the carbon volatilized in the arc to form hydrocyanic acid takes place. In practice when it is attempted to carry on this process on a large scale, it is found that in order to obtain any large quantity of hydrocyanic acid it is necessary to pass the gases at a comparatively slow rate through the arc, since if they are passed rapidly through it, they do not become sufficiently highly heated. Professor O. Dieffenbach and Mr W. Moldenhauer, of Darmstadt, Germany, overcome this difficulty in an interestingly simple manner. The direct current arc is produced in a vertical furnace between a central carbon electrode at the top as cathode and a deep bed of unformed lumps of coke filling the lower part of the furnace as anode. In this way it is possible to produce a conical shape of arc playing between the carbon rod (which is only slowly consumed), and the lumps of coke below The mixture of nitrogen and hydrogen gases is led upwards from the bottom of the furnace through the incandescent bed of coke and is thus preheated, before it gets into the zone of the arc proper. (941,768, Nov. 30, 1909.)

Generation and Storage of Ozone.—When the storage of ozone is accomplished by compression after generation the heat generated by the compression results in dissociating part of the formed ozone back into oxygen. To prevent this, Mr. Frank Ashley, of New York City, compresses the untreated air or oxygen, withdraws the heat generated thereby, and then allows the pressure to quickly drop a few pounds in the ozonizing

chamber, thereby expanding the oxygen and cooling it, and then subjecting it to the action of the silent electric discharge while under pressure. The ozone is then stored in a suitable storage reservoir without permitting it to expand much below the pressure under which it is generated. (942,046, Dec. 7, 1909.)

Ferro-Alloys.

Electric Furnace Construction.—Mr. E. F. PRICE of the Union Carbide Company and the Electro Metallurgical Company has formally proposed an electric furnace for the production of ferro-alloys, in which a considerable body of the charge, for instance a mixture of silica, iron ore and carbon for the production of ferrosilicon, is maintained around depending carbons of opposite polarity, thereby embedding the zone of reduction, retaining the heat within the furnace, increasing the production of metal, keeping the slag in a liquid condition, and protecting the electrodes from the oxidizing and cooling effect of the atmosphere. While the use of this deep charge-body is advantageous, it is found necessary in practice to employ a charge of coarsely granular materials, in order to permit the ready escape of the large volumes of gas which are evolved during reduction. A finely divided charge, however, could be preferable in many respects, enabling reduction to proceed more rapidly and smoothly, but when a deep body of finely divided material is maintained around the electrodes, the gases escape violently by blowing out portions of the hot unreduced charge and agitating the molten or semi-molten contents so as to short-circuit the electrodes. To prevent this disadvantage Mr. Price withdraws the waste gases laterally in different directions beneath the surface of the finely-divided charge. For this purpose an inclined shelf is employed inside the furnace and surrounding the electrodes above their lower ends. It is made of cast steel and is water-cooled. The charge drops from above, rests on this shelf and surrounds the electrodes, and passes downwards in the space between the rims of the shelf and the electrodes. In this way part of the space just below the shelf is kept free of the charge. Through this space, the waste gases are withdrawn. (942,349, Dec. 7, 1909.)

Furnace Construction.—Mr. Price also patents another method by which the gases and dust arising from the operation are continuously withdrawn without interference with the operation of charging the furnace or with the accessibility of the electrodes for adjustment or renewal. In this construction the furnace consists of an open vessel; opposite series of vertically disposed electrodes depend into the same and a stack is disposed upon the bath and between the electrodes whereby the air-current flowing to the stack passes around and between the electrodes. (942,350, Dec. 7, 1909.)

Ferro-Tungsten Furnace.—In electric furnaces, for the production of high-grade ferro tungsten, containing over 70 per cent, usually but a limited quantity of ore is treated in a single furnace-run; after reduction it is cooled in the furnace and subsequently removed by chiseling it out. Mr. CHARLES E. WILSON, of Hood River, Ore., proposes to use a furnace chamber in the form of a crucible with removable sides of refractory material and provided with a graphite bottom which forms one of the electrodes. Extending downward through the open top of the furnace is the other electrode in the form of a graphite pencil. The ore is fed into the furnace around the pencil. The slag making constituents of the ore form the heat generating material. As the reduction progresses the ferro tungsten agglomerates between the electrodes and settles upon the carbon-crucible, and as a core of the alloy builds up the crucible is lowered so as to gradually build up a ferro tungsten core enveloped in slag. When it has attained sufficient height the current is shut off, the contents of the crucible are allowed to cool, the removable walls of the furnace are taken off and the slag is easily broken off the ferro-tungsten core. (943,290. Dec. 14, 1909.)

Powdered Metallic Tungsten.—A process for the production of powdered metallic tungsten has been patented by Mr. WILHELM MAJERT, of Berlin, Germany. Finely powdered anhydrous sodium paratungstate is incorporated with lamp black in the proportion by weight of 500 to 35. This mixture is then agitated and heated in a reducing atmosphere. Metallic tungsten and sodium tungstate are formed and the latter is washed out, first by the use of slightly alkaline water, then of water aciduated with HCl or H_2SO_4, and then with pure water, until a neutral reaction occurs. The metallic tungsten residue is then washed with methyl or ethyl alcohol and then with benzol or with the lightest benzine, portions of which are allowed to remain incorporated with the residue, producing a plastic mass which may be left in tightly closed receptacles until it is desirable to utilize it. This prevents oxidation of the powder. The process is said to be specially useful for the manufacture of tungsten filaments for incandescent lamps. The same process is applicable with any hexavalent high-fusing-point metal. Suitable salts for the process are sodium paratungstate (Na_2WO_4) and sodium paramolybdate ($Na_2Mo_2O_7$) from which one molecule of tungsten or molybdenum may be reduced, leaving in the residue the neutral salts of sodium, tungstate (Na_2WO_4) or sodium molybdate (Na_2MoO_4). (946,551, Jan. 18, 1910.)

Miscellaneous.

Cleaning Process.—A very simple process for cleaning metal articles, especially silver, gold, etc., is as follows: The silver articles to be cleaned are placed in a tin-lined vessel (or an aluminium dish), filled with a hot alkaline solution. On account of the different position of silver and tin in the potential series, this represents a short-circuited galvanic couple, current flowing from the tin into the solution and then to the silver and back to the tin. The oxide film on the silver is thereby reduced to metallic silver by cathodic reduction, and the silver is cleaned. It is an electrolytic process without the use of any current from an external source. But the action soon ceases, because the tin becomes coated with a film which is difficult to remove, and which prevents effective electrical contact with the silver article. To provide such contact Messrs. James D. Phillips and Carl Hambuechen provide for the introduction of an auxiliary metal electronegative to the anode. Thus, if the anode is zinc, and it is maintained in electrical contact with the metal electronegative to it such as tin, silver, or solder, the latter metal will remain bright and may thus be used to secure an interrupted contact with the article being cleaned. For instance, to clean silverware, a vessel of tin or enameled ware may be used with a corrugated sheet of zinc lying loosely in the bottom. The upper portion of the ridges or corrugations are tinned or soldered, this tin or solder constituting the auxiliary metal and remaining bright, while making contact with the silverware placed in the vessel. The tarnished silver thus gets automatically cleaned. A suitable solution is prepared by dissolving one tablespoonful of baking soda and one-half tablespoonful of common salt in one quart of water. (945,865 and 945,866, Jan. 11, 1910.)

Carbon-Silicon Dynamo Brushes.—Carbons, when impregnated with a certain amount of silicon, are very hard and excellent means for polishing a commutator. The higher the silicon content, the greater the hardness and the less the conductivity. A patent of G. Egly (assigned to Gebr. Siemens) recommends to combine the carbon and silicon with nitrogen. The molded mixture of carbon and silicon is heated in a nitrogen atmosphere in order to produce a compound of nitrogen silicon and carbon. In this manner a very uniform binding of the entire mass and great homogeneity of the body is obtained. The kind of compound which is formed by nitrogen combining with silicon may be very different; when a sufficient quantity of silicon is employed and when the mixture is sufficiently heated apparently the chemical compound C_2Si_2N is formed. (942,379, Dec. 7, 1909.)

A Comparison of Welding Methods for the Repair of Castings.

In *Stahl und Eisen*, of Dec. 29, 1909, Director E. Lamberton, of Merzweiler (Alsace, Germany), gives an interesting comparison of different welding methods for the repair of gray iron castings in foundries, repair shops, etc. As the article is based on experience from German practice, we give it here in translation almost in full, but have added some notes on American practice. The methods discussed below may, of course, also be employed for welding steel, but cast-iron repairs are chiefly considered in the following article.

In every works it becomes necessary at times to repair broken castings. In foundries it is also occasionally necessary to attend to certain surface defects, flaws, etc., of castings.

All the welding processes to be described depend on simply fusing the parts to be welded together, without introducing any other material into the joint. While the different processes are more or less simple, all require certain precautions.

In the first place, any welding operation produces internal stresses or tensions within the portions which are being welded on account of the great internal temperature differences. With an unelastic material like cast iron this is a serious matter, since there cannot be any equalization of the stresses between the hottest parts and the colder adjoining parts, which often results in a new break. This can be remedied to a certain extent by careful preheating, and cooling as described below.

While no detailed rules can be formulated comprising all cases in practice concerning the method of preheating the pieces to be welded, yet one general principle should be observed. The conditions under which the welded piece is cooled after welding should be the exact reverse of the conditions under which it is preheated before welding and should be identical with the conditions under which it was cooled after the original casting operation was completed. It is, therefore, important to preheat the pieces (before welding) slowly and continuously up to the maximum temperature which can be reached by the available means. The pieces to be welded together should be held in a firmly fixed position during the welding operation, and preferably in such a way that the joint to be made is in a horizontal plane.

Another important point which needs attention is to keep the two surfaces to be welded free from rust and any impurity. If there is rust on one of the surfaces, then the iron oxide is decomposed by the fused iron, the oxygen of the iron oxide combining with the carbon in the fused iron; gas formation results and the weld is porous.

If the repair relates to the removal of blowholes or flaws, the unsound portion of the casting must be completely cut out. This is illustrated in Fig. 1, where *A* is the casting, showing the unsound portion; *B* shows to what depth it is necessary to drill a cavity, which is then to be filled up with fused iron. It is also advisable to make the surfaces of the cavity not smooth, but rather rough or corrugated to make it easier for the fused iron which is added, to stick to and unite with the surface.

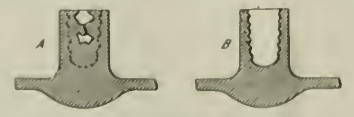


Fig. 1.—Removal of blowholes or flaws.

The above principles hold good in general whatever the special method of welding. They should be followed carefully in any case.

Mr. Lamberton's comparison refers to the following methods: The old welding process by simply adding fused iron (so-called burning on); electric welding; thermit welding; "autogenous" welding (oxy-acetylene or oxy-hydrogen welding). (It is true that in speaking of "autogenous welding" the oxy-acetylene or oxy-hydrogen process is now generally meant; but this is really incorrect. "Autogenous welding" means strictly joining or welding without use of a solder which would introduce a foreign material into the weld. Every one of the methods mentioned above is really an autogenous welding method.)

Burning On.—This old process consists simply in surrounding the casting to be repaired with a mold of sand. A large quantity of liquid metal is then run past the surface to be welded until it reaches fusion heat. The excess of metal runs

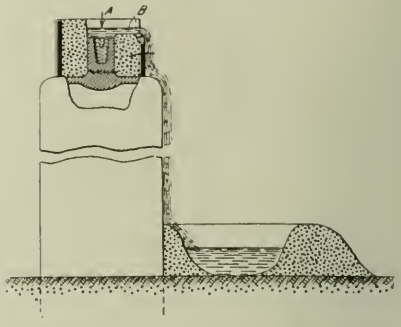


Fig. 2.—Repair of Fig. 1 by burning on.

off through an overflow channel which is preferably made of molding sand. Fig. 2 shows the repair of Fig. 1. The metal is poured at *A*. The outflow channel is at *B*.

Fig. 3 shows the repair of a machine frame, broken in operation. The arrangement of the mold is similar to the preceding case. The liquid metal is poured into *A*, and when it is seen to run off through *C* it is evident that fusion takes place at the desired depth; the channel *C* is then closed and further addition of fluid iron is stopped.

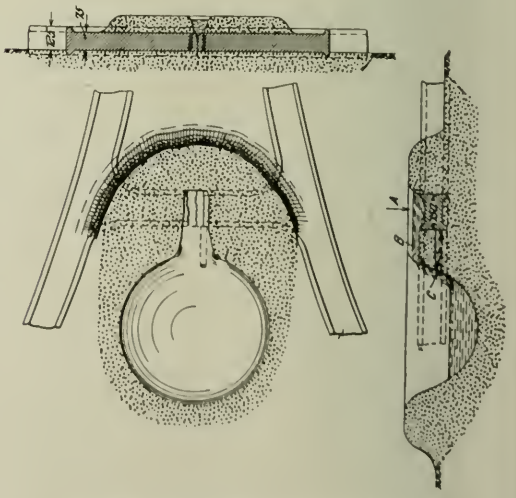


Fig. 3.—Repair of broken frame by burning on.

Fig. 4 shows the repair of a roll, to which a new boss is to be added. Some liquid iron is held for some time at *A* where the new journal piece is to be added, so that the contact surface is liquefied. The mold is completely filled with metal only after the weld has been produced in sufficient depth and after the overflow channel *B* has been closed.

With this process of welding by simple fusion it is important to arrange everything in such a way that the workingman who makes the repair can do it conveniently without being troubled by the heat of the iron running off through the overflow channel. The walls are covered with dry molding sand.

The molten iron which is added should have as high a temperature and fluidity as possible. The iron which is being added should come successively into contact with all points of the surface to be welded. This may be easily done with a

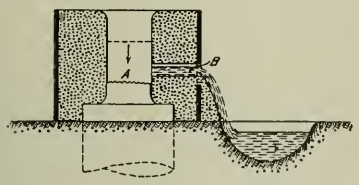


FIG. 4.—REPAIR OF ROLL BY BURNING ON.

hand ladle; but when large quantities of metal are needed and a crane must be used, it is difficult. It is necessary to make sure that fusion has been obtained at the desired depth; this may be done by inserting a perfectly clean and preheated iron rod.

After completion of the weld, the welded portion is covered with a layer of powdered charcoal and the whole is covered with dry sand to make sure that the cooling is slow.

The weld depends entirely on sufficient metal being run over the welding surface to bring it to fusion point. Then the flow is stopped and the metal retained in the mold will fuse with the casting. It is difficult to give general rules as to the quantity of liquid iron required; even to remove small surface defects 5 kg to 8 kg (11 lb. to 18 lb.) of metal are required to make success sure. For small repairs, like that of Fig. 3, 1 kg or 2 kg may be assumed per square centimeter (14 lb. to 28 lb. per square inch); for welding small axles or rolls 2 kg or 3 kg per square centimeter (28 lb. to 43 lb. per square inch). For large repairs these figures may be reduced one-half.

The above process can be used in any foundry, and is often employed with remarkable skill; but it must be remembered that

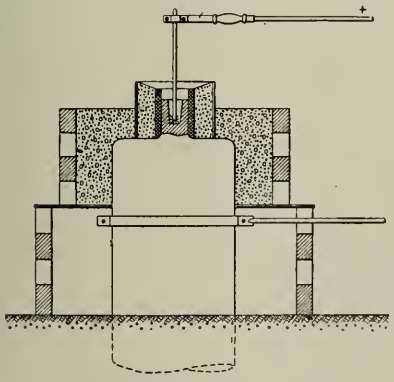


FIG. 5.—REPAIR OF FIG. 1 BY ELECTRIC WELDING.

the quantities of molten iron required for large repairs are considerable and that the handling of such large quantities is difficult and dangerous. (The old burning-on method is specially applicable for cast-iron repairs; for steel welding it offers greater difficulties.) The burning-on method is further restricted, however, to places where the necessary appliances are available, like foundries and machine works. Since this condition is often not fulfilled where a repair is to be made, the electric, thermit, oxy-acetylene, and oxy-hydrogen welding processes have been developed.

Electric Welding.—The chief disadvantage of electric welding is that it requires an electricity supply. Mr. Lamberton describes the Slavianoff process of electric welding, which is used in the works of Julius Pintsch, in Fuerstenwalde-on-the-Spree, in Germany. It is used there especially for the repair of broken locomotive cylinders, and is stated to give quite satisfactory results.

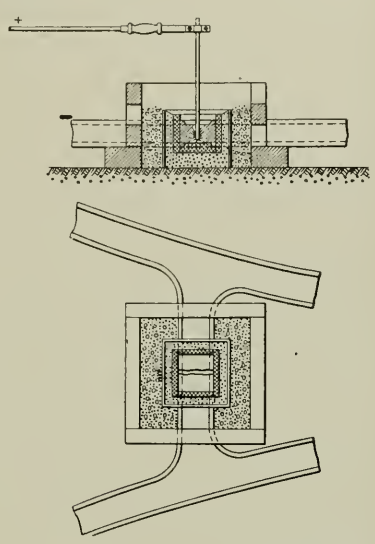


FIG. 6.—REPAIR OF BROKEN FRAME BY ELECTRIC WELDING.

An electric arc is formed between the piece to be welded and a rod of cast iron or steel (of the same composition as the piece to be welded). The method is clearly evident from Figs. 5, 6 and 7.

Fig. 5 corresponds to Fig. 3, showing the movable iron or steel rod which is connected to the positive pole of the circuit. It is 8 mm to 10 mm thick (5/16 in. to 13/32 in.). The piece to be

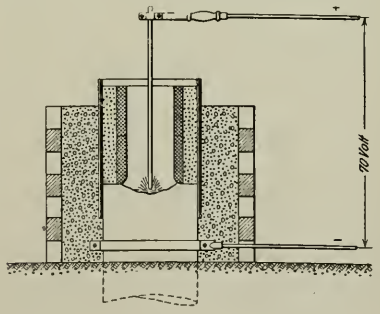


FIG. 7.—REPAIR OF ROLL BY ELECTRIC WELDING.

repaired is enclosed by pressed coke plates, surrounded by the molding sand in a sheet-iron box. The box is surrounded by charcoal or coke within walls of firebricks put together so as to leave open spaces. After the piece has been preheated by the charcoal or coke, the arc is produced by bringing the lower end of the iron rod in contact with the surface to be welded.

In Fig. 6, which corresponds to Fig. 4, it is necessary to enlarge the fracture so that the two surfaces to be welded are at a

distance of 20 mm to 30 mm (13/16 in. to 1 3/16 in.) from each other. The two surfaces are now heated to fusion temperature (beginning at the bottom), while fused iron drops off from the lower end of the rod *a* and unites with the metal of the two surfaces.

Since considerable heat is produced during the welding operation, it is possible to add cold iron in small particles. This is of special importance in Fig. 7 (corresponding to Fig. 5), where considerable quantities of iron are required to make the new boss. Of course, molten iron may also be added if it is available. Mr. Lamberton thinks a repair like that of Fig. 7 could not be well carried out by this process without the use of molten metal.

During the welding operation, especially when molten iron is added, it is important to keep the surface of the joint free from slag or other impurities to ensure a good union with the liquid metal. This is accomplished by repeated scraping off with an iron rod.

Direct current is used at 65 volts; the current is 400 amp to 600 amp according to the thickness of the iron rod *a*. The iron rod must be held during the welding operation so that the arc length is not more nor less than, say, 10 mm (25/64 in.). This requires skilled workmen. They hold the welding rod *a* by means of a pair of tongs, as shown in the illustration, and regulate the arc length by raising or lowering the rod.

Compared with the simple old fusion process of welding, the electric welding process gives a much stronger weld. This is probably due to the fact that no gases can form as the result of an oxidation process, so that the metal gets a very dense structure.

The above electric welding process has not been introduced into this country to our knowledge, although electric welding has been developed here to a large extent. It is used, however, in this country, not for making all kinds of repairs of castings, but rather in such cases where a manufacturer has thousands of

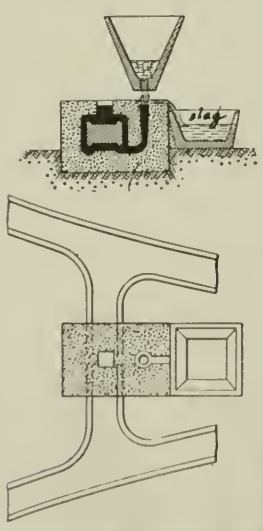


FIG. 8.—REPAIR OF BROKEN STANDARD WITH THERMIT.

pieces of the same kind to weld. The Thomson Electric Welding Company, of Lynn, Mass., has been the pioneer in this field. Their process is especially adapted for making a joint in solid sections, where an end of the piece, round or square, is abutted against the end of another piece of corresponding section, also for T-pieces, etc. Another development of their process is the welding of sheet metal surfaces together in spots, or so-called "electric riveting." The process of the Thomson Electric Welding Company is also employed by the Electric Welding Company, of Pittsburgh. Other processes of electric welding are employed by the Toledo Electric Welding Company, Toledo, Ohio, and by the Standard Welding Company, of Cleveland, Ohio. But as this article is purposely restricted to a comparison of welding methods for the repair of castings, a discussion of the above electric welding methods is reserved for a special article in a future issue.

Thermit Welding.—The thermit welding process makes

$$Fe_2O_3 + Al_2 = Al_2O_3 + 2Fe$$

setting free a large amount of heat and producing very hot molten soft steel. The only requisites are thermit (a mixture of metallic aluminium and iron oxide), and besides this a small

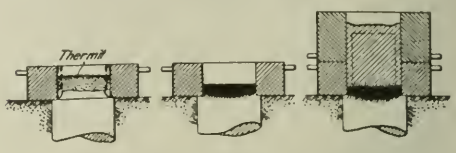


FIG. 9.—REPAIR OF ROLL WITH THERMIT.

amount of ignition powder, which, when lighted with a match, produces the required very high temperature to start the reaction. This process can, therefore, be carried out everywhere, since thermit can be easily transported, being neither explosive nor ignitible at any but very high temperatures.

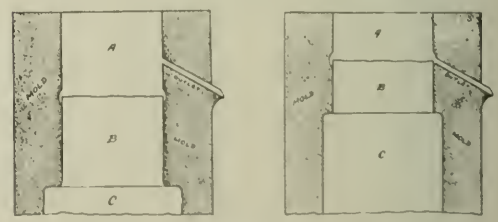
Since the thermit welding process, as carried out by the Goldschmidt Thermit Company, of New York, has been the subject of many articles in former volumes of this journal, no further description is here necessary. But it will be interesting to describe how, according to Mr. Lamberton, the typical repairs of Figs. 2, 3 and 4 are carried out with thermit.

For the removal of blowholes (Figs. 1 and 2) thermit is ignited in magnesite crucibles; the very hot molten metallic iron collects at the bottom, while the alumina slag floats on top. The slag is removed. The molten iron is carefully skimmed of any slag particles (by means of a clean iron rod to which any slag particles adhere) and is then poured into the hole of Fig. 1, *B*.

The welding of two broken machine parts is carried out as shown in Fig. 8 (corresponding to Figs. 3 and 6). The illustration is self-explanatory. To get a perfect weld, the section at the joint is enlarged so as to weld a collar around the joint, and thus strengthen it.

The welding of a boss to a roll by means of thermit according to German practice is shown in Fig. 9, which is easily understood.

It should be added that in the first two cases the joint is no



FIGS. 10 AND 11.—REPAIR OF ROLLS BY THERMIT.

longer cast iron, but thermit iron (soft steel). In special cases of cast-iron repairs this may be a disadvantage, but it is not necessarily so under ordinary conditions, so that Mr. Lamberton concludes that this particular point is not of importance.

Thermit is rarely used pure. Generally iron filings or steel scrap (about 5 per cent to 10 per cent) are added. These are thoroughly cleaned and are added to the crucible before the reaction is started.

With the thermit process it is also important to observe the same precautions of preheating and slowly cooling, described above. Large, solid pieces, which are liable to internal stresses on account of their form, must be preheated with special care. For this purpose a small transportable coke oven is often used abroad, while American practice prefers a gasoline torch operated with compressed air.

In other respects American practice does not differ from the above description of the operation of the thermit process with respect to the first two examples, namely, the removal of surface defects and the welding of two broken machine pieces. As to the third example, it may be said that the repair of broken rolls by thermit is used very extensively in this country, and the general practice is as follows:

In cases where a boss is entirely broken off the thermit process is simply used for the purpose of bringing the fractured surface of the roll to fusing heat, so that a new end may be burned on without difficulty. To accomplish this the roll is placed on the ground in a vertical position and a mold constructed of molding sand, thoroughly dried, is placed about the fractured end. This mold should be provided with an outlet or skim gate 8 in. above the fractured surface. Some steel works prefer to place this skim gate on a level with the fractured end, but it is better practice to locate it higher up. (Figs. 10 and 11.)

A quantity of thermit figured at the rate of 40 lb. to the square foot of surface to be welded is then placed in the mold and ignited. The heat of the reaction raises the temperature of the fractured surface to the fusing point and makes it possible to "burn-on" a section of metal to replace that broken off. This is accomplished by pouring in hot steel from the ladle suspended over the top of the mold as soon as the thermit reaction is completed. In the case of a cast-iron roll, molten iron is used instead of steel.

The pouring is continued until all traces of thermit steel and slag have been washed from the welding surface, when the outlet is plugged up and the mold filled to the top with metal. The waste used in washing off the surface is caught in a suitable receptacle and used to pour small castings.

After the weld has cooled it should be annealed by bringing the roll up to a temperature of 1200° Fahr. and holding it there for five or six hours, after which it is allowed to cool down slowly.

Rolls up to 44 in. in diameter have been successfully repaired in this way, and the process has also been used in welding on sections of steel to the shafts of large pinion gears. The cost of any of these operations, including thermit, labor, annealing and machine work, amounts to only a fraction of the cost of a new roll or shaft. The method of constructing the molds, as recommended by the Goldschmidt Thermit Company, of New York, is shown in Fig. 10 where *A* is the mold for boss, *B* the fractured end of the boss, and *C* the roll. A somewhat different arrangement, which is used by some American steel companies, is shown in Fig. 11.

Oxy-Acetylene and Oxy-Hydrogen Welding.—This method depends on the high temperature which is obtained in the oxy-hydrogen flame (about 4000 deg. Fahr.), and the even higher temperature obtained with the oxy-acetylene flame (about 6300 deg. Fahr.). The possibility of bringing the burner easily to the point to be welded, and, further, the fact that in repairing broken machine parts the welding operation proceeds gradually from point to point along the line or surface to be welded, are the main features of the process. The latter feature is of great importance, especially for repairs of curved surfaces, if pieces are broken out, as happens occasionally. A very interesting application of the process is the repair of broken gas-engine cylinders, which is difficult with other methods.

The repairs shown in Fig. 3 are made with the oxy-acetylene or the oxy-hydrogen flame by first chipping off pieces from both sides of the fracture so as to form a V-shaped opening. This is then heated and the V-shaped opening is filled carefully with uniform iron or steel particles. Then the welding with the oxy-acetylene or oxy-hydrogen flame begins, and care must be taken that the two surfaces of the solid pieces themselves become fluid, so that they combine firmly with the added material.

In cast-iron repairs a difficulty may be found in the affinity of cast iron for oxygen, so that the molten cast iron may get covered with an oxide film during the welding operation. Further, small globules may form at the joint and prevent the flowing together of the original surfaces with the added metal.

The easiest remedy in such cases is the use of one of the commercial fluxes, but not pure borax, since this may easily cause hardening of the joints.

To carry out such work requires a certain amount of skill and experience on the part of the operator. It is also of importance that the same material should be subjected only once to the welding process, since otherwise it may get hard and brittle. As to the material to be added for making the weld, a soft charcoal iron, low in carbon and as free from sulphur as possible, may be used.

The cost of the process is exceedingly low, but depends in such a degree on the greater or smaller skill and experience of the workman that reliable figures cannot be given. The main item of cost is doubtless the cost of preheating the piece to be welded. The other repairs described above may be made in an analogous way.

Like the thermit process, which has made rapid progress during the last five years in this country, the oxy-acetylene process has also made remarkable advances in the United States, especially during the past few years. There are quite a number of different companies interested in this process. The chief differences between the various processes occur in the types of burner employed, the types of acetylene generators used, and the method of producing oxygen..

Among the companies making oxy-acetylene apparatus in this country are the American Oxygen Company, of Philadelphia (formerly the American Ferrofix Brazing Company), the Beltzer-Delcampe Welding Company, of Bridgeport, Conn.; the Davis-Bournonville Company, of New York City; the Industrial Oxygen Company, of New York City; the Linde Air Products Company, of Buffalo, N. Y.; and the Worcester Pressed Steel Company, of Worcester, Mass.

These companies use differently designed blowpipes, the chief point of the design always being the provision of means by which both a perfect control of the flame and a thorough mixture of the two gases in easily adjustable proportions are realized. (The properly adjusted oxy-acetylene welding flame is neutral in character; *i.e.*, neither oxidizing nor reducing, except in very special circumstances with special alloy steels. The final regulation of an oxy-acetylene flame is always made by its appearance, gage adjustment being merely an approximation.) It is customary to make a distinction between low-pressure and high-pressure blowpipes, according to the pressure at which the acetylene is supplied to the burner. Both systems have certain advantages. A typical example of a high-pressure torch is that of the Davis-Bournonville Company, while low-pressure torches are those of the Industrial Oxygen Company, Linde Air Products Company and American Oxygen Company. However, a description and discussion of the different torches is reserved for a special article in a future issue.

As to the generation of acetylene, there are also quite a number of generators on the market, all based, of course, on the reaction between calcium carbide and water. A distinction may be made between water-fed and carbide-fed generators, according to whether water is fed into carbide or carbide into water.

An even greater variety exists in the methods of producing oxygen employed by the different companies.

The Davis-Bournonville Company, of New York City, use the chloride of potash process. A mixture of chloride of po-

tassium and manganese dioxide is placed in a retort and heat is applied. Thereupon the oxygen gas is given off.

The Industrial Oxygen Company, of New York City, employs an electrolytic product called "oxygenite," in the form of a gray pulverulent mass, resembling fine sand, releasing, when incinerated in a closed vessel, all the oxygen it contains, amounting to about 5¼ cu. ft. per pound, burning slowly and without flame.

The H. W. Dopp Company, of Buffalo, N. Y., use lavoisite, which is a dry and non-combustible powder made by the Castner-Electrolytic Alkali Company, of Niagara Falls. Water is added to lavoisite, thus causing an evolution of oxygen gas, 1 cu. ft. of oxygen per pound of lavoisite.

The Roessler & Hasslacher Chemical Company, of New York City, make a compound called oxone (fused sodium peroxide), which in contact with water produces very pure oxygen.

The Worcester Pressed Steel Company now get their oxygen compressed in cylinders, but formerly used the epurite process, epurite being a mixture of chloride of lime, sulphate of copper and sulphate of iron. Oxygen was produced by the contact of this material with water.

The Beltzer-Delcampe Welding Company use a compound called oxyvite.

The American Oxygen Company make their oxygen by a secret process.

While all the above methods make use of chemical reactions for the production of oxygen, an entirely different process is used by the Linde Air Products Company, which gets the oxygen gas from atmospheric air by liquefying the air and by fractional evaporation of the liquid air, thus separating the oxygen from the nitrogen.

Finally, oxygen is made by electrolysis of the water by the American Oxhydric Company, of Milwaukee. This is the only American company using the oxy-hydrogen instead of the oxy-acetylene welding process on a large scale and making oxy-hydrogen welding appliances. This concern compresses oxygen and hydrogen to tanks and supplies the gases in this form to their customers.

An oxy-hydrogen burner made by the Buffalo Dental Manufacturing Company, of Buffalo, is for use by lead burners for constructing and repairing all kinds of lead work, but this blowpipe is not suited to the brazing of iron and steel.

As noted above, a more detailed description of the different oxy-acetylene and oxy-hydrogen methods of welding is reserved for a future article.

Comparison of Different Welding Methods.—In summing up his discussion of the use of the different welding methods for the repair of gray iron castings, Mr. Lamberton reaches the following conclusions.

In foundries he pleads that the old burning-on process should not be superseded entirely by the new methods; the first consideration should always be whether the old process can be used. If a repair is unsuccessful, it would not be right to blame the burning-on process itself, since one cannot be sure that the repair would have been successful if carried out by another method. The new methods also require quite a good amount of skill and car (and we would add that the skill required for oxy-acetylene, oxy-hydrogen and electric welding is the skill of the mechanic, while the skill required for thermit welding is the skill of the foundryman).

Mr Lamberton thinks that for repairs of gray iron castings the old burning-on process has the advantage of a lower temperature, the internal stresses have a much better opportunity to equalize, while the piece to be welded is better preheated by the use of the very large quantities of molten iron.

Nevertheless, foundries should make use of the new methods where they offer advantages. For small and special repairs, Mr Lamberton thinks that the oxy-acetylene or oxy-hydrogen flame is advisable, although, partly at least, they may also be made by electric welding. Repairs of medium size he recommends to make by the old burning on method. For large repairs a combination of the old burning-on process and the thermit process is the only feasible method; the thermit is then used for the purpose of preparing and liquefying the solid surface, while the fused cast iron which is added produces the final union (being of the same composition as the piece to be welded).

Outside of foundries and repair shops, of course, the old burning-on method cannot be used, because the appliances are not available. For emergency repairs on docks, on board ship, in mines, etc., one of the newer methods must be employed. Mr. Lamberton then recommends the making of small repairs by oxy-acetylene or oxy-hydrogen welding, and the making of medium and large repairs by the thermit process.

Mr. Lamberton finally remarks that the equipment for oxy-acetylene or oxy-hydrogen welding is cheap enough while the thermit process requires no equipment at all besides the crucible and thermit. The electric process requires electric power, and cannot, therefore, be used everywhere. But the process yields an especially strong weld, and managers of those plants which have electric current available should acquaint themselves with this method. In conclusion, Mr. Lamberton says that the cost of the heat is approximately the same in the different processes, so that the choice of one of them should be based on the conditions of each case.

Wisconsin Research Fellowship.—Two engineering fellowships, each worth $400 a year, have been established by the regents of the University of Wisconsin. These are in addition to three graduate scholarships already offered in the College of Engineering, and will give still further impetus to the research work at that institution. This year thirty graduate students are doing advanced work in the research laboratories in the testing of materials, particularly of reinforced concrete; hydraulic problems; electrochemistry and chemical engineering. Some of the results of these researches have been published in seven bulletins of the university engineering series.

Refractories in Furnace Construction.

In the excellent little treatise recently issued by the Harbison-Walker Refractories Company, of Pittsburgh, Pa., on the open-hearth furnace and process, a concise summary is given on the principal refractories used, especially in **open-hearth bottoms,** both in the acid and the basic process. The summary, however, is of much broader interest, and in view of the prominent position of the Harbison-Walker Refractories Company in the manufacture of refractories for general and special purposes, these notes are here reproduced in full.

Acid Materials.

The only acid material used is **silica** sand, all acid furnace bottoms being made from it. A natural sand is often used containing various percentages of silica, a typical analysis being, for instance:

Water	0.24	per cent.
Silica	97.25	" "
Alumina and iron oxide	0.16	" "
Lime	0.08	" "
Magnesia	0.39	" "
Alkalies	0.36	" "
Loss on ignition	0.36	" "

Certain beach sands give very good results, and there are also a number of deposits in the West and Middle West, but it is impossible to predict by analysis alone the quality of a given sand. If of excessive purity there is difficulty in properly setting it in the furnace, and if the impurities run too high, although the sand will then easily frit or set, it is subject to great erosion in the operation of the furnace and will not make a sufficiently hard, dense bottom.

The physical characteristics have a wide bearing on the question of its adaptability, and two sands of practically identical analysis may considerably differ as to their suitability for bot-

tom work. Often a combination of two or more sands will secure the best results.

Neutral Materials.

Carbon, while almost perfectly resisting the action of the slag, is very rapidly destroyed on account of the great affinity of the metal for it. It has often been tried in the form of brick, as well as rammed with tar, and, although the matter is again and again reviewed by those not familiar with previous experiments, its practical use is out of the question for open-hearth bottoms. (As to the use of carbon, carborundum and silico-carbides as refractories for electric furnace construction, see, for instance, the article by F. A. J. FitzGerald, our Vol. II, page 441.)

Bauxite, a hydrous oxide of alumina, analyzing approximately:

Silica, SiO_2	4 to 7 per cent.
Iron peroxide, Fe_2O_3	3 to 5 " "
Alumina, Al_2O_3	60 " "
Water, H_2O	30 " "

although exceedingly refractory, is subject to excessive shrinkage at furnace temperatures. It is very difficult to so calcine as to entirely eliminate this shrinkage, and when so calcined all plasticity is lost through loss of its combined water. In modern open-hearth practice it is never used.

Chromite, a sesqui-oxide of chromium, usually termed chrome ore, which has been rather more thoroughly tried out abroad than in this country, is extremely infusible and practically neutral, but its infusibility renders it very difficult to set or sinter thoroughly, and is has been found practically impossible to so thoroughly set the bottom that it will not be subject to mechanical erosion rather than any chemical reaction. It will analyze as follows:

Sesqui-oxide of chromium, Cr_2O_3	38 to 40 per cent.
Alumina, Al_2O_3	24.5 " "
Iron peroxide, Fe_2O_3	17.5 " "
Silica, SiO_2	3.25 " "
Magnesia, MgO	15 " "

In spite of its high refractoriness it has, when selected with due regard to its analysis, and when properly prepared, a large and rapidly widening field in daubing and patching around the slag line, back walls, ports, jambs, etc.

Of course, a neutral bottom would be the ideal one for open-hearth practice, inasmuch as any change from the basic to acid process, or *vice versa,* could be made without change in the furnace; but as indicated above, the materials at our disposition do not allow a satisfactory neutral working bottom.

Basic Materials.

Lime, or calcium carbonate, $CaCo_3$, is, in its calcined form, CaO, theoretically an excellent material for bottoms, but it is practically out of the question due to the rapidity with which it slakes when exposed to air. When reheated, the rapid driving off of the gas and water reduces the lime to a powder, in which condition it is rapidly worn away.

Dolomite, a magnesian limestone, is one of the materials very commonly used for patching bottom, and in some cases even for the entire bottom particularly by those to whom its comparatively low first cost appeals, but it is open to the same objection as lime, only in a lesser degree, *i.e.,* slaking on exposure to the atmosphere, and if a stock is more than 10 days or two weeks old this becomes an important factor. Of course, when the plant calcines its own dolomite this is a less important consideration. Furthermore, it is impossible to so set a dolomite bottom that it shall be as dense and vitreous as magnesite, and in spite of most watchful care portions of the bottom will from time to time become detached and float. In a dolomite bottom where, through erosion or absorption of metal, a hole or depression is once started, disintegration progresses with much more rapidity than with magnesite, and there is considerably more liability of serious breakouts. A fairly representative analysis of dolomite is as follows:

Silica	5.52 per cent.
Alumina and iron	3.74 " "
Calcium	29.20 " "
Magnesia	17.31 " "
Loss on ignition	43.82 " "

In a number of instances the absorption of the impurities from the dolomite has proved an exceedingly serious matter in the manufacture of high-grade steels and has resulted in an increased use of magnesite. Dolomite sets somewhat more rapidly than magnesite.

Magnesium carbonate ($MgCo_3$), when calcined to the oxide MgO, commercially known as magnesite, is unquestionably the best material for open-hearth bottoms, and all original bottoms are now installed with it. Even by those using a large percentage of dolomite in repairing it is admitted to be the best material available, absolutely the only argument in favor of dolomite being its low first cost. The well-known Austrian magnesite of the following approximate analysis

Silica, SiO_2	2.84 per cent.
Alumina, Al_2O_3	0.929 " "
Iron peroxide, Fe_2O_3	8.571 " "
Lime, CaO	1.120 " "
Magnesia, MgO	85.32 " "
Carbon dioxide, CO_2	0.50 " "

gives such a proportion of iron, etc., as makes the material set in the furnace bottom, forming a hearth of maximum density, subject to minimum erosion by the metal and practically unaffected by the slags. Furthermore, on indefinite exposure to the atmosphere almost no deterioration takes place, while with a dolomite bottom under like conditions, should furnace shutdown occur, the bottom would of necessity be renewed. The large deposits of Austrian magnesite insure ample supply for years to come.

Grecian magnesite has been used as a bottom material, but on account of its high refractoriness there is difficulty in fusing it in place, it being necessary to introduce impurities in the way of silica, clay, or oxide or iron in amounts sufficient to reduce its refractoriness. The resultant is too variable a quantity to give reliable results, and Grecian magnesite is very seldom employed.

A Novel Type of Dryer.

In view of the interest shown in the articles on drying in recent issues of this paper, and in view of the great importance of this class of apparatus in many metallurgical and chemical processes, the following brief notes on a recently perfected dryer which operates on a somewhat novel priciple should prove interesting.

The fundamental idea of this dryer, built by the Atlas Dryer Company, of Cleveland, is that of bringing into direct contact with the material to be dried a greater quantity of heated air than is possible with the usual form of cylindrical rotary dryer.

The manner in which this is done is clearly shown in Figs. 1 and 2. From the basic idea of drawing the heated gases through a screen and through the material to be dried was evolved the sectional cylinder, Fig. 1, so designed that the hot gases and air can be drawn through the periphery and into direct contact with the material, the latter being kept constantly "cascading" by the vanes or lifting blades, shown in Fig. 2, as the cylinder rotates in a clockwise direction when viewed from the furnace end. The inclination of the cylinder, of course, keeps the material moving longitudinally.

The cylinder, as will be seen from the illustrations, consists of six curved sections held together by six channel irons running the entire length of the sections. In the channels are slotted openings for the admisison of the heated air and gases. By reference to Fig. 2 it will be seen that there are six broad curved lifting blades directly opposite and inside the openings in the channel irons; these perform the dual duty of guarding

the air passages, thus preventing the drying material from falling out, and of lifting and cascading the material. The broad blades are assisted in the latter function by the small lifting blades.

A casual consideration of such a cylinder with peripheral openings of considerable size is apt to lead to the impression

FIG. 1.—SECTION OF DRYING CYLINDER.

that the material within the cylinder will, in operation, fall through the openings. A study of Fig. 2, either imagining it to rotate or actually rotating it in a clockwise direction, will show how this is impossible. Take the air inlets at the bottom and right of the view; they are cut in a surface which is in a radial plane; as it rotates under and to the left, material from the blades already up on the ascending side does not commence to fall into the "pocket" formed between the blade and the channel iron until the radial plane of the web of the channel is well past the vertical and the openings, consequently, at the upper side of the "pocket." Material from the small lifting blades does not fall into the larger until that plane is almost horizontal, hence none of the material can leave the cylinder.

In the equipment shown in Fig. 3 the furnace is coal-burning, with a mechanical stoker *F* driven from the rotating mechanism. This stoker is furnished with the dryer. By making changes in the furnace gas any other fuel may be used

In the comparatively short time that these dryers have been on the market they have been operated with a great degree of success in gunpowder plants, cement works and for glass manufacturing and paving work, where they handle large quantities of sand, gravel, etc., up to 200 tons per 10-hour day. Chemicals which will stand a moderate degree of heat, and, in fact, nearly everything which can be dried without vacuum is suitable "feed" for this dryer.

The method of installation of the dryer is shown in Fig. 3. The cylinder is surrounded by a steel casing, into which pass the heated air and gases from the furnace. An exhaust fan *E* draws the heated air through the openings into the cylinder, through the falling material within, and out at the exhaust. Set along the base of the casing are doors *G*, which may be opened if desired to reduce the temperature of the air and gases from the furnace before their delivery into the cylinder. The relief stack shown in Fig. 3 is used in starting the fire—if solid fuel is used—to carry off the smoke until the furnace is hot, and when the dryer is shut down.

In the instance of a large provision and packing company this dryer replaced another, modern type with a resulting increased capacity of more than 140 per cent. Sixteen pounds of water were evaporated per pound of combustible consumed on the stoker bars, and tankage, blood and bones were dried without deterioration, without blowing out dust and without danger of fire, as stated by Mr. W. M. Cummer, of the Atlas Dryer Company, Cleveland, Ohio, to whom we are obliged for much of the information in this article.

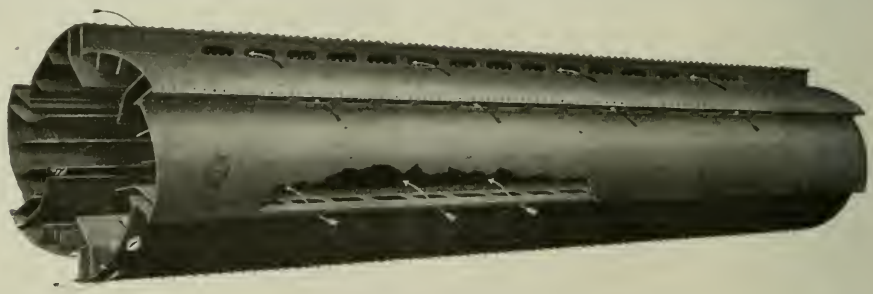


FIG. 2.—PERSPECTIVE OF DRYING CYLINDER.

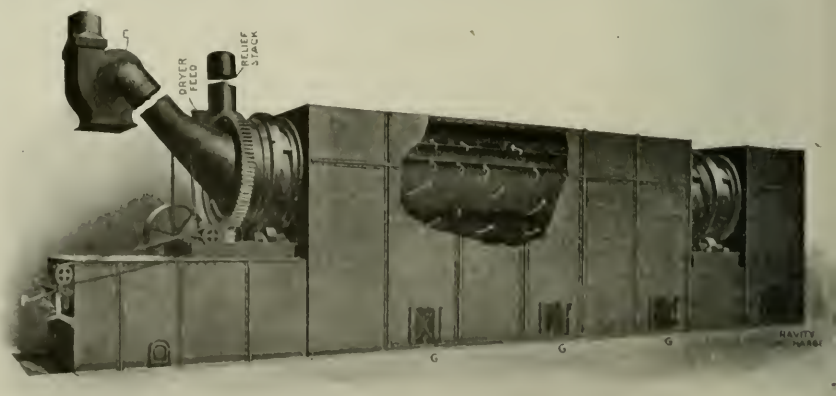


FIG. 3.—DRYER COMPLETE

In this dryer the hottest gases are brought in contact with the wettest material, which is fed in at the furnace end. The further the gases go from the furnace, the cooler they become and the drier also is the material with which they come into contact. The location of the exhaust blower at the front end gives the strong suction where the material is wet and heavy, and permits of a gradual reduction of pressure to practically zero at the discharge end, where the dried material is light.

Successful practical operation has demonstrated that material does not leak out of the cylinder nor is it drawn out as dust in the exhaust.

Production of French Bauxite.

The composition and properties of red French bauxite were described in our November issue (Vol. VII, page 497). During the last three years there has been shipped by the French company handling the product, an average of 46,000 tons of bauxite per year. In the two years preceding this period the shipments only reached 32,000 tons per year. Orders have been received calling for the shipment of 60,000 tons during 1910.

Not only has the amount of bauxite mined during the past five years equaled the tonnage shipped, but it has exceeded it, and the owners now have on hand, either in warehouses or at the mines, a ready stock of 400,000 tons.

The Continental-American Ore Company, of 33 Broad Street, Boston, Mass., are the American agents for this bauxite.

Notes.

Electric Steel Plant.—While the 15-ton Heroult electric furnace of the U. S. Steel Corporation at South Chicago has been in continuous operation since last summer, the second Heroult furnace of the same capacity at the Worcester works has been started in January, and is reported to work very satisfactorily. While at South Chicago the electric furnace is used for refining molten bessemer steel, open-hearth steel is electrically refined at the Worcester plant. The product is used chiefly for steel wire.

Contract for Girod Furnace.—C. W. Leavitt & Company announce that they have closed their second American contract for the installation of a Girod electric steel refining furnace.

Proposed Chemische Reichsanstalt.—Considerable interest has been created in the German chemical world by the decision to form an Imperial Chemical Institute. This institution is to undertake for the chemical industry similar functions to those which the Imperial Physical Institute performs for engineering. The undertaking is being subsidized by the State, and it is expected that the annual maintenance will cost about $10,000.

Cornell University is giving a course of lectures on industrial chemistry by practising chemical engineers. As part of this course Dr. John E. Teeple, of New York City, recently delivered five lectures on the manufacture of sulphuric acid.

Engineers' Society of Western Pennsylvania.—The thirtieth annual meeting of the Engineers' Society of Western Pennsylvania was held in the rooms of the organization, Pittsburgh, Jan. 18, Mr. E. K. Morse, who succeeded to the presidency on the death of George T. Barnsley in October, presiding. The annual report of the board of directors, including the reports of committees, sections and the treasurer was read, indicating the general prosperity of the society during the past year. The membership of the society at the close of the year was 835. The following officers were elected for the ensuing year: President, E. K. Morse; vice-president, J. O. Handy; treasurer, A. E. Frost; directors, A. R. Raymer, Willis Whited.

Western Society of Engineers.—As the result of a letter ballot the Western Society of Engineers has elected the following officers: President, John W. Alvord; first vice-president, O. P. Chamberlain; second vice-president, A. Bement; third vice-president, W. K. Hatt; treasurer, Albert Reichmann. W. W. Curtis is elected a new member of the board of trustees, and the retiring president, Andrews Allen, becomes an ex-officio member of the board of direction.

Mining Institute of Scranton.—This organization ended its year on the evening of December 23 by holding its annual banquet at Scranton, Pa., with a large attendance. Superintendent Phillips, of the Lackawanna Railroad, Superintendent Jennings of the Pennsylvania Coal Company, Prof. A. H. Welles, and Superintendent William H. Allen, of the Scranton Coal Company, were the speakers.

Canadian Mining Institute.—The twelfth annual general meeting of the Institute will be held in the city of Toronto, March 2, 3 and 4. The headquarters will be at the King Edward Hotel. Among those who are expected to prepare papers are: H. F. Bain, Dr. A. E. Barlow, P. McN. Bennie, William Brewer, R. W. Brock, H. P. H. Brumell, Colin St. G. Campnell, Thomas Cantley, Fritz Cirkel, J. M. Clark, Eugene Coste, Dr. James Douglas, John A. Dresser, F. W. Gray, Allan Greenwall, R. E. Hore, Frederic Keffer, J. D. Kendall, Thomas Kiddie, Frank E. Lathe, B. B. Lawrence, R. H. Richards, Dr. Heinrich Ries, Edgar Slansfield, W. J. Woolsey and H. H. Yuill. H. Mortimer Lamb, Windsor Hotel, Montreal, Quebec, is the secretary of the Institute.

Examinations in Chemical Technology.—The Council of the Institute of Chemistry of Great Britain and Ireland will arrange to hold examinations in chemical technology in April. The examinations will be open only to Fellows and to those associates who have been registered as such for at least one year who produce evidence of practical technological training.

The Chemical Society of London, in honor of the completion of fifty years' fellowship by their past presidents, Sir Henry Roscoe, Sir William Crookes, Dr. Hugo Müller and Dr. A. Vernon Harcourt, will entertain them at a dinner of the society to be given some time near the end of May or the beginning of June.

New South African Engineering Institution.—The first annual meeting of the South African Institute of Electrical Engineers was held recently under the chairmanship of Mr. J. R. Bradley. Applications for membership were received from 158 persons, of which 71 have been elected members, 65 associate members, 14 associates and 8 students. Mr. C. M. R. Campbell was elected president, and Mr. W. Eldson-Dew and Mr. M. Rohman were elected vice-presidents.

The French Academie des Sciences have awarded the Hébert prize to M. Paul Janet for his work on alternating currents covered in the new edition of his "Leçons d'Electricité," the Hughes prize to M. Meslin for his work in optics and allied fields, and the Gaston Planté prize to M. Jean Perrin for his work on cathode rays.

Electric Process Quartz Vessels.—A company with a stock capital of 200,000 marks has been organized for the erection of works at Biebrich am Rhein to manufacture quartz vessels (fused silica) by the electric furnace process.

The Hardinge Conical Mill Company of New York City has established a tentative branch office at 415 Bailey Building, Seattle, Wash. Mr. V. G. Lunt, former secretary to Mr. Hardinge, is in charge of this branch, which will operate as a special sales office.

Hoisting Apparatus.—The Brown Hoisting Machinery Company, of Cleveland, Ohio, issues a full library of catalogs covering the different forms of machinery made by them. In the list are catalogs dealing with crabs and winches, blast furnace-hoists and stock distributors, buckets and tubs, locomotive-coaling stations, sewer and trench machines, locomotive ore-handling cranes, coal and coke-handling machinery, all-steel window sashes, and "Ferroinclave." This latter is a corrugated roofing which is erected on steel supports and then covered with cement. The circular illustrates several novel uses to which the product may be put, such as the construction of stairs, side-walls, highway bridges, culverts, water tanks, stacks and coal bins. We will deal with some particularly interesting subjects found in these catalogs in a future issue.

Export Office Opened.—The General Fireproofing Company, of Youngstown, Ohio, has opened an export office at 396 Broadway, New York City, in charge of Mr. A. D. Level, who has been for several years with the United States Steel Products Export Company.

The Société électro-métallurgique piémontaise at Ixelles has organized a new company to take up the handling of auriferous ores by the *Froment* process. The capitalization has been set at 2,500,000 francs.

Cyanide Plant Opens.—The old mill of the Natividad y Anexas Mining Company in the Sierra Juarez district, State of Oaxaca, Mexico, has been closed down and the ores will be treated by the new cyanide plant, which has just been finished.

Bonus for Ethyl Alcohol Plant.—The citizens of Parry Sound, Ontario, have voted a by-law to loan $30,000 to the Union Iron Works for the establishment of an iron-working plant, and also a $10,000 bonus for the erection of an ethyl alcohol plant.

A large Carbide works will be erected in Darfe in Camonicatal by the Aktien-Gesellschaft Ferriere di Voltri. The factory is situated favorably in a very good limestone district which is also rich in water-power.

New Factories.—Carl Zeiss and Company are building a large factory at 29 Margaret Street, London, E. C. The London chemical firm of Allen & Hanbury, Limited, is erecting an extensive plant at Hull.

Passaic Steel to Operate Again.—The property of the Passaic Steel Company, sold recently under the direction of the Chancery Court, has been taken over by a corporation composed of members of the bondholders' committee, and will be operated under the name of the Passaic Structural Steel Company. The new corporation paid $400,000 for the plant and $80,000 to the city of Paterson, N. J., for back taxes. The works were sold by the Passaic Rolling Mill Company to the Passaic Steel Company a few years ago for $1,800,000.

Aluminium.—The British Aluminium Company of 109 Queen Victoria Street, London, has issued a folder giving full particulars, resistance weights, and the like, of standard aluminium conductors ranging in size from 0.000924 sq. in. to 1.0606 sq. in. in cross-section, in comparison with the most nearly equivalent sizes of copper conductors. This comparison serves to show that the aluminium conductors weigh but half as much as copper conductors of equal capacity.

Metallurgical Apparatus.—Among the products of the Patterson Foundry and Machine Company, of East Liverpool, Ohio, of interest to metallurgists and chemists are grinders and pulverizers, filter presses, engines and pumps, and elevating and conveying machinery. A small size filter press built along the lines of the standard "Patterson Round Rail" press and a line of small grinding cylinders are made specially for laboratory use.

Gasoline and Distillate Hoist.—The R. Kelly Machinery Company, of Los Angeles, Cal., recently placed a new gasoline and distillate hoist on the market. The engine is of the marine type and is built in units of 10-hp each. If additional power is required it can be obtained by adding extra cylinders. The hoist can readily be taken apart for the purpose of transportation, and where desired the base and sides are made of one solid casting, insuring the greatest possible strength.

Hopkins Fumeless Zinc Process.—A company has incorporated in Great Britain with a capitalization of $45,000 under the name of the Hopkins Fumeless Zinc Process Company. The purpose of the concern is to acquire and turn to account any improved processes for smelting zinc, especially that process covered by the patents of Reverend E. H. Hopkins. In this issue is an article fully describing the process.

Crucible Firm Organized.—The North British Crucible Company, Limited, has been incorporated in England to manufacture crucibles and like products. The capitalization is fixed at $25,000 and the address registered is 24 Blythswood Square, Glasgow.

Rustless Metal Organization.—The Rustless Metal Company, Limited, has been incorporated in England with a capital of $10,000. Of this $7500 is issued as common and $2500 as preferred stock.

Purifying Water with Ozone.—The United Water improvement Company, of Philadelphia, issues a pamphlet arranged in rather popular form describing their process for the purification of water by ozone. The ozone, electrically produced, is brought into close contact with, and partly dissolved in, water divided into a foamy consistency in the pipes arranged for the purpose. By this means it is said that the ozone combines with the carbon constituents in the pathogenic organisms, thus destroying them. The company lists apparatus in all sizes, from the small installations for domestic use up to municipal plants.

Oxy-Acetylene Welding and Cutting.—The great convenience of the oxy-acetylene process of welding iron, steel and other metals, even aluminium, is now so well recognized that it is being introduced into many factories and repair shops. It is rapidly replacing riveting in sheets up to ½ in. thickness, since it is less expensive, more efficient, and gives neater results. For cutting iron or steel plates the oxy-acetylene burner is much more rapid than pneumatic tools, punches or shears. It is, therefore, not surprising that there are now quite a number of concerns in this country furnishing oxy-acetylene apparatus. We have just received the catalog of the American Oxygen Company, 430 Perry Building, Philadelphia, Pa. It illustrates their torch, special features of which are said to be the method of mixing the gases, the regulation of the oxygen gas by a shut-off cock in case of flash-outs. Their acetylene pressure generator withstands a pressure of 50 lb. But the working pressure is given as 12 lb. The company has an oxygen plant in Camden, N. J., for the production of 10,000 cu. ft. of oxygen daily. While they furnish oxygen gas in high or low-pressure tanks, they are also developing a small portable oxygen generator for use at such places which are too far away from their oxygen-supply station.

Additional Power Installation.—On January 1. the Virginia (Minn.) Electric Power & Water Company began the operation of the additional power apparatus, which has just been installed by the Allis-Chalmers Company. The new apparatus consists of a 37-kw, low-pressure turbine direct-connected to a 2300-volt, 60-cycle, three-phase alternator. The turbine receives steam at a pressure of 16½ lb. absolute and a 28-in. vacuum is maintained by a Tomlinson type "C" condenser. Provision is made for supplying the steam ordinarily from the exhaust of a reciprocating engine, but in case of necessity high-pressure steam can be supplied direct from the boilers through a reducing valve.

Reconstruction of Works.—The American Platinum Works, of 251-255 New Jersey Railroad Avenue, Newark, N. J., have acquired by purchase the real estate and building formerly owned and occupied by the Caffrey Leather Company and J. Liebstein & Sons, located at 225, 227, 229 and 231 New Jersey Railroad Avenue, Newark, covering half a block on that avenue (110 ft.) and having a frontage of 250 ft. on Oliver Street and 30 ft. on Chestnut Street. The present buildings will be reconstructed and modernized and additional structures will be erected, which, when finished, will give this company one of the most complete plants in the world for the refining of platinum, gold and silver.

Addition to Plant.—The Northern Engineering Works, builders of "Northern cranes," Detroit, Mich., have purchased additional land adjoining their plant on which they are preparing to make extensions of their crane plant.

Power from Exhaust Steam.—The fact that only a small portion of the available energy in steam is turned into work by engines when exhausting to atmosphere, and that reciprocating engines are not adapted to operate with high vacuums has caused considerable investigation lately as to the best methods

of getting more work out of the steam. . The method of using a low-pressure turbine working on the exhaust of a high-pressure reciprocating engine has lately come into use. However, in the majority of cases, the amount of steam exhausted by the engine is not proportional to the amount of steam needed by the turbine. This is especially true in the case of rolling-mill engines, hoisting-mill engines, etc., where the exhaust is intermittent. A new apparatus, known as a regenerator, has recently been put on the market for absorbing any surplus steam and storing it up to use in the turbine when the exhaust supply ceases. A booklet entitled "Power from Exhaust Steam," published by the American Regenerator Company, Monadnock Block, Chicago, Ill., goes into this subject quite fully and shows by calculations how the maximum amount of available energy in exhaust steam can be turned into useful work. There are also diagrams showing the comparison of efficiencies of various combinations of reciprocating engines and steam turbines.

Addition to Hydroelectric Plant.—The Portland (Ore.) Railway, Light & Power Company is again increasing the capacity of its hydroelectric plant. It has recently placed an order with the Allis-Chalmers Company for a 3750-kw waterwheel type, 11,000-volt, 60-cycle, three-phase, 360-r.p.m. alternator. This will be of the same general design as the two 2500-kw, 11,000-volt, 33-cycle alternators already supplied by the same company, and will be semi-enclosed. A 50-kw, 120-volt exciter will be direct-connected to an extension of the alternator shaft.

Gold in New York State.—Mr. John M. Clarke, New York State geologist, recently made a report to the Legislature dealing with the supposed gold sands of the Adirondacks, which have attracted so much attention from promoters and prospectors. He concludes that while they contain traces of the precious metal, they do not afford any basis for mining enterprises. "While deficient in the precious metals, the Adirondacks region possesses the elements of large mineral wealth. Its production of iron ores, talc, garnet, graphite, pyrite, and building and monumental stones of various character is a very important contribution to the mineral output of the country. There is abundant opportunity for future development of such resources, and it is in that direction rather than in the seeking of the precious metals that energy and capital can be best employed."

Combination of Firms.—The Trent Engineering Company, composed of the firms of L. C. Trent & Company, of Salt Lake, and the Trent Engineering Company, of Reno, Nev., having combined, have opened a branch office in Los Angeles, Cal., for the extension of its business and for the better convenience of its clients.

Foundry Supply.—Among the catalogs received at our new Pittsburgh office may be mentioned catalog No. 37 of the J. D. Smith Foundry Supply Company, of Cleveland, Ohio. The book is quite complete and covers foundry supplies of all kinds, facings and general equipment. Standard natural draft crucible ovens, core ovens and core-making machines are worthy of special mention.

Dependable Engineering Specialties is the title of the fully illustrated catalog No. 9 of the Wm. Powell Company, of Cincinnati, Ohio, which covers a great many types of valves, globe, gate, check, blow-off and throttle, as well as cocks, lubricators, injectors, whistles and other details. A number of tables of useful information are included. Metallurgists who have in operation steam or gas engines will find this catalog convenient.

Metallurgical Machinery.—Among the most complete sets of catalogs we have received is an assortment from the Wellman-Seaver-Morgan Company, of Cleveland, Ohio. Two general divisions are treated and are outlined in two little illustrated booklets: I. "What We Do in Iron and Steel Works Machinery, Etc." II. "What We Do in Mining and Power Machinery." Each field is covered by catalogs on various specialties, such as open-hearth and reheating furnaces, charging and manipulating apparatus, cranes, buckets, gas producers, reversing valves, hoisting engines and electric hoists, cages and ore cars, tube and ball mills, concentrators, etc. The publication on "Ore and Coal Handling Machinery" is worthy of special commendation.

Sherarduct.—A number of circulars from the National Metal Molding Company, of Pittsburgh, Pa., announce their product "Sherarduct," an iron pipe conduit rendered non-corrosive by sherardizing. This process of galvanizing has frequently been mentioned in these columns and, it will be recalled, consists of enclosing the cleaned metal, together with zinc and zinc oxide dust, in a chamber which is then heated to about 600° Fahr. The zinc alloys, with the iron and a coating of pure zinc, remains on the surface. The United States Sherardizing Company, of New Castle, Pa., owns the American patents and has given licenses to several companies which are now "dry-galvanizing" a variety of articles.

Gas Producers and Valves.—A couple of artistic catalogs from the Forter-Miller Engineering Company, of Pittsburgh, Pa., describe their specialties, water-seal gas producers and water-seal reversing valves, together with open hearth, crucible, reheating and annealing furnaces built by this firm.

The Buckeye Engine Company of Salem, Ohio, has issued a trio of publications. The first deals with steam engines in different forms in sizes up to 8000 hp, while the second treats of four-cycle gas engines from the small sizes to the heavy twin double-acting tandem engines of 6000-hp capacity. The comparisons between thermal efficiencies of steam and gas engines and the typical analyses of various power gases are instructive. "The Sun That Never Sets" is a neat booklet on the Buckeye blue printing machine.

Small Gas Engines.—In the gas-engine field, a field of constantly growing importance in the iron and steel and general metallurgical industries since the more universal introduction of the gas producer, might be mentioned publications of the Foos Gas Engine Company, of Springfield, Ohio. This concern appears to specialize in engines of small and moderate size, say up to 150 hp. The machines are four-cycle, single-acting; the horizontal type has one, and the vertical two, three, or four cylinders. "Foos Junior" equipments are made in from 2-hp to 8-hp units.

Electric Control Apparatus.—From the Pittsburg office of The Electric Controller and Manufacturing Company comes a set of folders illustrating their line of magnetic brakes, motor starters, controllers, magnetic switches and solenoids, electric fault finder, and the well-known lifting magnets. This company has adapted its products particularly for the iron and steel industry, and has now a great many successful installations to its credit.

Laboratory Materials.—A valuable reference book for purchasing agents, chief chemists, and college professors is the catalog of the Scientific Materials Company, of Pittsburgh. The list is remarkably complete and includes some 3000 different forms of apparatus used in chemical laboratory work, each one of which is listed in all the sizes ordinarily used. With the alphabetical classification and the 2000 illustrations accompanying the descriptions, no difficulty is experienced in rapidly locating anything desired.

Additional Catalogs Received.—Among the publications received at our Pittsburgh office which it is not possible to review this month are bulletins from the Lincoln Electric Company and the Triumph Electric Company on small motors and generators; the Bruce-Macbeth Engine Company and the Miller Improved Gas Engine Company, on gas engines; William B. Scaife & Sons Company, on high-pressure tanks and water filters; the Bickford Firebrick Company, and the Grasselli Chemical Company.

Personal.

Mr. **John Fritz**, of Bethlehem, Pa., has been elected an honorary vice-president of the Iron and Steel Institute of Great Britain.

Mr. **Philip Argall** announces that in order to give prompt attention to his rapidly increasing business, he entered on Jan. 1 into partnership with Mr. Philip Henry Argall, late of the Selby Smelter, and Mr. George Oates Argall, general manager of the Iron Silver Mining Company, of Leadville, Col., with the purpose of conducting a general business as consulting, mining and metallurgical engineers, under the name of Philip Argall & Sons. The offices of the firm are in the Majestic Building, Denver, Col.

Dr. **Ernst J. Lederle,** consulting sanitary engineer for the New York City Board of Water Supply and a member of the New York State Water Supply Commission, has been appointed commissioner of the New York City Health Department. Dr. Lederle occupied the position before, during the Seth Low administration, beginning in 1902.

Mr. **G. H. Dowell,** who for a number of years has been employed by the Phelps-Dodge interests at the Douglas, Ariz., smelter, has been appointed general manager of the Old Dominion. Mr. George Kingdon, who has been acting superintendent since the resignation of Mr. R. B. Hegardt some time ago, will henceforth be superintendent of mines, and also general superintendent of the United Globe property controlled by the Old Dominion Copper Mining and Smelting Company.

Mr. **E. B. Raymond,** who has been General Superintendent of the Schenectady works of the General Electric Company, is leaving that concern to enter the employ of the Pittsburgh Plate Glass Company in the position of second vice-president. Mr. Raymond is a graduate of the Massachusetts Institute of Technology, class of 1890.

Mr. **Donald G. Miller** is doing engineering work at Pilares de Nacogori, Sonora, Mexico, for the Montezuma Copper Company.

Mr. **Fred B. Crosby** has left the power and mining department of the General Electric Company to take up a position as technical writer with the publication department of the Westinghouse Electric & Manufacturing Company.

Mr. **Frederick H. Keyes,** formerly general manager of the Robb-Mumford Boiler Company, on Jan. 1 associated with Messrs. T. W. Sprague, H. D. Jackson and others to carry on a general consulting engineering business. The offices of the firm will be at 88 Broad Street, Boston.

Mr. **Arthur A. Brown,** formerly manager of the Pittsburgh office of the Westinghouse, Church, Kerr Company, recently accepted a position in the sales department of the Westinghouse Electric & Manufacturing Company. Mr. Brown's headquarters will be in New York City.

Mr. **George C. Vivian** has severed his Utah mining connections, and has returned to Colorado, where on Jan. 1 he took charge as manager of the Golden smelter of the North American Smelter & Mines Company.

Mr. **W. D. Healy,** formerly of Detroit, Mich., has entered the employ of the Union Pacific Cobalt Company as manager.

Mr. **A. B. Emery** has been appointed assistant general manager of the Southern Division of Mexico Mines of the American Smelters Securities Company. Mr. R. E. Adams succeeds Mr. Emery as superintendent of the Velardeña district. Mr. W. M. Drury has been appointed assistant general manager of the Northern Division of Mexico mines of the company.

Mr. **Andrew McFarland,** of Colorado, has taken charge as general superintendent of the Lucia Mining Company, Gabrial, Durango, Mexico.

Mr. **C. M. Eye** has resigned as manager of the Benguet Consolidated Mining Company, at Bagulo, Philippine Islands, to take charge of the Colorado Mining Company at Aroroy, in the Island of Mashate His successor with the Benguet company is Mr. Amos W Cole, recently with the Oriental Mining Company, in Korea.

Mr. **Kamajiro Kajiura,** engineer for the Besshi Copper Mines Company, at Nuhama, Iyo, Japan, has been in this country recently studying the smelter fume and smoke problem. His company is operating its smelter on an island 10 miles from the coast of Japan, but even at that distance is having trouble with the farmers on the mainland, who complain of the fumes as a nuisance.

Mr. **I. Wayne Von Leer,** of New York, has been appointed manager of the Arizona United Mining Company, at Johnson, Ariz.

Mr. **Frederick N. Pease,** Assistant Chemist of the Pennsylvania Railroad has been appointed chief chemist to succeed the late Dr. Charles B. Dudley.

Professor **Theodore W. Richards,** of Harvard University has been reappointed research associate of the Carnegie Institution. With the appointment he receives a new grant of $2,500 to enable the continuation of his researches regarding atomic weights and physico-chemical constants. At the last meeting of the Chemists' Club, of New York City, which was in the form of a Harvard night, Professor Richards was made an honorary member of the club.

Dr. **Rudolph Messel** has been appointed Honorary Foreign Secretary of the Society of Chemical Industry to take the place of the late Dr. Ludwig Mond. Professor A. Liversidge, F.R.S., has been appointed a vice-president of the society, succeeding Dr. Mond.

Dr. **L. F. Guttman,** formerly of the chemical department of the College of the City of New York, has been called as assistant professor of chemistry to Queen's College, Kingston, Ontario, Canada.

Mr. **W. M. Tattersall** has been appointed keeper of the Manchester (England) Museum, to succeed Dr. W. E. Hoyle.

Professor **J. Culver Hartzell** has resigned as head of the department of geology and chemistry at the University of the Pacific. The resignation is to take effect at the close of the present academic year.

Dr. **Albert P. Sy** has become director of the chemical laboratories and professor of chemistry at the University of Buffalo, succeeding Dr. H. N. Hill, resigned.

Dr. **Austin M. Patterson** has been elected Editor of *Chemical Abstracts,* to succeed Dr. W. A. Noyes. Mr. John M. Miller becomes associate editor.

Dr. **Henry M. Payne,** who recently resigned his position as professor of mining engineering at the University of West Virginia, has been appointed field manager of the Hydraulic Mining Cartridge Company, of 42 Broadway, New York City.

Dr. **Benzo Katsura,** Professor of Metallurgy in the Imperial Tokio (Japan) University, was a recent visitor to the United States and made a tour of the metallurgical plants of the country. He secured many publications for the University library.

Dr. **J. L. Simonsen,** assistant lecturer and demonstrator in the chemical department of the University of Manchester (England), has been called to the chair of professor of chemistry at the University of Madras (India). Dr. A. Holt takes the place left vacant by Dr Simonsen at Manchester.

Dr. **G. Gürich,** docent in the department of geology at Breslau, has been appointed director of the Geological Institute at Hamburg, to succeed the late Professor Gottsche.

M. **Emile Picard** has been chosen president of the Paris Academy of Sciences for 1910. The place he formerly held as vice-president will be filled by M. Armand Gautier.

Dr. **Charles Graham,** formerly professor of chemical technology at the University College, London, has bequeathed his residual estate, which is estimated to amount to $175,000, to the college for research in the School of Advanced Medical Studies at the University of London.

Dr. **Ernst J. Berg** is now head of the electrical engineering department at the University of Illinois, having left the General Electric Company to take up this new position. He is a native of Sweden, receiving his education at the Royal Institute of Technology, Stockholm, graduating in 1892, and coming at once to America. While with the General Electric Company

he became chief assistant to Dr. Steinmetz. For the past two years he was Consulting Professor of Electrical Engineering at Union University. Recently he received the degree of Sc.D. from that institution

Tau Beta Pi.—The twenty-third chapter of Tau Beta Pi, the honorary engineering fraternity, was installed at Cornell University on Jan. 17, by Professor L. E. Moore, assistant professor of civil engineering at the Massachusetts Institute of Technology, and president of the fraternity.

The **Society of Sigma Xi** held a business meeting and banquet in Boston on the afternoon and evening of December 29, on the occasion of the meeting of the American Association for the Advancement of Science.

Phi Lambda Upsilon.—There is now at Columbia University a chapter of the honorary chemical fraternity, Phi Lambda Upsilon. Its faculty members are Profs. C. F. Chandler, M. T. Bogert, J. L. R. Morgan and H. C. Sherman. Mr. J. B. Barnitt is president of the chapter, Mr. Charles Crew is vice-president, and Mr. Robert Schwarz secretary-treasurer.

American Electrochemical Society

Seventeen applicants were elected to membership in the American Electrochemical Society at the meeting of the board of directors of that organization held in Philadelphia on Friday, Dec. 31, 1909. Announcement is made that the next meeting of the society will be held in Pittsburgh the latter part of April.

The list of new members elected follows:

Messrs. Augustus H. Eustis, Mining Engineer, Boston, Mass.; Walter L. Badger, Chemist, Great Western Sugar Company, Brush, Col.; William S. Barwick, Chemist and Metallurgist, The Vancouver Engineering Works, Vancouver, B. C.; Charles E. Skinner, Engineer, Research Division, Westinghouse Electric Manufacturing Company, Wilkinsburg, Pa.; George M. Colvocoresses, Superintendent, Millerett Silver Mining Company, Gowganda, Ontario; Lorain E. Imlay, Superintendent, The Niagara Falls Power Company, Niagara Falls, N. Y.; Harold Binney, of Binney, Mastick and Ogden, New York City; James D. Pickard, Mechanical Engineer, The Colorado Vanadium Company, Boulder, Col.; Edgard de Souza, Chief Electrical Engineer, The S. Paulo Tramway Light and Power Company, Ltd., S. Paulo, Brazil; C. H. Bowman, President, Montana State School of Mines, Butte, Montana; Jorge Newbery, City Electrical Engineer, Buenos Ayres, Argentine Republic, S. A.; Frederick H. Loveridge, Electrical Engineer, Western Electric Company, Chicago, Ill.; T. K. Armachela, Assistant Engineer, Public Works Department, Mysore Government, Bangalore, S. India; Floyd D. Taylor, Student of Electrochemical Engineering, The Pennsylvania State College, State College, Pa.; Herbery F. Sill, Assistant Professor of Chemistry, Carnegie Technical Schools, Chemical Dept., Pittsburgh, Pa.; Bradley Stoughton, Consulting Electric Furnace Engineer, New York City; William J. Cummings, C. E., Chief Engineer, E. I. DuPont Powder Company, Haskell, N. J.

American Chemical Society

The following have been elected to membership in the American Chemical Society.

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NEW BOOKS.

MORGAN, J. LIVINGSTON RUTGERS. Physical Chemistry for Electrical Engineers. Second edition, revised. 259 pages, illustrated. Price, $1.50. New York: John Wiley & Sons.

NAGEL, OSKAR. The Transportation of Gases, Liquids and Solids by Means of Steam, Compressed Air and Pressure Water. A complete description of the theory, construction, operation and application of jet machines, montejus, spray nozzles, etc., for chemical, metallurgical, mining, mechanical and civil engineers, with 193 illustrations. 212 pages. Price, $2.00 net. New York: Oskar Nagel (P. O. Box 385).

THORPE, Sir E. History of Chemistry. In two volumes. Vol. 1. From the earliest times to the middle of the nineteenth century. 217 pages. Price, 75 cents net. New York: Putnam.

ALLEN, ALFRED H., M. D. Commercial Organic Analysis. Fourth edition. Vol. 1, introduction. Alcohols, yeast, malt liquors and malt, wines and spirits, neutral alcoholic derivatives, sugars, starch and its isomerides, paper and paper-making materials, vegetable acids. Edited by Henry Leffman and W. A. Davis. 586 pages. Price, $5.00 net. Philadelphia: P. Blakiston's Son & Company.

LEACH, ALBERT ERNEST. Food Inspection and Analysis. For the use of public analysis, health officers, sanitary chemists and food economists. Second edition, revised and enlarged. 972 pages, illustrated. Price, $7.50. New York: John Wiley & Sons.

WATERBURY, LESLIE ABRAM. A Vest Pocket Handbook of Mathematics for Engineers. 197 pages, cloth. Price, $1.50. New York: John Wiley & Sons.

MACLAURIN, RICHARD COCKBURN. Light. Cloth, 257 pages. Price, $1.50. New York: The Macmillan Company.

NAGEL, OSKAR. Producer Gas Fired Furnaces. Cloth, 187 pages. Price, $2.00 net. New York, the author, P. O. Box 385.

BOOK REVIEWS.

The Mechanical World Pocket Diary and Year Book for 1910. 300 pages, price 6d. (Retail price in New York, 25 cents.) Manchester, England: Emmott & Company, Ltd.

This handy little reference book has undergone a number of improvements this year. First of all, it has been reduced in bulk by the use of thinner paper, although considerable new matter has been added.

The gas engine section has been thoroughly revised and a new section on oil engines added. A section on ball bearings and some things on centrifugal pump design have also been added. There are numerous other additions, most of which relate to the construction and design of machine details, such as gears, bearings, etc.

The 300 pages of text are well filled with data condensed in a most concise form. The collection of tables is exceptionally well chosen and will be found most useful to those who have to do with the installation and operation of prime movers.

* * *

The Mechanical World Electrical Pocketbook for 1910 175 pages, price 6d. (Retail price in New York, 25 cents.) Manchester, England: Emmott & Company, Ltd.

This is a companion to the Mechanical World Year Book, and is evidently intended for mechanics and electricians rather than engineers. It contains very short descriptions of various apparatus and gives directions for wiring and installation of apparatus, testing of circuits and making simple measurements. The wiring practice is, of course, that laid down by the British code and does not apply in this country. There is practically no data in tabular form that is not to be found in all handbooks; that is, tables of squares and cubes, areas and circumferences, logarithms, etc. This book is far behind its companion, the Year Book, as a reference book of useful data.

Les Découvertes modernes en Physique. By **O. Manville.** Second edition. 8vo, 462 pages. Price, 8 francs. (Retail price in New York, $2.10.) Paris: A. Hermann et Fils.

The work consists of two parts: the first deals with the relation of electricity and matter; the second with the rôle of ions and electrons in physical phenomena. The first part has chapters on the electric conductivity of liquids, of gases, the ionization of gases, and the electronic theory of matter. The second deals with electrolysis, electric discharge in gases, electric conductivity of metals. A supplementary chapter treats of the modern conceptions of the relations of matter and ether. Our general impression of the work is that it contains a great deal about interesting, modern developments, rather uncritically worked up. Twelve pages of notes and errata (mostly the latter), which the reader is asked to kindly insert at the proper pages (a labor of several hours), complete the volume.

* * *

Analyse Thermique et Metallographie Microscopique. By **E. Rengade,** Docteur des Sciences, Chef de trauvaux, pratiques à la Faculté des Sciences. Cloth, 176 pages, 53 figures and 26 photographs. Price, $1.80. Paris: Librairie Hachette et Cie.

This book is intended for beginners. It has a preface by H. Le Chatelier which clearly sets forth the guiding ideas and topics the book treats of. Solutions, alloys, glasses, solders, rocks, etc., have until recently not been considered worthy of intimate study by chemists. By the newer methods these classes of natural objects have been made the subject of close scrutiny and the methods which this little work describes so tersely and clearly have been introduced into many laboratories. The first portion of the text deals with Gibbs' rule, which is made the starting point for the development of the succeeding chapters. Numerous illustrations add greatly to the elucidation of the topics discussed and at the end there is a chapter on the apparatus and the methods of thermic analysis. The use of the thermo-electric couple is given and apparatus whereby continuous records of the variations of the current can be secured. There is a special chapter on the iron-carbon system.

The second portion of the work is devoted to metallography. This part is illustrated by some excellent micro-photographs. The technical preparation of the metal is given and the simple character of the methods is in strong contrast with the great insight thus obtained into the structure of materials, particularly steels.

* * *

Die Lack und Firnisfabrikation, by Professor **Max Bottler.** 130 pages, 29 illustrations. Price, 4.50 marks. (Retail price in New York, $1.50.) Halle, a. S.: Wilhelm Knapp.

This is No. 16 of a very interesting and valuable series of monographs on chemical, technical and manufacturing methods, edited by Mr. L. Max Wohlgemuth and issued by the publisher, Wilhelm Knapp.

In his preface the author says: "In the industries resting upon chemical bases, the manufacturer of lacquers and varnishes occupies a conspicuous position. In recent years this manufacturer has made important progress, partly in improvement of long known processes, but partly also in wholly new discoveries or processes. In the work before us will be found practical considerations of the newer inventions and technical advances. Special attention is given to the different methods of operation."

The author has well stated the plan and scope of the work. The nine chapters discuss the history of lacquers and varnishes, their classification, the raw materials of the industry, preparatory treatment and generalities, oil varnishes and fatty lacquers, oil of turpentine and benzine lacquers, alcoholic and other "volatile" and "non-volatile" lacquers, coloring, decolorizing and filtering lacquers and varnishes, preparation of printing and lithographing varnishes, soap lacquers, resinate lacquers, and casein lacquers.

Chemists seek in vain for good books in English on this subject, to serve as a guide for those directly or indirectly interested in the industry. For such as can read German this book will prove a boon, and all chemists, whether varnish makers or not, will find it a valuable addition to their working libraries.

* * *

Cost of Mining. By **J. R. Finlay.** 415 pages. Price, $5 net. New York: McGraw-Hill Book Company.

It has ever been the aim of this journal to be broad, to look at the domain of chemical and metallurgical engineering from such a standpoint that things could be seen in their proper perspective, but at the same time to scrutinize the essential details necessary for practical success. In short, we have endeavored to be far-sighted in the right way and to be near-sighted in the right way without being piddlers. This being our aim in technical journalism, we are glad to call attention to Mr. Finlay's "Cost of Mining" because the author attains successfully in his book such a consummation.

Mr. Finlay has achieved success in various fields of human activity. At Harvard, he was noted as an athlete on the field and on the water. Later, after serving his engineering apprenticeship, he has managed successfully mining properties of the first and second magnitude in Cripple Creek, the Cœur d'Alenes, Lake Superior iron district, Joplin, the Flat River district and elsewhere. He has at different times acted as consulting engineer, lectured at Harvard University, and lately assumed charge as general manager of the Goldfield Consolidated Mines Company which is making the largest net profit of any gold property in the world.

Mr. Finlay is a master of lucid English and composition, and has contributed many good articles to the various mining journals. With such all-round ability in the author, little comment is needed on the book.

Mining and metallurgy are closely associated. The first finds the valuable ore and gets it out of the ground. The second takes the valuable ore and turns it into a product that can be sold at a profit. To make a rabbit stew, first catch and skin your rabbit; next be sure you have a fire and a stew-pan in which to prepare the rabbit for the table.

Looked at broadly, the question of cost of mining and treating of ores is of prime importance. For mining and reduction procedures are governed by the principles of economic science as well as by those of natural science and engineering science. Mr. Finlay, we are glad to note, always analyses the subject from the tests of cold-blooded business. This is perfectly proper in a book entitled "Cost of Mining." The evaluation of mining property is first taken up and incidentally a clear light is shed on the warped view that the public takes of mining investments. Next, the cost of development work or finding the ore, and stoping it, raising it, and reducing it is dealt with in a philosophic manner. Finally having laid down certain general principles, the author analyzes, from the viewpoint of these principles, the reports of representative coal, iron, lead, copper, silver, gold and zinc companies.

Many illuminating passages might be cited. On page 11 it is neatly shown how the efficiency of labor varies, being high in hard times and low in good times. The chapter on the U. S. Steel Corporation is especially good. On pages 371-3 a clear exposition is given of the dictum of the first Lord Brassey, a great English contractor, that the same work costs the same money all over the world regardless of the wages paid the unskilled laborer.

Everywhere in the book is seen due regard for the laws of economic science, which to our mind are as far reaching as the law of gravitation, though less immediate in their action.

This book is quite to our liking. It can be recommended alike to the student of finance, to the practical business man, to the mining engineer, to the chemical or metallurgical engineer, or in point of fact to anyone interested in the wresting of mineral wealth from Nature's bosom.

Metallurgical and Chemical Engineering

A Consolidation of Electrochemical and Metallurgical Industry and Iron and Steel Magazine

Vol. VIII. New York, March, 1910 No. 3

PUBLISHED MONTHLY BY THE

Electrochemical Publishing Company

239 West 39th Street, New York

Pittsburgh Office, 413 Lewis Block.
European Office, Hastings House, Norfolk St., Strand, London, Eng.

J. W. Richards, Ph.D.....*President*
E. F. Roeber, Ph.D.....*Editor and Secretary*
C. E. Whittlesey.....*Treasurer*

Yearly subscription price for United States, Mexico and United States dependencies, $2.00; for all other countries, $2.50 (European exchange, 10 shillings, 10 marks, 12.50 francs).

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Entered as Second-Class Matter at the Post Office at New York, N. Y., under the Act of Congress, March 3, 1879

New York, March, 1910.

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The Eighth International Congress for Applied Chemistry.

On page 118 we print the official report of the preliminary meeting in which the organization of the Eighth International Congress was considered that is to be held in this country in 1912 with Prof. Edward W. Morley as honorary president and Dr. W. H. Nichols as acting president. It is a pleasure for this journal to endorse here fully the sentiment of chemical democracy expressed by Dr. Nichols at that meeting— that the central and vital point to be observed is that every chemist in the United States should feel that he is part of the Congress from the very creation of the organizing committee down through all the details of management to the very close of the Congress and that everything should be done which could in any way contribute to the realization of that thought. We endorse with equal sincerity the other leitmotiv of Dr. Nichols' suggestions "that quality and not quantity should count, that science should count against mere commercialism or ostentatious show." We recommend the official statement on page 118 to the earnest attention and consideration of all our chemical and metallurgical readers.

Ferrosilicon.

The extended article on the manufacture and transport of ferrosilicon, printed elsewhere in this issue, will be found to contain a considerable amount of useful and interesting information. But, we state frankly, that in publishing this information we do not intend or expect to encourage the erection of further ferrosilicon works at present. Up to a few years ago the higher grades of ferrosilicon, which require the use of the electric furnace, were almost exclusively made in Europe. The product came largely from that string of mountain electric-furnace plants, notably in southern France, which had originally been erected for calcium carbide manufacture, but had to look out for some other manufacture when the European carbide boom collapsed. During the last years the manufacture of ferrosilicon has been taken up in this country (which is really the pioneer country of the manufacture of other alloys in the electric furnace) and ferrosilicon is now being made by a large company with works in Niagara Falls, N. Y., and in Virginia, while another company makes metallic silicon in Niagara Falls, N. Y. Besides these there is a Canadian works for manufacture of ferrosilicon, operating in Welland, Ont., while a company which made ferrosilicon on a smaller scale in Lockport, N. Y., has recently discontinued operation due to the market conditions, which have continued to remain unsatisfactory in spite of the considerably increased duty placed on ferrosilicon by the new tariff. This is significant.

* * *

The simple explanation of the continued unsatisfactory ferrosilicon market is, of course, overproduction. When the

manufacture of high-grade ferrosilicon and other high-grade ferro-alloys in the electric furnace was taken up, it offered good profits. Steel manufacturers were also quick to realize the advantages of the high-grade electric furnace alloy, the use of which (in spite of its higher price compared with low-grade blast-furnace ferrosilicon) not only reduces very essentially the amount of undesired impurities (especially carbon) introduced with the alloy into the steel bath, but also has a higher energy content, heating the bath by the oxidation heat of silicon, instead of chilling it. But after all, the demand for high-grade ferrosilicon, as well as for other electric-furnace ferro-alloys, is a limited one in the nature of things. This is especially true, since the one ferro-alloy for which there is a really large market—ferro-manganese—is made almost exclusively in the blast furnace for good reasons which need no discussion at this place. It is not difficult to understand how overproduction came about with the prices of ferrosilicon practically cut in half. Now most of the French works are operating on a largely reduced scale. Of course, time will straighten out matters. The weakest producers will go to the wall and the fittest will survive. Then the new tariff may exert the effect to be expected.

* * *

While we do not know that any serious accidents have happened in this country on account of dangerous gases evolved from electric-furnace ferrosilicon, such accidents have happened in Europe and have attracted a great deal of attention in England. Concerning the nature of the gases and the possibilities of preventing such danger, the article in this issue contains much interesting information. So far the majority of the French manufacturers have agreed on making only certain grades, which are free from danger, while the British Government is likely to issue regulations on the transport of ferrosilicon. All these developments should also be interesting in this country, both for the producer and the consumer.

Cost of Power.

There still exists considerable confusion in the minds of many engineers as to the cost of developing power, especially water-power. Those who are interested in getting reliable and conservative figures on the basis of best modern American practice may be referred to a paper presented by Mr. Henry L. Doherty before the American Institute of Electrical Engineers on Dec 16, 1909; in the discussion of this paper some very valuable figures on the cost of steam-power and water-power were given both by Mr. Calvert Townley and Dr. Cary T. Hutchinson. It is not our intention to go here into the details of those estimates, but we will simply give the following table of Dr. Hutchinson for the cost of energy in cents per kw-hour for steam and water-power plants, a medium-head hydroelectric generating and transmission plant being assumed in the latter case. These values are based on the assumption of a fixed annual charge of $13.60 per kilowatt, plus an output charge of 0.5 cent per kw-hour for the steam plant, and a fixed charge of $20 per kilowatt of demand on the water-power plant. As shown, at a high value of load factor the hydroelectric plant can show a much larger profit than the steam plant, while the profits decrease as the load factor becomes less. In the last column of the table is shown the price, at various load factors, at which energy from the hydroelectric plant must be sold in order to produce a profit of 6 per cent on the investment.

COST AND SELLING PRICE OF ENERGY PER KW-HOUR AT VARIOUS ANNUAL LOAD FACTORS.

Per cent. Load factor	——Cost of kw-hour in cents—— Steam plant.	Water plant.	Selling price, water power.
15	1.54	1.52	2.44
25	1.12	0.91	1.46
33	0.97	0.69	1.10
40	0.89	0.57	0.92
50	0.81	0.46	0.73
60	0.76	0.38	0.61
75	0.71	0.31	0.49

* * *

The above figures, while apparently high, are undoubtedly correct for the average hydroelectric power and transmission plant. Under special conditions where the hydraulic development is easy and no transmission line is required they could be reduced. This will be true for many electrochemical plants which can be located right near the water-power. And the fact that electrochemical plants have a very high uniform load factor makes them ideal customers of the power-plant. Nevertheless, such prices as are reported from Norway—$4 or $5 per hp-year—can never be expected in this country under present conditions. For within the region over which power can be transmitted and distributed economically from the power plant, enough customers can always be found in this country to which power can be sold with greater profits. It is different in Norway. The question is not whether a power plant could generate power at such and such a very low cost, but whether it could sell it to electrochemical customers at so much less a price than it could get from others to whom the power would have to be transmitted over a distance. The margin between the Norwegian price of $4 or $5 per hp-year and the price which can be obtained from ordinary customers is so large that it would pay for quite an extended transmission and distribution system.

Electrolytic Amalgamation.

The notice of a patent for an electrolytic amalgamation process which appears in this issue under Current Metallurgical and Chemical Patents, gives us an opportunity to comment briefly on the probable rationale of this class of novel processes about which much has been heard in a somewhat general way from various parts of the West during the past few years. However the apparatus and the solution employed may differ, all these processes agree in making the amalgamation plate the cathode of an electrolytic cell; and the claim generally made for such processes is that in this way it is possible to recover on the amalgamated plate "floured gold," "rusted gold," and, in general, gold in a condition in which it cannot be recovered in ordinary amalgamation practice. There are three essentially different reasons why such an arrangement may be effective. In the first place, cathodic reduction, which always goes on at the mercury cathode surface, will reduce any oxides, etc., which may have formed on it; further, since in such processes a mercuric salt is generally used in solution, fresh metallic mercury is constantly being precipitated on the amalgamated plate. In this way the surface is always maintained in best condition, ready to alloy with any particles of free gold which come in contact with it. In this respect nothing is changed in the principle of the ordinary amalgamation process; the current simply keeps the mercury plate automatically in perfect condition

A second reason to which the effectiveness of electrolytic amalgamation may be assigned is suggested by some observations on gold telluride ores made by Dr. W. H. Walker and described by him in an American Electrochemical Society paper in 1903. He noticed that electrolytic methods working on this class of ore recover their values in greater part by direct cathodic reduction than they do by first dissolving the mineral in the electrolyte with subsequent precipitation of the gold on the cathode. Good contact with the cathode is, therefore, an essential requirement; for this provision is made in electrolytic amalgamation processes so that particles of ore of the class mentioned, when coming in contact with the cathode plate, are directly reduced by the current and the reduced gold then alloys with the mercury. A third possible effect which will vary with the nature of the solution used is the solution of gold particles in the electrolyte in form of a gold salt with subsequent precipitation of the gold on the cathode; in this respect the process would be like an electrocyanide or an electric chlorination process. From a practical standpoint the great saving in time, obtainable with electrolytic amalgamation, seems of prime importance. While these processes have attracted the attention of some able metallurgists, they have been developed in other places more or less by the rule of thumb. The general activity along these lines would indicate, however, that some concrete results have been obtained and that a distinct progress of lasting benefit to the metallurgical profession will be the ultimate outcome. Naturally, we should expect the new processes to be used on a commercial scale first for the treatment of such ores which offer special difficulties with present practice.

The Primitive Ideas of Some Connellsville Coke Operators.

At a meeting of the American Association for the Advancement of Science many years ago the late Dr. Samuel P. Langley suggested that while it was comfortable to conceive of the advancement of science as comparable with the progress of a large and well-disciplined army, a more truthful simile was that of a disorganized mob, bent on progress, but with groups constantly making sallies in this or that direction, so that at a given time the general trend of movement was hard to discern, although by observations taken at intervals a distinct forward movement of the whole mass could be noticed. A like simile might be applied to progress in the industrial arts. One invention or discovery waits upon another. Perhaps it is our reference just now to Dr. Langley which puts us in mind of flying machines; at any rate, we may observe that 20 years ago he clearly demonstrated the feasibility of mechanical flight, and, indeed, Lilienthal accomplished many "glides" before he met his death in 1889. Yet it was not until recently that perfect mechanical flight was accomplished. The success was really due to the development of a suitable internal combustion engine. Indeed, it is interesting to recall that one of Langley's troubles at one time in conducting his whirling table experiments was that the gas engine erected to drive it would not go.

* * *

Our mention of gas engines brings us to our subject. Twenty years ago efforts were made to introduce by-product coke-making into the United States, but these efforts were not successful. The value of the solid and liquid by-products was recognized, but it was doubted whether those values would be maintained in the face of heavy production. It was not admitted that the gas would have a great value. Naturally, it was not regarded as practicable to pipe it long distances to centers of population, when the much richer natural gas was available for the same purpose. The advent of the large gas engine, together with the means for economically converting the energy it produces into electrical energy, has multiplied by many times the commercial value of gas from by-product coking. The by-product coke oven has really awaited the perfection of means to make useful the gas which it was practically as well able to produce 20 years ago as it is to-day. Unlike the case of tar and ammonia, there is an unlimited market for gas for converting its energy into electrical energy.

* * *

In the past six months the coke operators of the Connellsville region have confessed an absolute ignorance of the developments in retort coking and the use of the by-products. The time was that Connellsville coke was regarded as the only really good blast-furnace fuel. Obsessed with the idea that perforce a much larger quantity of Connellsville coke would be demanded by the iron industry in 1910 than ever before, Connellsville coke operators last September and October pushed the value of their furnace coke to $3 a net ton at ovens, and then waited for the rush of customers. Through the growth of by-product and beehive coking in other fields the proportion of the total blast-furnace trade of the North, which depends upon Connellsville coke, has been reduced, and while the Connellsville operators waited for buyers to step to the counter, provision was made to use other cokes. There has been a tumble in Connellsville coke prices since the middle of January, and the industry now finds itself in the position of having much unsold production, but with no consumers, since their old ones are apparently provided with one coke or another.

* * *

In May of last year Connellsville furnace coke sold on contract for second half delivery at about $1.60 per net ton at ovens. There is no reason to suppose the price was unprofitable. An inspection of cost sheets indicates that at an average operation the cost of production, before the slight advance in wages in January of this year, was about $1.25 per ton, without depreciation and other fixed charges. The pushing of the price to $3 merely on the supposition that blast furnaces would have to pay the price or not make pig iron was a command to the industry to seek other fuel. When another fuel has been found, the consumer does not return to Connellsville. The customer is lost. Connellsville coke operators are anxious now to sell at much lower prices than those demanded late last year, but the business is gone. Report has it that they are now seeking to establish a "central selling agency" so as to maintain uniformly a high price for their coke. When they sell their coke at their own bare cost of production they are selling it for a greater price than it is worth, for they have produced it in the most wasteful and extravagant manner. They have failed utterly to realize that the day of by-product coking is fully arrived. They are exerting a tremendous pressure toward the adoption of modern methods, but they are killing their own business and writing an enormous depreciation upon their coal reserves.

Eighth International Congress of Applied Chemistry.

On the evening of Feb. 3, 1910, an informal gathering took place at which there were present among others most of those to whom had been delegated the task of providing for the creation of an organization for the Eighth International Congress of Applied Chemistry, by the London Congress last June; a representative of the Association of Manufacturing Chemists, the American Chemical Society, the American Electrochemical Society, and the Society of Chemical Industry were each also present at this informal meeting.

The consensus of opinion was that the greatest success could be expected only if the most effective system and mode of organization could be had, and if each and every chemist in the United States could be made to feel that he himself directly or indirectly through his professional, business or educational affiliations, had a personal share of responsibility in the conduct and management of the Congress from its very start and to its very end.

This was regarded as the proper and correct guide in proposing any plans or schemes of organization.

The Eighth Congress is to convene in 1912, with Prof. Edward W. Morley as honorary president and Dr. W. H. Nichols as acting president, at a time and place to be determined by the organization of this Congress. The most important part of the Congress, in fact that by which its value and real success will be measured, is the amount of original matter, both scientific and technical, which it will be able to present to its members.

To attain this end fully every chemist in America who has or may have any original matter to present to this Congress should begin without delay to prepare such matter, and have it in shape so that it may be presented to the Congress in ample time for proper printing, classification, and distribution to members and the technical and scientific press.

The Congress, being held in the United States, will with a great deal of right and naturally look to a very good showing from the chemists of the United States, and every chemist in this country, which is to be host to our foreign colleagues, should constitute himself a committee of one to get from himself, or from his friends, as much scientifically or technically valuable material as possible so that the proceedings and publications of the Eighth Congress may correctly reflect the true mental attitude of the chemists of the United States toward their profession, both as a pure science and as a part of the industrial activities of this country.

It is the hope that the Program Committee will be able to begin its activities effectively before the close of 1910, but in the meantime it behooves every chemist in the United States actively and energetically to consider how and in what way he can best contribute to the success of this Congress, and particularly in the direction of papers and communications to the Congress embodying the advance in this field since June, 1909, the date of the last Congress.

At a meeting to be held in April or May, 1910, by those charged with the duty of providing suitable organization for the Eighth Congress, some definite action as to such organization may be looked for. Those who have that responsibility are making every effort to get as many suggestions as to divisions of organization, mode or organizing, and membership of the organization as possible. Every one interested in having this organization on as broad foundation as possible is earnestly invited to present any suggestions that may be helpful in that direction in writing by the middle of April, 1910, so that all these suggestions may be properly classified and collated and put in condition for most thorough consideration before the meeting above referred to actually takes place.

Such communications may be addressed to the temporary secretary, Dr. B. C. Hesse, 90 William Street, New York City

Bureau of Standards' Analyzed Samples.

The Bureau of Standards, Washington, D. C., is ready to distribute Bessemer steels with 0.6 and 0.8 carbon and an iron C to replace that originally prepared by the American Foundrymen's Association. The new C has the following composition: Total C, 2.78; graphite, 2.22; Si, 1.84; Ti, 0.074; P, 0.192; S by oxid, 0.0354; S by evol., 0.0335; Mn, 0.744.

A new iron B was found to need remixing and partial analysis. It is hoped that the sample will be ready to distribute by the time this notice appears in print.

A vanadium steel and an ore of manganese await the reconciling of differences in the analytical results reported.

Three Lake Superior iron ores are undergoing analysis; one to serve as a standard for iron, phosphorus and silicon; the second for aluminium, calcium and magnesium; and the third for manganese.

Standardization of Bullion and Assay Values.

The Sectional Committee B for the standardization of bullion and assay values of the Institution of Mining and Metallurgy was formed for the purpose of recommending rules which shall, as far as possible, be followed by assayers, metallurgists and mining men in reporting the results of assays and of metallurgical, dressing and other tests of which assays form an essential part, so that ambiguity shall be avoided and the possibility of misinterpretation or misuse of reports shall be minimized. The committee has now submitted to the council of the Institution the following report, which is now to be considered, before being finally adopted.

Reports of assay results, extraction tests, metallurgical and ore-dressing trials, etc., are sometimes so worded as to permit a wrong interpretation, and the committee urges the necessity for definite statements as to the nature and condition of the sample as assayed, and, where advisable, an indication of the method of assay. Thus, in the case of such metals as copper, tin, lead, antimony, etc., where both "wet" and "dry" methods of assay are in use, results should be reported as obtained by "fire" asay or by "wet" assay, with further details as to method, if adjudged advisable. Similarly, it should be stated whether the sample was assayed in the condition in which it was received, or "air-dried" or dried at —° C.

In reporting that an assay for such a constituent as tin, arsenic, tungstic acid, etc., corresponds to so much "black tin," "white arsenic," "wolfram" or other recognized saleable product, the report should give the assay result for such constituent, together with a definite statement as to the figures used in calculating such saleable product. Thus, an assay showing 1 per cent of metallic tin might be reported as 1 per cent of metallic tin, equivalent, if all could be extracted as such, to 1.5 per cent of "black tin" containing 66.6 per cent (or two-thirds) of its weight of metallic tin. Similarly, in reporting results, where assay values might be mistaken as representing available produce, it should be stated that such are merely "gross" assay values, a statement or estimate of the actual "commercial" values being added whenever possible.

In such matters as ore-dressing, ore-treating processes, furnace work, etc., where so much depends on the way in which results are reported, on the nature of the sample and on the method of taking it and its relation to other samples, the committee emphasizes the necessity for careful phraseology and for the inclusion of exact data, as also the necessity for care in preparing, describing, labeling, sealing and otherwise ensuring the correctness and authenticity of samples submitted for tests or assays.

The bullion and assay values committee submit the following recommendations for adoption by the Institution:

(1) Assay reports shall state the exact condition of the sample as to dryness when assayed.

(2) Assay values of gold and silver ores and products shall

be represented in pennyweights and decimals or in ounces and decimals, and not in oz., dwts. and grains. They shall be expressed in terms of fine gold and fine silver, respectively, not as "bullion."

(3) Assay values of alluvials shall be reported in grains and decimals of a grain of "fine"gold, or in pence (at 2d. per grain of "fine" gold), or cents per cubic yard.

It is recommended that, in the absence of specific information, 1 cu. yd. of ordinary alluvial, excluding boulders, be taken as equivalent to 3000 lb. (1½ short tons).

(4) In reporting assay values of cyanide and other solutions, the results shall be given in parts by weight in a stated volume of the solution. (In the case of cyanide solutions, the use of the "fluid ton of 32 cu. ft." is recommended. It closely approximates to 2000 lb. and is in common use.)

(5) When it is necessary to state or estimate the money value of an ore, etc. (other than of gold), it shall be accompanied by the assay value, and the basis on which the former has been calculated from the latter shall be stated.

(6) Laboratory sieving tests shall be made with the I.M.M. standard sieves, or, when other sieves are used, the widths of the apertures shall be stated.

The report is signed by G. T. Holloway as chairman of the committee.

With the above recommendations, if approved, will be included the following, which have already been adopted on the recommendation of the weights and measures sectional committee:

(*a*) The word "ton" shall represent a weight of 2000 lb. avoirdupois (2,166.6 oz. troy). (It is advisable to abandon the use of the terms hundredweights and quarters, and to express fractions of a ton in pounds or in decimals of a ton.)

(*b*) The word "gallon" shall represent the Imperial gallon measure of 10 lb. of water.

(*c*) Temperatures shall be expressed in degrees Centigrade.

(*d*) Returns of gold and silver shall be expressed in terms of fine gold and fine silver, respectively, not as "bullion."

(*e*) Gold contents of ores, etc., determined by assay, shall be expressed in money values as well as in weights; and in this connection the value shall be taken (as a convenient constant) at 85 shillings, or $20.67 United States currency, per troy ounce of fine gold.

The Iron and Steel Market.

February has been a dull month in the market, demand being light all along the line, while prices have definitely receded on pig iron and have shown weakness on a few finished steel products. The market failed to bear out any of the prognostications. In December, when new buying was light by comparison with the six preceding months, opinion was divided as to whether the market would show a decided improvement early in the new year or would merely hold its own. It has done neither; it has lost ground, and has lost more in February than in January.

The American iron trade suffers from a delusion that the hanging of new calendars upon the walls of business offices is an incentive to the placing of orders. Time and again it has staked its all upon the improved business which "the new year" would bring, and as often it has been disappointed. In the late months of 1908 there was a well-defined boom and in all quarters the belief was expressed that this small boom was but the precursor to the big one which the new year would bring, but the new year brought a crash in prices as well. Even when the remarkable period of activity of 1905-06-07 was being ushered in there occurred the delusion of the calendar, for the improvement which occurred in the late months of 1904 was regarded as but the forerunner of a still greater improvement to occur early in 1905, whereas pig iron lost ground almost steadily during the first seven months of that year and steel products would probably have done the same had they not been well held by a trade understanding.

The feature of the trade revival in the second half of 1909 was the uneven distribution of the improvement; merchant steel bars constituted the market leader, carrying with them plates and shapes; wire products did well, and so did sheets and tin plates, while merchant pipe was heavy, the buying being largely because of the low price, jobbers buying for fear there would be an advance and mills fearing to advance prices lest buying should be discouraged. Rails showed almost no movement.

Students of the psychology of market movements may be interested in the influences produced by the different finished steel products being more or less completely under the influence of different systems of selling developed by the respective subsidiaries of the United States Steel Corporation. There are temperamental differences between the managements of the different subsidiaries. The common ground upon which they meet is that of their reports to headquarters of results achieved in tonnage booked. It is not impossible that in the efforts of given subsidiaries to show results comparing favorably with those of other subsidiaries the importance of immediate results is magnified and the importance of providing for the future is minimized. Without attempting to pursue the query at this time, it may be remarked that the result of the operations of the last seven or eight months of 1909 was to leave the different branches of the finished steel trade with orders booked for quite varying distances ahead. Steel bars were sold for nearly six months ahead, plates and shapes for perhaps four months ahead, and pipe and wire for scarcely any distance at all ahead, while jobbers in these two latter lines were more than comfortably stocked. Both by the necessity of physical conditions, since these products are all made from the same semifinished material, and by the temperament of purchasers, who regard the iron and steel trade as an entity and not as a chance group of unrelated elements, the strength of the market as a whole is largely in relation to the strength of its weakest element. Consumers of steel bars are not likely to be enthusiastic buyers for far forward delivery when they observe that prices of wire products are being cut and production is perhaps being curtailed.

Pig Iron.

The spectacular buying of Bessemer pig iron from June to October of last year advanced the market rapidly from $14.50, valley, to $19, a price which was first done Oct. 12, there being altogether more than 100,000 tons sold at that figure. It is an open secret that the large independent steel interests buy Bessemer iron at such times partly because they really expect to need it, partly because they may need it, and partly because such purchases help to improve the market for their finished products. Many contracts for billets and finished steel products are on a scale basis, the price determined by the average selling price of Bessemer pig iron.

It was expected that Bessemer pig iron would draw other grades of pig iron along in the advance, but basic pig iron halted at $17.25, leaving a spread of $1.75 between the grades. In 1905-06 the average spread was about 25 cents and in 1907-08, when Bessemer ores were 25 cents per ton higher relative to non-Bessemer ores, than in the preceding years, the average was about 75 cents. Expectations that basic pig would advance to reduce the spread were disappointed, for in November basic receded to $17 and in December a slight further recession occurred. Toward the close of January it developed that basic could be bought at $16.50, leaving the position of Bessemer at $19 altogether untenable.

There was danger of a break and to crystallize sentiment into a moderate decline the United States Steel Corporation, early in February, bought 40,000 tons at $18, valley, leaving the trade under the impression that upon occasion further purchases might be made. The Bessemer market rests at this. Basic, openly offered at $16.50, valley, has been quietly offered

at $16, and two or three forced sales have been made at slightly lower prices, so that the situation is again involved. In foundry pig iron there have been recessions in nearly all districts.

Virginia furnaces came down to $15 at the furnace for No. 2 plain, silicon 2.00 to 2.50 per cent, and 2500 tons of Southern iron were sold on the basis of $13, Birmingham. In eastern Pennsylvania there were slight recessions. The valley market dropped to $16.25. The total decline from the top point in November has been approximately $2 at Birmingham, $1 in Virginia, 50 cents in eastern Pennsylvania, and $1 in the valley market.

Steel.

The supply of unfinished steel has increased in all districts. Chicago producers have become sellers of forging and rolling billets, having been out of the market for several months. In Pittsburgh and the Central West deliveries of billets and sheet bars have become very good, to such an extent that some consumers have requested shipments postponed. Prices have not openly yielded, but it is improbable that any large tonnages could be sold except at material concessions. There is no demand, consumers being amply covered by contracts. Nominal prices are approximately as follows, f.o.b. maker's mill, Pittsburgh or Youngstown:

Bessemer billets, $27 to $27.50; open-hearth billets, $27.50 to $28; sheet bars, $28.50 to $29; rods, $33.

Strictly new buying of finished steel products has been light. Specifications on old contracts have been fairly good. The steel-car shops are running practically full, booked for four to six months. The fabricating shops are about equally well filled.

The plate and shape mills in the Pittsburgh and in the Chicago districts are comfortable taken care of until about June 1; the Eastern mills have already begun to make concessions of $1 a ton. Demand for ordinary sheets has become light, while specialties are still in demand, particularly electrical and stamping sheets.

Some sheet mills which were extremely optimistic last fall and book little business have not much work in sight, while others are comfortably filled.

Early in January steel bars were quoted by all interests at $1.50, base, the December market having been $1.45 on contract and $1.50 for small, prompt lots. The former position has been resumed, the market being $1.45 on contracts, but it is a position whether this constitutes an actual recession, the minimum of $1.50 not having been absolutely established.

Regular prices, at Pittsburgh, except where otherwise noted, stand:

Standard steel rails, $28 for Bessemer; $30 for open-hearth, f.o.b. mill except Colorado.

Plates, $1.55 for tank quality, ¼ in. and heavier.

Shapes, $1.55 for I-beams and channels, 15 in. and under, zees and angles, 2 x 3 and larger.

Steel bars, $1.45 to $1.50, base.

Iron bars, $1.65 to $1.70, Pittsburgh; $1.55 to $1.60, Philadelphia; $1.55 to $1.60, Chicago; $1.50 to $1.55, St. Louis.

Wire nails, $1.85, base; plain wire, $1.65; galvanized barb wire, $2.15.

Black sheets, 28 gage, $2.40; galvanized, $3.50; blue annealed, 10 gage, $1.75 to $1.85; painted corrugated roofing, $1.70 per square; galvanized, $3.

Tin plates, $3.60 for 100 lb. cokes

CORRESPONDENCE

Iron-Copper Alloys.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—We note with interest Mr George W. Sargent's comments on the paper "Physical Properties of the Iron Copper Alloys." We are realizing full well the limitations of data obtained from experiments conducted on a laboratory scale and under special conditions of working, and it is, therefore, of interest and of advantage to have the opinions of practicing metallurgists regarding the probability of repetition of interesting results from operations conducted on an industrial scale.

We can appreciate Mr. Sargent's statement concerning a lack of harmony between the preface of our paper and the actual work done, and that the two together give an erroneous impression. This is caused, no doubt, by the fact that the article in *Electrochemical and Metallurgical Industry* was only an abstract of the paper presented at the meeting of the American Electrochemical Society. While the report of our experimental work was reprinted practically in full, the introductory matter was necessarily very much condensed.

Our paper points out that the first impressions regarding the presence of copper in steels were that it was decidedly deleterious; that later work showed that small amounts did not have the pronounced detrimental effects usually attributed to copper.

Finally, we quote the experimental data of other investigators, and corroborating our results, tending to the conclusions that additions of copper up to 4 per cent were accompanied by material increase in the tensile strength of the steel.

Mr. Sargent believes that red shortness would be the general result of copper additions, especially in the familiar steel-making processes, due to the oxidizing reactions and the consequent formation of oxides of copper. We do not believe that this contention is supported by chemical facts or metallurgical experience.

These facts are:

Iron will oxidize to ferrous oxide more readily than copper will to cuprous oxide.

Either iron or copper oxide will be reduced by carbon, manganese, aluminium, or silicon, the common deoxidizing agents of steel-making.

Consequently, we should expect that if oxidation of the bath were carried to the extent of formation of dissolved oxides, ferrous oxide would be the first to form, and brittleness would result whether copper were present or not. Again, it is to be expected that the customary additions would have an even greater effect in reducing any copper oxides than they now have on the oxides of iron.

Experience in the metallurgy of copper supports this view. As reverberatory refining of copper is conducted, the iron is oxidized and scorified with the additions of sand. To quote from Schnabel's "Handbook of Metallurgy," Vol 1, page 237: "The separation of these elements (impurities, including iron, lead, zinc, etc.) can be brought about by oxidizing fusion of the coarse copper." "By oxidizing fusion of coarse copper, its foreign constituents (with the exception of the precious metals) can be oxidized (because their affinity for oxygen is greater than that of copper) and slagged off."

It seems likely, as was intimated in our paper, that the unfavorable influences are the effect of increase of carbon on the structural relations of copper and iron, and the affinity of copper for any sulphur which might be present, with the formation of sulphide of copper and its incorporation in the body of the steel.

Chemical Engineering Laboratories, James Aston.
University of Wisconsin

The Patent Treaty With Germany.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—Enclosed please find a translation of a very important recent decision* of the German Reichsgericht (Supreme Court). It refers to the recently concluded patent treaty between the United States and Germany

*An abstract of this is given elsewhere in this issue, in the article entitled "A Reichsgericht Decision on the Scope of the Patent Treaty between the United States and Germany."—Ed.

The Imperial Court interprets this patent treaty in a most liberal way and comes to a conclusion which is causing general indignation among industrial circles in Germany.

The patent treaty in question was concluded because the Committee on Patents of the House of Representatives threatened to introduce a working clause similar to the one of the British and German law into the American Patent law. In order to free the citizens of both countries from the burden of such a working clause the treaty was concluded.

Now, this working clause has not been introduced into the United States Patent law, and there does not exist any reason for such a patent treaty between the countries.

Meanwhile the treaty is exerting its influence in litigations under the German patent law to the greatest detriment—as many interested parties claim—of the industries of the German Empire, and there is a strong agitation to induce the Government to cancel the treaty.

It is pointed out that Great Britain, which peremptorily refused to conclude a similar treaty with the United States, has now the laugh on Germany.

To refer back to the decision, the court decrees that a German owning a German patent is forced to work his invention either in Germany or the United States, while an American owning a German patent needs not work his invention either in Germany or the United States because the country of his nationality does not enforce the working clause.

Consequently, a German owning patents may avoid the working clause by assigning his patent to an American.

MANUFACTURER.

Silundum.

To the Editar of Metallurgical and Chemical Engineering:

SIR:—Professor Tucker's excellent paper on "Silundum," which is in full accordance with the experience obtained in the commercial production of said material, has caused a controversy between Messrs. E. G. Acheson and F Bölling, as well as your editorial remarks in your last issue.

Leaving aside the question of the priority I first only beg to state one fact which possibly might elucidate the matter, and secondly call your attention to the rôle of an element which hitherto has been overlooked by most of the experimenters though often contained in carbon-silicon compounds:

(1) Dr. Acheson says that the heating of carbon, silicon and oxygen may either result in the formation of graphite, silicon or carborundum, or of a compound of all the three elements. No doubt about this. But there are only a relatively small number of analyses existing of siloxicon, and so far it does not seem to be a definite chemical individual. We find in many other cases, too, that a practically useful material such as siloxicon need not possess a uniform chemical composition. There is no reason why it should not be possible to combine the first two elements also in such a way as to form a product of variable chemical formulas. On the contrary, the presence of dissolved carbon and silicon in many metals makes it probable that an alloy of these two elements exists, and that is what silundum seems to be and what further scientific investigation is expected to affirm. It may have spontaneously been formed in some furnaces long ago, yet it had not been discovered.

(2) Among the earliest literature about the reactions of carbon and silicon at high temperatures we find compounds of these elements with nitrogen described. As the coke employed in the carborundum furnace contains nitrogen, there is no doubt that this element, preferably reacting at high temperatures, influences the product of the furnace in a certain way; about the character of this influence we know even less than about that on iron and steel. Nitrogen is found in carborundum as well as in silundum, as stated on another occasion (*Zeitschrift für Elektrochemie*, 1909, p. 727), and I regret that industrial work did not leave me the time to pay the necessary attention to this question.* But if once the binary silicon-carbon diagram will have been devised the further evolution of the Acheson diagram must necessarily take notice of the nitrogen as well as of the oxygen.

CHICAGO, ILL. R. AMBERG.

*The so-called oxygen contents of siloxicon is mostly given as the difference between the total and the silicon and carbon, analytically determined.

The Trust from the Standpoint of an Engineer.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—Your journal has on its editorial page alluded so often to the trust question, both in its general aspects and particularly to the metallurgical trusts, that possibly my views will interest your readers, for I have had business relations with several of the large trusts, and thus have first-hand knowledge. Lately I have read a very remarkable exposition of the economics of the trust question by the Hon. Bourke Cockran in his brief for the American Tobacco Company before the Supreme Court of the United States. This elucidates the theory of monopolies from the standpoint of English common law. My reasoning of the case is along quite different lines, but the end product is in each case the same. Let me say that I hold no brief for the American Tobacco Company, as I usually insist on smoking non-trust cigars.

In the first place, let it be stated that by a trust is meant a large corporation seeking to dominate the market for any one commodity and actually possessing preponderant trade influence in that market. This is the usual definition.

Now the trust is but a natural outgrowth of industrial evolution. We know that evolution is that process by which things grow through countless small processes of differentiations and integrations from the tiny homogeneous units to the large heterogeneous unit. The great contribution of the nineteenth century to the philosophy of the ages was the formulation by Darwin of that doctrine in the biological world and its application by Herbert Spencer to all branches of science. So it is just natural to point to the United States Steel Corporation for instance, as the product of a perfectly natural industrial evolulutionary process. Demagogues may rail at it, at Mr. Carnegie and at Mr. Morgan. But facts are facts. We see the United States Steel Corporation pursuing the even tenor of its way (not, of course, the market value of its stock) and we can only admit that it is a product of the age.

As no one but a simpleton will deny the above argument that the "trusts" are the logical outcome of the natural evolution of industrialism, let us inquire why it should exist in the light of the evolutionary theory. Naturally, it exists because the fittest survive. This is simply tautology. So let us ask why it should survive, or, better, why one trust survives and another goes to the wall. First of all, it can be stated that it exists because the weight of public opinion is on its side. For does anyone suppose that if the mass of people in this country believe all the things said about the Standard Oil Company that it would survive a day? I doubt whether legislative enactments would be used at all, but it would seem probable that physical violence would be rightly used. No! the average of public opinion is that the Standard Oil Company, though far from perfect, is, on the whole, serving a beneficent purpose in the United States. It undoubtedly took rebates, but, you could sweep this wide country and if you can show me 100 small country cross-road grocers that would refuse to accept a reduction of 50 cents per barrel of flour if they handled exclusively one brand, I misjudge human nature. It is just human nature to trade with your friends. And rebating, so common once, is similar to the practice of giving a special favor to your friends.

Now, the trust stands in spite of adverse public opinion of the discontented minority. Why does it sell its goods to this American people, who, though easily fooled for a short time, do

not stay fooled long? Obviously because on the average it produces and sells its goods on quality and price. Against this it may be argued that the Steel Corporation is charging a high price for rails and is making large profits at prices higher than prevalent in Europe. But, without the evidence, let me assume as a fact that steel rails can be produced at a factory cost of $15. This figure is, to the best of my knowledge, about correct. Why, then, do not outside companies step in and erect large plants if profit is so large? There is a multiplicity of causes as in any phenomenon. But the chief reason, in my opinion is that the risk of the business, the value to the community of ore in the ground and other charges reduce the net return on the investment to about 15 per cent. Now, there are so many kinds of smaller manufacturing companies doing a business of from $100,000 to $1,000,000 per year in this country at a profit of greater than 15 per cent that there is not present in the United States the kind of capital that cares to go into the rail business when, for example, the automobile business exists.

If the American people want rails sold at less than $28 per ton, let it subject, by removing the tariff, the rail manufacturers to the competition of Germany, where manufacturing profits are much less.

It can be laid down as a fact that in this country, therefore, there is always plenty of competition if the profits are large enough. As Mr. Cockran well says, "competition may be potential or active."

Now, any trust may fail (and great will be the fall thereof), provided it is managed in an non-progressive, unintelligent and crassly selfish manner. And conversely, a trust will succeed provided it is managed in an up-to-date, intelligent manner and in a spirit of enlightened self-interest. And, furthermore, its degree of success depends upon the measure of intelligence of its management. The same is true of a railroad or any concern of any kind at all. Efficiency spells success. Somehow or other, intelligence seems to win in this world in the long run. In short, intelligently managed concerns, large or small succeed. If the trust makes mistakes, its mistakes are just so much the more costly as the trust is big. And some one of the independents, the one who uses the best intelligently directed head work, will take the first place in the hegemony.

It can be laid down as a fact that purely economic laws govern the rise and period of life and degree of usefulness of these large corporations. Purely economic laws govern their action. And the laws of economics grind slowly, but they grind exceeding fine. This is what history teaches from the days of Adam Smith down.

Comparisons are invidious, and it would be idle for me to draw comparisons to any great extent and, besides, a breach of confidence. But it is a matter of common knowledge that the United States Steel Corporation, by a policy of persuasion rather than of coercion, in 1906 and the first part of 1907 did restrain steel prices from going to extravagant high levels, and in the last three months of 1907 did hold prices from going to the everlasting bow-wows. In 1908 the prices were allowed to subside gradually, but the retreat was an orderly one, so that to-day of all the metal markets the iron and steel market is alone in a healthy and stable condition

The copper market in 1907 was a runaway one Consumers were in a panic and lake copper sold as high as 27 cents. This was due, first of all, to natural causes, such as shortage of coal and labor in the mining centers, but, second, to a more or less concerted action on the part of the producers by exaggerating actual conditions So the copper producers danced and have been paying the piper for the past year. Undoubtedly there was no actual conspiracy on the part of the producers, but action may be concerted, though entirely tacit and even unconscious

But the steel business was managed intelligently for the interests of the entire business and the copper business was handled in a spirit of narrow selfishness We see that this was a boomerang

And if the Supreme Court of the United States construes the Sherman law so that the policy of keeping prices stable and reasonable is a crime, my great respect for that body will be lessened.

It is thus seen that the policy of a trust should be to conserve the best interests of the trade and that its own interests are so best served. Stable markets are better for the consumer than unstable markets, even when the average price of the latter is somewhat lower than the average price of the former.

The trusts can act efficiently in other ways. For example, unprofitable plants can be shut down. The highest salaries can be paid to business and technical talent. General expenses or the amount paid for intelligence and diffusing it, can be large absolutely and still be a small percentage of the total cost of production. Purchases can be made in large volume, and hence low. Plants can be remodeled and new processes put in, as for example, the Héroult electric steel furnace by the Steel Corporation, and undoubtedly this is the line where commercial inertia inhibits progression. Laborers can be given steady employment, and hence are much more efficient Profit-sharing makes them partners in the concern. Money can be borrowed much more cheaply and in large amounts.

Two large trusts are typical and illustrative by reason of the peculiar nature of their business of the general excellence that trusts can have. These are Swift & Company, "the meat trust," and the United Fruit Company, the "banana trust." The former does a business of $300,000,000, in round figures, per year on a 3 per cent margin. The latter makes a marginal profit of 10 cents per bunch containing over 150 bananas. Let any business man deny that these margins are excessive. The retail margin of the small butcher and Italian who peddles bananas is from 6 to 10 times the margin of the large wholesaler. Only on one assumption can these two concerns be harmful to the body economic, and that is that the officials are stealing flagrantly from the stockholders and falsifying the books. The high personal standard of honor among the directors of these two companies precludes this idea and dismisses the assumption as absurd.

For my own part I cannot but admit that (1) the badly managed trust goes to the wall quickly; (2) the well-managed trust helps everyone but those who have become suddenly rich by its incorporation and spend all their money and all their time in riotous living. For the idle and inefficient, who lose their jobs or their property by its competition, have at least the solace of kicking, and in their own minds attributing their misfortunes to the machinations of others. They are, indeed, then happy in their misery.

Finally, let me state that Mr. Cockran's argument before the Supreme Court is that any company which has increased its production and kept its prices from advancing while general prices have advanced is not restricting trade. For increase of production is increase of trade. Nor is restriction of intelligent competition restriction of trade. In short, restraint of trade is one thing and restrain of competition is another thing. Mr Cockran further points out that if trade is restrained by conspiracy or acts of violence it is certainly against the common law of our country and England, and, hence, punishable by fine or imprisonment.

In short, Judge Parker was right in 1904 when he said that the trust question could be solved by the application of common law Possibly ex-President Roosevelt was also right in his solemn affirmation that a good trust is good for the country and that a bad trust is bad for the country. But in the modern schools of psychology the theory of value is prevalent, and to evaluate the good in men, women and things is an eternal problem. It were far better for the American people to stick to the common law, not to try to legislate morals into themselves, and ever to seek to evaluate the things as they see them in the light of things as they are

Hartford Conn. Woolsey McA. Johnson

Temperature Conversion Table.

Through the courtesy of Dr. Leonard Waldo, consulting engineer, of 49 Wall Street, New York City, we are enabled to reproduce herewith his very convenient Centigrade and Fahrenheit temperature conversion table.

Dr. Waldo's table is self-explanatory. It will be found to be based on an arrangement well known from logarithmic tables, but novel for the conversion of temperatures. Besides convenience it has the advantage of exactness, i. e., the values of temperature found from the table are exact, not approximate.

C°	0	10	20	30	40	50	60	70	80	90
	F	F	F	F	F	F	F	F	F	F
—200	—328	—346	—363	—382	—400	—418	—436	—454		
—100	—148	—166	—184	—202	—220	—238	—256	—274	—292	—310
—0	—32	—14	—4	—22	—40	—58	—76	—94	—112	—130
0	32	50	68	86	104	122	140	158	176	194
100	212	230	248	266	284	302	320	338	356	374
200	392	410	428	446	464	482	500	518	536	554
300	572	590	608	626	644	662	680	698	716	734
400	752	770	788	806	824	842	860	878	896	914
500	932	950	968	986	1004	1022	1040	1058	1076	1094
600	1112	1130	1148	1166	1184	1202	1220	1238	1256	1274
700	1292	1310	1328	1346	1364	1382	1400	1418	1436	1454
800	1472	1490	1508	1526	1544	1562	1580	1598	1616	1634
900	1652	1670	1688	1706	1724	1742	1760	1778	1796	1814
1000	1832	1850	1868	1886	1904	1922	1940	1958	1976	1994
1100	2012	2030	2048	2066	2084	2102	2120	2138	2156	2174
1200	2192	2210	2228	2246	2264	2282	2300	2318	2336	2354
1300	2372	2390	2408	2426	2444	2462	2480	2498	2516	2534
1400	2552	2570	2588	2606	2624	2642	2660	2678	2696	2714
1500	2732	2750	2768	2786	2804	2822	2840	2858	2876	2894
1600	2912	2930	2948	2966	2984	3002	3020	3038	3056	3074
1700	3092	3110	3128	3146	3164	3182	3200	3218	3236	3254
1500	3272	3290	3308	3326	3344	3362	3380	3398	3416	3434
1900	3452	3470	3488	3506	3524	3542	3560	3578	3596	3614
2000	3632	3650	3668	3686	3704	3722	3740	3758	3776	3794
2100	3812	3830	3848	3866	3884	3902	3920	3938	3956	3974
2200	3992	4010	4028	4046	4064	4082	4100	4118	4136	4154
2300	4172	4190	4208	4226	4244	4262	4280	4298	4316	4334
2400	4352	4370	4388	4406	4424	4442	4460	4478	4496	4514
2500	4532	4550	4568	4586	4604	4622	4640	4658	4676	4694
2600	4712	4730	4748	4766	4784	4802	4820	4838	4856	4874
2700	4892	4910	4928	4946	4964	4982	5000	5018	5036	5054
2500	5072	5090	5108	5126	5144	5162	5180	5198	5216	5234
2900	5252	5270	5288	5306	5324	5342	5360	5378	5396	5414
3000	5432	5450	5468	5486	5504	5522	5540	5558	5576	5594
3100	5612	5630	5648	5666	5684	5702	5720	5738	5756	5774
3200	5792	5810	5828	5846	5864	5882	5900	5918	5936	5954
3300	5972	5990	6008	6026	6044	6062	6080	6098	6116	6134
3100	6152	6170	6188	6206	6224	6242	6260	6278	6296	6314
3500	6332	6350	6368	6386	6404	6422	6440	6458	6476	6494
3600	6512	6530	6548	6566	6584	6602	6620	6638	6656	6674
3700	6692	6710	6728	6746	6764	6782	6800	6818	6836	6854
3500	6872	6890	6908	6926	6944	6962	6980	6998	7016	7034
3900	7052	7070	7088	7106	7124	7142	7160	7178	7196	7214
C°	0	10	20	30	40	50	60	70	80	90

C°	F°
1	1.8
2	3.6
3	5.4
4	7.2
5	9.0
6	10.8
7	12.6
8	14.4
9	16.2
10	18.0

F°	C°
1	.56
2	1.11
3	1.67
4	2.22
5	2.78
6	3.33
7	3.89
8	4.44
9	5.00
10	5.56
11	6.11
12	6.67
13	7.22
14	7.78
15	8.33
16	8.89
17	9.44
18	10.00

Examples: 1347°C = 2444°F+12°.6F = 2456°.6F: 3367°F = 1850°C+2°.78C 1852°.78C

The table covers the whole temperature range from the temperature of liquid air to that of the arc.

Advantages of Enriched Air Blast.

By Jos. W. Richards.

If we consider what effect air enriched in oxygen can have upon the working of a blast furnace, the first and immediate effect will undoubtedly be the increase in maximum available temperature before the tuyeres. What that will be can be calculated with exactness for air of any given percentage of oxygen.

The next question is, what effect this increased temperature will have upon the running of the furnace. Here calculation cannot be made with exactness, and only general observations can be made as to the effects which these high temperatures will produce.

The normal percentage of oxygen in pure air by volume is 20.8, nitrogen (argon, etc.) 79.2. For the purpose of discussing the effect of enriching the air in oxygen, upon the working of a blast furnace, we can leave out of consideration the presence of water vapor in the blast, or of the small amount of CO^2 in the air, and confine our attention to the oxygen and accompanying nitrogen.

Using cold pure air in the blast furnace, the combustion of the carbon being practically entirely to carbon monoxide, CO, the theoretical maximum combustion temperature attained is 1678° C. (Metallurgical Calculations, Part II, p. 282). If the air is preheated to 1000° C., the theoretical maximum is 2520° C., an increase of 842° C., or 84° for every 100° heat in the hot blast.

The function of hot blast is to increase the total amount of heat available per unit of carbon burnt before the tuyeres. This increase in available calories results in raising the temperature of the furnace gases in the smelting zone without changing their quantity, and thus renders available for smelting down purposes a larger quantity of heat units per unit of carbon burnt, at a *higher* initial combustion temperature. The net results are that the unit of carbon burnt before the tuyeres smelts down more iron and slag and smelts quicker, because of the higher initial combustion temperature. This produces less CO before the tuyeres per unit of iron smelted, resulting in a better production of CO^2 above the tuyeres, and therefore better utilization of carbon as a whole in the furnace.

The weak spot of the blast furnace is the smelting zone, and any factor which increases the amount of available heat units in that zone, and their initial combustion temperature, works great economy in the furnace.

More or less dried blast works similar results for a similar reason. The removal of moisture increases the available heat units in the smelting zone by the heat previously necessary to decompose the moisture. This results in more heat units available, and at a higher initial combustion temperature. Irregularities of temperature due to changes in the weather are also eliminated, making for more regular running, and saving the margin of excess fuel usually carried against such emergencies as a rapid increase in air humidity.

More available heat units in the smelting zone at a higher initial combustion temperature is the secret of the efficiency of hot blast and of dried blast.

It is possible to still further increase these two factors of economy. By increasing the percentage of oxygen in the air we *do not* increase the heat units available per unit of carbon burnt, but we increase, for the same available heat units, their initial combustion temperature. Operating with a blast of air preheated to 500° C., let us say, pure air gives a theoretical maximum combustion temperature of 2096° C. This is not the temperature to which the materials being smelted are raised, but is the initial temperature of the gaseous products of combustion at the moment of their formation, which products by cooling down impart their heat to the materials being smelted.

If, now, the air used is increased, say, 5 per cent in oxygen, to 25.8 per cent oxygen and 74.2 per cent nitrogen, the amount of inert nitrogen carried in is decreased 24 per cent for a given quantity of oxygen and it can be calculated that the maximum temperature of combustion will be increased to 2466°, an increase of 370°. It should be noted, however, that this increase of temperature is not accompanied by any increase in the number of available heat units per unit of coke burnt; these remain unchanged, but they are available at a much higher initial temperature.

We must next show what effect this higher initial temperature of the same number of heat units can have on the furnace running. It effects the rate of transfer of heat to the substances being smelted, i. e., the rate of smelting down. The gases produced are 16 per cent smaller in volume and at nearly 18 per cent higher temperature. They will, therefore, cool off in giving up their sensible heat to the materials being smelted very much faster; how much faster is somewhat problematical. As an estimate, we could say that the smelting operation might proceed 16 plus 18, or 34 per cent faster. This would mean an increased capacity of the furnace of one-third.

Faster running would further decrease radiation losses per unit of iron made; smaller volume of nitrogen would decrease the velocity of the gases through the charge and therefore increase the efficiency of reduction, i. e., the percentage of CO^2 in the escaping gases, the temperature of the escaping gases would also be decreased, more of their heat being thus left in the furnace. The temperature of the furnace would decrease more rapidly above the tuyeres than it now does, giving a lower average temperature to the outside furnace shell and therefore decreased radiation and conduction losses to the air. The escaping gases would be relatively richer in combustible, and therefore better fuel gas.

Summarizing these advantages, we feel confident in predicting that when a blast furnace is run on air enriched 5 per cent in oxygen, that it will run normally one-third faster than when run with ordinary air, and will save probably one-sixth the fuel requirement.

Lehigh University.

The Silver Cyanide Mill of the San Rafael y Anexas Co., Pachuca.*

By E. Girault, *General Manager.*

This mill, built in 1908, belongs to the type introduced the year before in the San Francisco Hacienda by Mr. A. Grothe, to whose foresight and personal effort we owe the most efficient

Fig. 1.—General view of the cyanide mills of the San Rafael y Anexas Company, Pachuca.

and rapid cyanide process known in Mexico, and to whom credit must be given for the valuable service thus rendered to the silver industry of Mexico.

Ores.—The ores from the San Rafael Mines, derived from the Vizcaina vein (the mother lode of the district), contain from 70 per cent to 75 per cent silica and from 10 per cent to 20 per cent of limestone. The silver, mixed with a small amount of base sulphides, is found mostly in the state of Ag_2S. The gold occurs in a nearly constant proportion of 4 grams to 5 grams per kilogram of silver in the low, and 3 to 4 in the high grade ores.

All ores above 300 grams of silver to the ton are cyanided.

Treatment.—The ore is hand-picked; crushed in Blake crushers and sampled in the mine yard; ground in cyanide solution in stamp batteries; concentrated on Wilfley tables; classified in Dorr classifiers; reground in Krupp tube mills; settled in Dorr pulp thickeners, and agitated in Pachuca pneumatic tanks.

The solutions are filtered in Moore vacuum filters and precipitated on zinc shavings.

Plans I and II show the flow through the mill.

Power and Water Supply.—The mill is driven by electric power supplied by the "Compania de Potencia de Regla" Westinghouse motors being employed.

Each battery of 20 stamps has a 75-hp motor; each Krupp mill one of 100 hp.

The total power consumption is 1.68 hp-day per ton of ore.

The water supply is drawn from the mine; the water is slightly alkaline.

Crushing.—There are 80 stamps in use; 40 Krupp, of 850 lb., set on timber foundations, and 40 Allis-Chalmers, of 1250

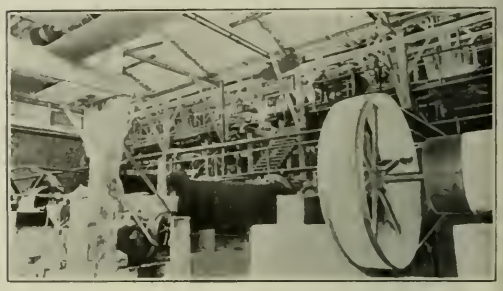


Fig. 2.—Inside view of mill.

lb., bolted to concrete foundations. The light stamps drop 7¾ in. and the heavy ones 6¾ in. 104 times per minute. The shoes and dies of forged steel last from 90 to 100 days.

Tyler's ton-cap screens, Nos. 58 and 365, equivalent to No. 10 mesh, 18- wire, 1.32 mm opening, and to No. 12 mesh, 20-wire, 1.067 mm, respectively, are used in the batteries.

The 850-lb. stamps crush from 3 tons to 3½ tons through 10-mesh per day; the 1250-lb. stamps crush from 6 tons to 6½ tons through 10-mesh, and from 5½ tons to 6 tons through 12-mesh per day.

From 7 tons to 8 tons of solution of one of ore is used in the batteries.

With No. 10 screens the capacity of the batteries is 400 tons per day, but at present crushing is limited to 350 tons to avoid overloading of the tube mills.

One pair of spring rolls, 36 in. x 16 in., and two more tube mills have been ordered; when installed No. 8 screens will be used in the batteries, and it is expected to raise the crushing to about 500 tons per day.

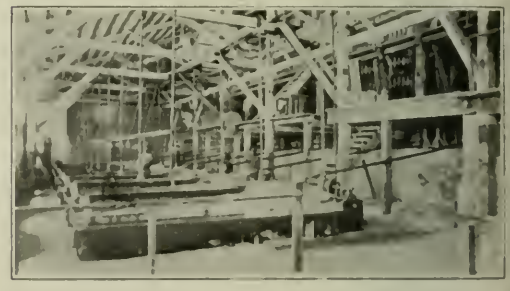


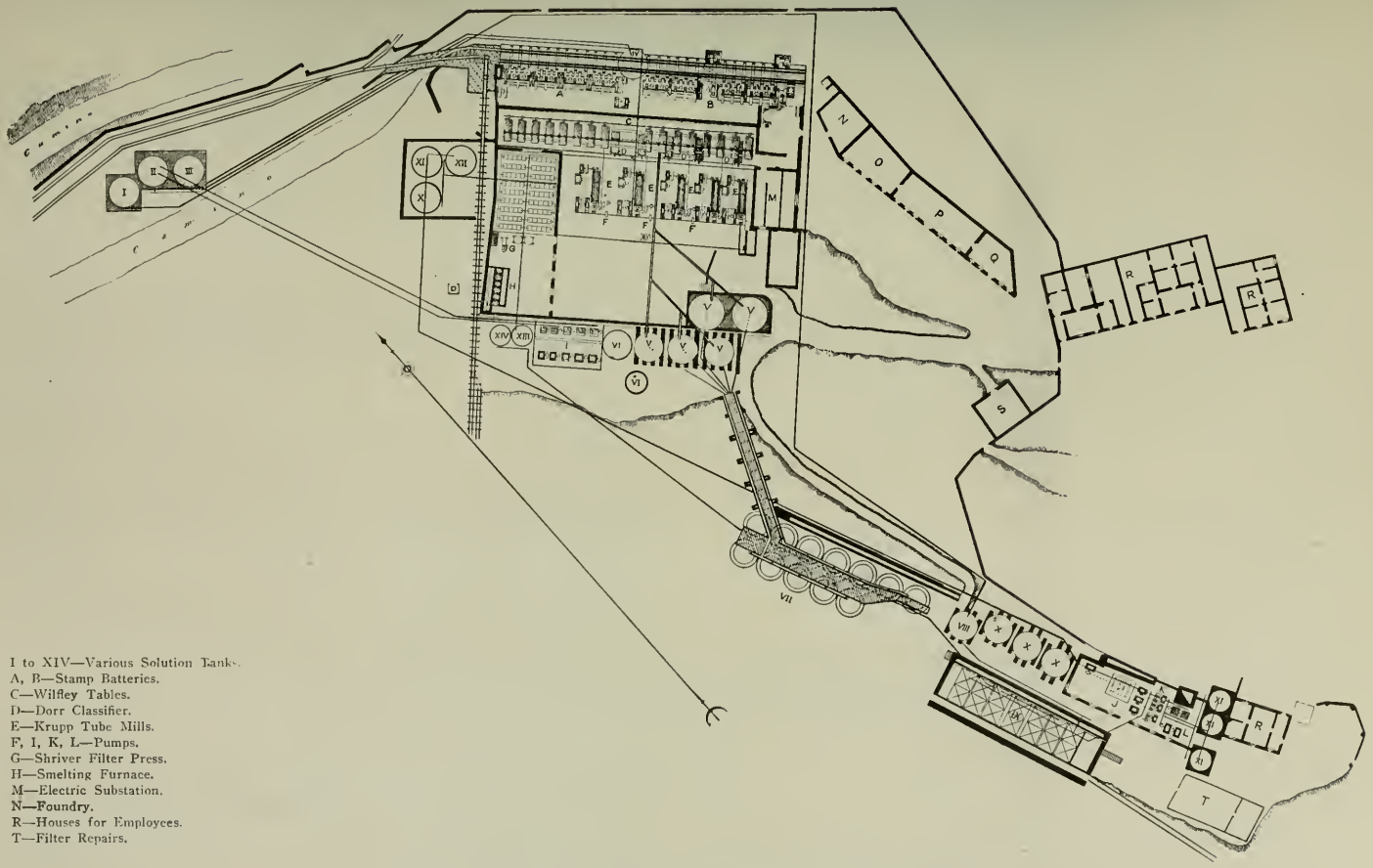
Fig. 3.—Concentration plant.

The following is a sizing test of the battery pulp, No. 58 ton-cap — No. 10 mesh screens being used.

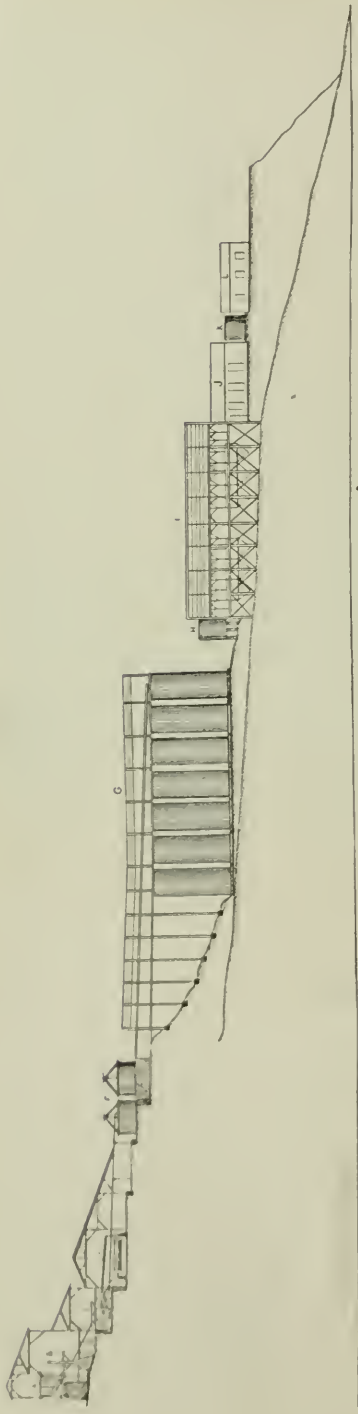
+ 40	37	per cent
+ 60	9	" "
+ 100	7	" "
+ 150	13	" "
+ 200	4	" "
— 200	30	" "

The total power consumption in grinding, including crushers, stamps and tube mills, is 1.05 hp per day per ton.

*A paper presented before the Instituto Mexicano de Minas y Metalurgia. From Informes y Memorias del Instituto vol. I, No. 3. Dec., 1909.



PLAN OF CYANIDE PLANT, SAN RAFAEL, PACHUCA, MEXICO.



CYANIDE PLANT, SAN RAFAEL, PACHUCA.

B—Stamps. C—Wilfley Tables. D—Dorr Classifiers. E. Krupp Tube Mills. F—Dorr Pulp Thickeners. G—Pachuca Tanks. I—Moore Filter. J—Compressor and Pumps. K—Rich Solution Tank. L—Houses for Employees.

Mill Solution.—This is stored in two tanks, II and III, of 200 cu. m. each; these tanks are used alternately to supply the mill and to receive the barren solution, being worked in cycles in order to avoid the enrichment of the solution.

According to the class of ore treated from 6 kg to 12 kg of

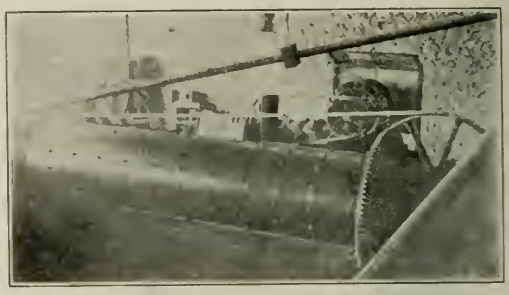


FIG. 4.—TUBE MILL.

lime are added in the bins per ton of ore. The alkalinity of the solutions is kept at 1 kg of CaO to the ton.

The mill solution averages 0.25 per cent of KCy; the cyanide loss in crushing is about 600 grams to the ton of ore. The extraction in the grinding process is high, averaging during October 38.3 per cent of the silver and 70 per cent of the gold contained in the ore.

Concentration.—Sixteen Wilfley tables, one for each five stamp battery, are used.

FIG. 5.—PACHUCA TANKS

From 17 tons to 30 tons of ore per day are fed to each Wilfley without overloading them.

The Wilfley tables are also used for unwatering the pulp and as auxiliary classifiers; the slime overflow being sent to pulp thickeners.

During the month of October 22.09 per cent of the values were recovered in concentrates containing:

Silver	25.586	kg per ton.
Gold	126	gr. per ton.
Iron	34.45	per cent.
Silica	12.25	" "
Lime	3.40	" "
Sulphur	40.50	" "

The concentrates were sold to the smelters.

Classification.—Five Dorr classifiers, one for each tube mill, are used.

From 50 tons to 70 tons of pulp, all the return from the tube mill, are fed to each classifier, the pulp being in the proportion

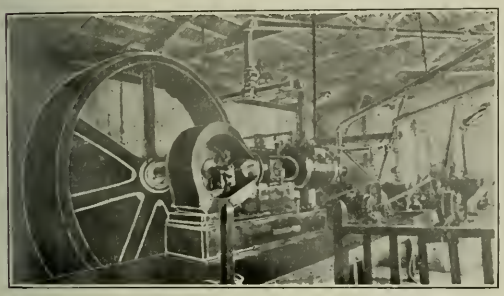


FIG. 6.—AIR COMPRESSOR.

of 1 to 2.38 of solution; the discharge contains 1 of solution to 2 of dry sand.

The Dorr classifiers have been working continuously for one year without giving any trouble.

The slime discharge classified averaged in October:

+ 100	2.5	per cent.
+ 150	15.5	" "
+ 200	7.5	" "
— 200	74.5	" "

Tube Mills.—Five Krupp No. 5 tube mills, 4 ft. x 20 ft., with Neal's baffle at both ends and El Oro liners, are in use.

Their capacity is rated at 50 tons of sand per day, the heads averaging:

+ 100	48.0	per cent.
+ 150	39.6	" "
+ 200	4.2	" "
— 200	8.2	" "

The tails, 74.5 per cent — 200, as noted above under classification.

Pulp Thickeners.—Five Dorr pulp thickeners, receiving slime diluted 1 to 10 and discharging 1 of dry slime to 1.2 of solution, are employed to unwater the pulp previous to agitation in the Pachuca tanks.

Three of the thickeners, 20 ft. x 10 ft., are supplied with 70 tons of dry pulp each; the other two, 24 ft. x 10 ft., will take 100 tons per day, and all overflow clear solution.

Pachuca Tanks.—Ten 15-ft. x 45-ft. Pachuca tanks are used for the agitation of the pulp at present; two more have been ordered and are soon expected.

The average charge is 100 tons of dry slime and 120 tons of solution.

One part of slime to one of solution (120 tons of slime to 120 of solution) has been tried, as well as 1 to 2, and 1 to 3 solution, but the best proportion was found to be 1 of ore to 1.2 of solution.

Compressor.—One Rand-Ingersoll air compressor, 18¼ in. diameter by 16 in. stroke, delivering at 100 revolutions 700 cu. ft. of air per minute to elevation, is employed for supplying air to the Pachuca tanks, to the Moore filter tanks, etc.

The amount of air required for an energetic agitation in the Pachuca tanks is estimated in San Rafael at 100 cu. ft. per minute at 25 lb. pressure.

The pulp is agitated during 36 hours, as it has been found that during this period the silver extraction increases about 20 gr.

Filtration.—Two units of the type A No. 3 of the Moore filter are employed; a third unit has been ordered.

The leaves are 10 ft. x 6 ft., and their capacity is 2.5 tons of dry slimes per day; 160 leaves in four baskets of 40 each are used, and 80 more are kept in reserve for repairs.

Two traveling cranes. worm-geared with 30 hp for traveling, are used for transferring the baskets.

According to the condition of the leaves a 1-in. cake is formed in from 20 minutes to 30 minutes.

One hour barren solution wash, and 15 minutes water wash are given to the cakes.

On each basket one 16-in. x 20-in. spiral pipe serves as vacuum chamber and holds the cake during the five minutes of the transfer.

In order to clarify the solutions, sand filters are used before the zinc boxes.

During October the water discharge assayed 4.5 gr. of silver to the ton; the cake, 55 grams unwashed and 50 grams washed.

During November, treating higher-grade ores and employing three baskets only, one of them in bad state, the cake discharged assayed 71 grams of silver unwashed and 63 grams washed.

Precipitation.—Zinc shavings, 0.006 in. thick, are used in precipitation.

Two tons of solution to the ton of ore are precipitated.

The heads in the boxes assay from 200 grams to 300 grams of silver; the tails, 2 grams.

The clean-up is made weekly.

The short zinc, which remains on a 20-mesh screen, is returned to the boxes; the shorts on 40-mesh are melted.

The precipitates are pressed and dried to 20 per cent moisture on a 24-in. x 24-in. Shriver press.

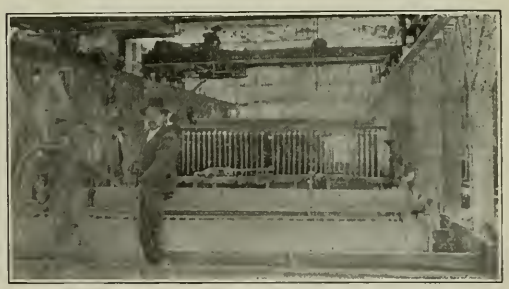


FIG. 7.—MOORE FILTER PRESS.

Melting.—Coke furnaces and Dixon's No. 300 graphite crucibles are employed.

The precipitate is melted with the following flux:

15 per cent broken glass.
6 per cent borax.
4 per cent sodium carbonate.

The short zinc, which remains on the 40-mesh screen, is fused with:

20 per cent broken glass.
8 per cent borax.
6 per cent sodium carbonate.

The bars, without remelting, assay from 920 gr. to 950 gr. silver and 5 gr. of gold to the kilogram of silver.

The slags are crushed quarterly in a battery and concentrated on a Wilfley table. The concentrates are melted and the tails treated in the cyanide plant or sold.

Resume.

The resume of the November, 1909, results is given below:

Tons crushed, 8393.

Assays per ton of ore:

Ore Contents:

Silver	0.901 kg	(28.967 oz. troy).
Gold	4.430 gr	(0.151 oz. troy).

Concentrates:

Silver	25.348 kg	(814.938 oz. troy).
Gold	141.600 gr	(4.552 oz. troy).

Heads of Pachuca tanks:

Silver	0.397 kg	(12.763 oz. troy).

Filter discharge:

Unwashed silver	0.071 kg	(2.283 oz. troy).
Washed silver	0.063 kg	(2.025 oz. troy).

FIG. 8.—PRECIPITATION BY ZINC SHAVINGS.

Extraction:

In concentration, silver	24.19 per cent.

Cyaniding:

Crushing	31.74 per cent.
Agitation	36.18 " "
Total extraction, by assays, silver	92.11 " "
Total extraction, by calculation, silver	91.43 " "
Total extraction, by calculation, gold	97.08 " "

Consumption per ton of ore:

Sodium cyanide, 128 per cent	1.078 kg.
Lead acetate	0.317 "
Lime	6.706 "
Flint pebbles	1.227 "
Zinc	0.843 "
Coke	1.646 "
Borax	0.084 "
Soda carbonate	0.044 "

The cost per ton was:

General expenses	$0.78
Grinding and concentrating	1.18
Cyaniding	1.32
Pumps and compressor	0.15
Filtration, including royalty	0.33
Precipitation and melting	0.44
Total Mexican money	$4.20

Properties and Behavior of Furnace Electrodes.

By Carl Hering.

In an article in the October issue of this journal (Vol. VIII, p. 442, 1909) the writer described an analysis of the laws of the proper proportions of electrodes and the losses of energy in them, based on the fundamental principle that there should be no chilling of the furnace by the electrodes or unnecessary heating within the walls. Owing to the great mathematical complications involved by including in the algebraic deductions some of the refinements which belong more properly to what is termed the second approximation in a complicated case, the writer preferred the method more generally used in such cases and recommended by our ablest scientists, to take up the simpler case first by itself, and consider the refinements later in the form of corrections. He then showed how most of these corrections could readily be taken care of by determining the physical constants in such a way as to include those factors, thus making the simple solution practically exact.

In any case, no matter how complicated, and as far as it concerns the electrode as a whole, that is, the end results only, an electrode may always be considered to be replaced by another whose conductivities are uniform throughout and equal to the mean, virtual or average conductivities of the original one; this equivalent electrode then follows the simple laws developed by the writer. The end results, that is, the flows of heat at the two ends, will be the same in the equivalent electrode as they are in the original, but the behavior of the parts intermediate between the ends will not.

The whole subject is discussed more in detail, accompanied by experimental determinations, tables and curves, in a paper on "The Proportioning of Electrodes for Furnaces" to be read before the American Institute of Electrical Engineers and to be published in the *Proceedings* of that society for March.

Although the problem of calculating electrodes so as to include these corrections is thereby solved in a very simple way, it is nevertheless quite instructive and of interest to examine what the nature of the changes in the behavior of the intermediate parts of the electrodes are, which are caused by the different physical properties of the materials, including the temperature variations in the conductivities. The purpose of the present article is to do this.

When a mathematical analysis becomes so complicated that it confuses and obscures rather than clarifies the situation, one may often obtain very useful results by means of analogies. In the present article the effect of those changes will be examined in this way.

The comparisons will be made between a flow of electricity (an electric current), a flow of water, and a flow of heat.

Analogies.—In Fig. 1 let E be a very large electrically charged sphere whose potential above the earth is E. Let the conditions be that a wire R of a uniform resistance must be attached to it and to the earth; a current C would then flow whose value is $C = E/R$, supposing the charge to be practically infinite, and R to be the resistance through which the current passes. This current is a flow of coulombs per second. Let it be required to stop this flow without opening the circuit. This can evidently be done by passing a current equal to C from some other source into the wire at the extreme end a near the sphere. Or it may be done by a current equal to $2C$ applied at the middle point b; or by a current $10C$ applied at a point c, which is 9/10 of the whole distance from the sphere.

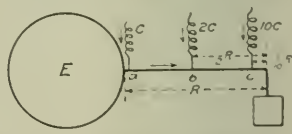


FIG. 1.— ELECTRICAL ANALOGY.

It will be seen that the current which is required increases in the same proportion as the point of application is moved nearer the zero end. In all cases E is always the same, and C increases as fast as R diminishes, so that their product is always equal to E. But the power required is $C\,E$, hence it will increase in proportion to C—that is, it will require the least power when the current is applied at a, and most when applied at c.

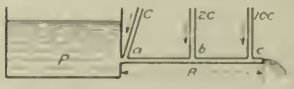


FIG. 2.—ANALOGY WITH FLOW OF WATER.

It will be noticed that when the current is applied at the

middle point *b*, it amounts to the same thing as a current entering the wire all over its whole length, at a uniform rate per inch. Or in other words, the points *a*, *b* and *c* may be considered to be the centers of gravity of currents entering all over the wire, evenly for the point *b*, and unevenly for the points *a* and *c*. If for any other distribution of the entering current the center of gravity of the distribution is known, the above laws may be applied.

In Fig. 2 let *P* be a tank of water with a pipe *R* attached to it having a resistance to the flow; and assume for the sake of the analogy, that this resistance also follows a law like Ohm's law. Then exactly the same argument applies to stopping the outflow of water from this tank dynamically by streams of water entering the pipe at various points *a*, *b* and *c*, or all over, either regularly or irregularly, the center of gravity of such distributed influx being known.

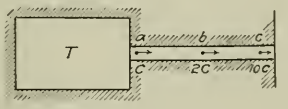


FIG. 3.—FLOW OF HEAT IN ELECTRODE.

In this case it is the hydraulic pressure *P* of the tank which is always the same, and the various currents *C*, now the current or flow in gallons per second, must always be such that $C = P/R$. The current and the power to produce it will again increase in proportion as the center of gravity of its application moves away from the tank.

In Fig. 3 let *T* be a furnace having a constant temperature *T*, and let *R* be an electrode offering a resistance to the flow of heat, and having its other end kept cold. The analogy applies again. The flow of heat *C* from the furnace is again such that $C = T/R$, the temperature *T* being now that quantity which is always the same. This flow can again be stopped by the application of another flow in the same direction from an independent source. If the resistance again follows Ohm's law, as it does, the same conclusions apply as those found above, namely, that the flow of this applied heat will be least when applied near the furnace end, and will increase as the center of gravity of its application is farther from the furnace. The flow or current is now measured in calories per second, which is a quantity of the same nature as watts. In this case the heat current itself is power, hence the flow or current and the power to produce it are numerically the same.

The number of watts required to stop the flow from the furnace will therefore increase in the proportion as the center of gravity of the inflow from the other source is farther removed from the furnace. It will be greater or less than $2C$, depending upon whether the center of gravity of its point of application is to the right or left of the middle point *b*, respectively.

Now, this inflow of watts may be produced by an electric current in the electrode *R* itself, as it actually is done in practice. The rate of production of these watts along the electrode will then depend upon the local resistance—that is, the electrical resistivity at the various places. If it is evenly distributed—that is, if the resistivity is constant for all temperatures, it will be equivalent to all the watts being generated at the middle point *b*. In that case, from the analogy, the power required will be twice that represented by the watts of heat flow from the furnace if there were no additional current. This is the case assumed by the writer in the analysis referred to above.

Varying Electrical Resistivity.—Now let it be assumed first that the thermal conductivity is constant throughout, and that the electrical resistivity falls with the temperature, as it does with carbon. Then it is evident that more watts will be generated in the colder half than in the hotter one. Hence, the center of gravity for the whole length will be moved toward the colder side of the middle; hence, the conditions will approach to those in Fig. 3, in which the watts are applied at *c* near the cold end. From the analogy it follows that the watts now required to stop the furnace flow will be *greater* than they were for an electrode having the same total resistance but evenly distributed. And they will become greater the faster the resistivity falls with the temperature. As these watts are the electrode loss, it follows that *a falling resistivity increases the loss*, because it moves the center of gravity of the generation of the watts toward the cold end.

Hence, a falling resistivity is an undesirable feature for an electrode. In unscientific terms, it may be said that it crowds the heat toward the cold end where it is not wanted. In Fig. 4, if *b* is the normal heat gradient for a constant resistivity, then a falling resistivity will tend to give it a shape more like the curve *c*, which is undesirable.

Let it now be assumed that the resistivity rises with the temperature, as it does with iron. In an iron electrode more watts will be generated in the hotter than in the colder half. This moves the center of gravity of the total watts toward the hot end; hence, the conditions will approaoch those in Fig. 3, in which the watts are applied at *a* near the hot end. It has been shown that the watts required to stop the flow of furnace heat will then be *less* than when applied at the middle. Hence, in iron electrodes less watts will be required to stop the flow of furnace heat than if the resistivity were constant. And the number of watts will become less and less as the resistivity rises more rapidly with the temperature.

Hence, as these watts are the electrode loss, it follows that *a rising resistivity diminishes the loss*, and is therefore a desirable feature in an electrode. It may be said that it crowds the heat toward the hot end, which is the right place for it to be. Referring to Fig. 4, it tends to change the normal heat gradient from *b* toward *a*, which is very desirable; the more rapidly this curve falls at the hot end and the less rapidly at the cold, the better.

Varying Thermal Conductivity.—Now let it be assumed that the electrical resistivity is constant and that the thermal conductivity varies with the temperature. The watts will then be generated evenly throughout the whole length.

When the thermal conductivity falls with the temperature it means that the heat gradient will fall more rapidly in the hot half and less rapidly in the colder half than it would for a constant conductivity. Referring to Fig. 4, it will tend to change the heat gradient from *b* to *a*. This, as was pointed out above, is in the right direction. Hence, *a falling thermal conductivity is a desirable feature.*

When, on the other hand, the thermal conductivity rises, it means that the heat gradient will fall less rapidly in the hotter half and more rapidly in the colder one than for constant conductivities. Referring to Fig. 4, this will tend to change the normal gradient *b* to the undesirable one *c*. Hence, *a rising thermal conductivity is an undesirable feature.*

Examples.—Recent determinations of these physical constants by the writer show that up to about 1000° C., and probably above that also, carbon has a falling electrical resistivity, and a rising thermal conductivity (not a falling one, as has been incorrectly claimed.) Hence, in this material both features are in the undesirable direction, as they both tend to crowd the higher temperatures toward the colder end, where they are not wanted; and as was shown above, this increases the minimum loss. As both these tendencies are in the same direction, each in turn helps to increase the other, making the final heat gradient worse than it was for either alone. The heat gradient for a carbon electrode will therefore be somewhat like the curve *c* in Fig. 4.

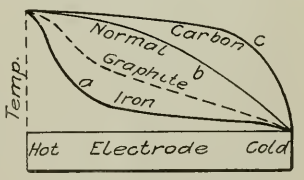


FIG. 4.—TEMPERATURE GRADIENTS.

On the other hand, these same determinations have shown that for iron the electrical resistivity rises rapidly and the

thermal conductivity falls. Hence, both are in the favorable direction, as they both tend to crowd the higher temperatures back toward the furnace where they belong; and this tends to reduce the minimum loss. Again, as both features act in the same direction, each in turn will increase the other, making the final heat gradient better than it would be for either alone. The heat gradient for an iron electrode will therefore be somewhat like the curve *a* in Fig. 4.

For graphite these tests showed that the electrical resistivity falls, though much less rapidly than for carbon, at least for moderate temperatures; it is, in fact, not very different from a straight line. Graphite, therefore, shares this undesirable feature with carbon, though it possesses it to a less degree. But, on the other hand, the tests showed that the thermal conductivity falls, and quite rapidly, much more rapidly than for iron. Hence, it shares this good quality with iron. As these two features are now opposed to each other in their action on the heat distribution in an electrode, they tend to neutralize each other; but as the rate for the falling thermal conductivity is considerably greater than that for the rising electrical resistivity it predominates, the result being that it moves the heat gradient below the normal one *b*, Fig. 4—that is, toward the one for iron, *a*, which is the favorable direction. Hence, graphite, like iron and unlike carbon, tends to crowd the heat toward the hot end where it belongs.

This feature was also confirmed by a comparison of the resistivities of graphite *at* various temperatures with those obtained by the writer, which are the mean values for various ranges of temperatures. The rate of change of the latter was found to be less than that obtained by the arithmetic means of the former, as it should be.

Thermal Insulation.—Incidentally, the different shapes of the heat gradients in Fig. 4 indicate the error that would be committed by assuming any fixed coefficient for the leakage of heat from the electrode to the walls, as has been suggested. This leakage, however little or much it may be, depends, of course, on the differences between the temperatures of the electrode and of the wall at neighboring points. Now, the heat gradient in the wall will presumably be approximately a straight line joining the two ends of the heat gradient curves in Fig. 4. Hence, for carbon the leakage would be from the electrode to the wall, while for iron and probably graphite, too, it would be from the wall to the electrode; hence, not only different in values and in place, but even with a reversed sign! This shows the great mistake that might be made by measuring such a coefficient for carbon, for instance, and then applying it to an iron or graphite electrode; it would make matters worse than to let it alone.

Electrode Means.—The empirical values above referred to are the proper mean values to be used in the formulas developed by the writer and given in the earlier paper above referred to. They are the values of an equivalent electrode whose two conductivities are constant throughout the whole length of electrode and are equal to this value for each particular range of temperature, but have a different constant value for a different range. Such an electrode *as a whole* will act just like the original one, differing only in the distribution of the heat between the ends—that is, in the heat gradient, as was discussed in this article. As these values are a mean peculiar to electrode conditions, being neither exactly an arithmetic or geometric mean, the writer proposes to designate them by the term "electrode mean."

It may be of interest to add here that when these electrode mean values are used all the laws of electrode losses deduced by the writer in his former article seem to apply correctly even when the variations of the conductivities with temperature are very great. The explanation seems to be that these experimentally determined mean values embody all the correction factors due to these variations.

The results of these experimental determinations are in part surprising, as they show that some of our former figures have been considerably in error, in one case even in the direction (sign) of the variation. This is in part due to the fact that they are the electrode means and therefore include the effects of the variations pointed out in this article.

Carbon turns out to have a much higher mean thermal conductivity than was formerly supposed, bringing it up to about half that of graphite instead of 1/8 and 1/10, as has been claimed. This high mean value is undoubtedly due in part to the properties described in this article. For graphite it is also higher, being above that for iron. These data are published elsewhere in detail, but briefly it may be said here as a confirmation of what has been deduced in this article, that for a range of temperature of about 1400° C. to 100° C. the minimum losses are about in the following ratios: Copper, 1; iron, 1.5; graphite, 5.0, and carbon, 7.5. While as to the size, the proportions are about: Copper, 1; iron, 6.6; graphite, 14.5, and carbon, 45.

The writer's researches have, therefore, shown that in almost, if not even in all these features, the material which has been most used for electrodes, namely, carbon, is the worst, while by far the best, if it could be taken advantage of, would be copper.

Another result of the writer's investigations showed that, although the two conductivities of electrode materials are interesting, instructive and useful quantities in an investigation of this kind, and enable the behavior of the electrodes to be studied, neither of them alone are direct measures of the desirable qualities of a material for electrodes, as was clearly described in his earlier articles and papers. This led to finding what physical properties are the true relative and absolute measures of the desirable qualities of materials when used for electrodes, and this, in turn, shows that when these latter constants are determined, the two conductivities are no longer concerned and, therefore, need not even be known at all for calculating the proportions of electrodes or the losses in them. In fact, these conductivities have in the past even misled us to quite wrong deductions and conclusions, some of which have become so deep-rooted that it will be difficult to correct them. By using these new and more rational physical properties, and deductions from them, the extreme simplicity of the calculations brings them almost into the same class as those based on Ohm's, Faraday's and Joule's laws. These two new quantities are the electrode voltage and the specific section, as also their derivatives for specified temperature ranges, as explained more in detail in the paper above-referred to.

Philadelphia, Feb. 14, 1910.

Definition of Industrial Terms.

At the recent annual meeting in Boston of the Division of Industrial Chemists and Chemical Engineers of the American Chemical Society, a committee was appointed by the chairman, under authority of the by-laws of the section, "On the Definition of Industrial Terms." In constituting this committee, it is the desire of the division that it co-operate as far as possible with trade organizations in the several lines of industry in formulating definitions of their products which shall be as accurate and as fair as possible to the interests involved. The organization of such a committee has been suggested by various interests who desire a more authoritative definition of some industrial terms such as "petroleum ether," "gasoline," "benzol," "bronze," "benzine," "steel," "Portland cement," etc.

The general committee is desirous of receiving suggestions from all who are interested in the subject as to terms which are considered the most important to be defined at the present time, owing to ambiguities which may now exist, and as to the best methods of accomplishing the work. All communications should be addressed to Mr. Clifford Richardson, 30 Church Street, New York City, who is chairman of the committee, the other two members being Messrs. J. B. F. Herreshoff and George T. Adamson.

The Magnetic and Electrical Properties of Iron-Silicon Alloys.

By Charles F. Burgess and James Aston.

The belief was held for a long time that, as far as magnetic properties were concerned, purification was the method to be looked to for improvement of iron; that the highest permeabilities were to be reached the nearer our material approached an analysis of Fe 100 per cent.

That this is not true is now, of course, common knowledge. The work of Barrett and Brown, in 1901,[1] on alloy steels, made by Hadfield, gave for irons with additions of 2¼ per cent of aluminium or 2½ per cent of silicon permeabilities excelling those of pure iron; investigations of the author have shown this same effect by additions of about 2 per cent of tin or about 4 per cent of arsenic.

It is not probable that the influence of these elements, all non-magnetic, is directly, or in a strict chemical sense, the cause of the marked improvement of magnetic quality. It is more likely that the effect is due indirectly through some change in the physical structure of the iron itself. This supposition is seemingly confirmed by the influence of the heat treatment of high-purity iron. Annealing at lower temperatures

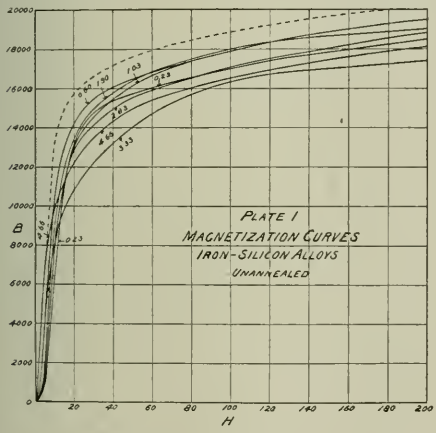


FIG. 1.—MAGNETIZATION CURVES, ALLOYS. UNANNEALED.

(about 700° C.) results in improvement of quality throughout the entire magnetization curve, due to the removal of all cooling and forging strains; while annealing at the higher temperature of 100° C. results in a deterioration of quality in the upper ranges, and a higher permeability at low magnetizing forces. And, of course, we have the concomitent increase in the grain size of the iron.

From the investigations of Hadfield, Barrett and Brown has resulted the present-day materials so largely used in transformer work, in which the percentage of silicon varies from about 3 per cent to 6 per cent. In these silicon steels the heat treatment is of importance comparable with that of the silicon content, and the practice seems to be annealing at high temperatures. The result is a material of very high permeability for the lower magnetizing forces; the saturation limit is reached at comparatively low densities, however, and beyond this point the quality is not exceptional. This characteristic makes the steel particularly suited to alternating-current working, where low working densities are the rule. The low hysteresis losses, together with the added benefits of high electrical resistance and slight deterioration due to ageing, are the special features of the silicon steels.

[1] "The Conductivity and Magnetic Properties of Alloys of Iron," W. F. Barrett and W. Brown, Jour. Inst. Elec. Engrs., Vol. 31, p. 675.

Our investigations[2] covered tests of 15 iron-silicon alloys, of which the following seven bars were analyzed and form the basis for discussion.

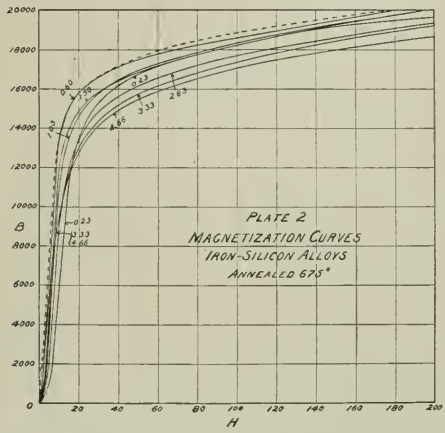


FIG. 2.—MAGNETIZATION CURVES, IRON-SILICON ALLOYS. ANNEALED 675°.

Bar.	PER CENT SILICON. Added.	Analysis.
96F	0.25	0.233
96A	0.50	0.603
96B	1.0	1.033
96C	2.0	1.897
109A	3.0	2.826
113V	7.0	3.334
96H	5.0	4.655

The agreement between added and actual silicon content is close, except for bar 113V.

As mentioned in articles which have appeared previously in this journal, electrolytic iron of high purity (99.9+) formed

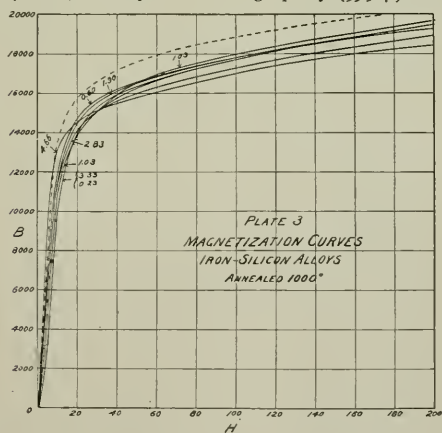


FIG. 3.—MAGNETIZATION CURVES, IRON-SILICON ALLOYS. ANNEALED 1000°.

the basis of the alloys. All effects noted, therefore, may be ascribed, as far as due to chemical composition, to the added element alone. These alloying materials were melted into ingots, which were forged and turned into bars for testing.

[2] A research on "Electrolytic Iron and Alloys" at the University of Wisconsin, under grant of the Carnegie Institution of Washington.

The results of the tests are noted in Tables I and II and in Plates 1, 2, 3, and 4. Four heat treatments were adopted—as forged, annealed at 675° C., annealed at 1000° C., and quenched at 900° C. Comparison is made with a standard bar of electrolytic iron of high magnetic quality; this is indicated in the plates by a dotted curve.

The values of retentivity given in Table II reflect the conditions indicated by the magnetization curves, and vary with the maxima reached by the curves. In general, the coercive forces become less with annealing, and reach a minimum after heating to 1000°. Quenching has had a slight tendency toward increase. The very low value of 2.7 is reached by bar 96H (Si

TABLE I.—DATA OF MAGNETIC TESTS.

		H = 10				H = 20				H = 50				H = 100			
Bar.	Per cent Silicon.	Unann.	Ann. 675°.	Ann. 1000°.	Quench 900°.	Unann.	Ann. 675°.	Ann. 1000°.	Quench 900°.	Unann.	Ann. 675°.	Ann. 1000°.	Quench 900°.	Unann.	Ann. 675°.	Ann. 1000°.	Quench 900°.
117A	0	13100	10000	13350	11800	15750	15950	15450	14600	17600	17700	17050	16950	18850	18850	18200	18150
96F	0.23	6500	6000	9750	7850	13100	13250	13700	11550	15750	16700	16250	15850	17000	18350	17700	17700
96A	0.60	10700	13100	11750	12150	14150	15800	14700	14950	16450	17500	16550	16900	17850	18600	17800	18100
96B	1.03	8100	10700	10700	11800	13100	14450	13650	14650	16150	16850	16550	16650	17850	18200	17950	17950
96C	1.90	9200	11600	11200	9150	13450	14850	14450	12250	16350	16800	16500	15850	17950	18200	17750	17600
109A	2.83	8200	8900	11200	10000	12800	13250	14050	13050	15500	16150	16000	15950	17050	17700	17300	17550
113V	3.33	8000	9100	9750	10700	10800	12950	13850	14000	14050	15750	16350	16150	16350	17350	17750	17600
96H	4.66	9750	9300	12050	9100	12200	12700	14550	13200	14950	15400	15750	16250	16500	17100	17000	17600
M E.	Com.			12300				14400				16100				17700	
S E	Com.			10100				13100				10300				18000	
252	Com.			10700				13500				16400				17900	
182	Com.			11550				13850				15800				17550	

In the unannealed condition all the curves fall considerably below the standard, particularly in the upper ranges; the best results are obtained with bars of lower silicon content, and the lowest with those of highest silicon, viz., 113V, with 3.33 per cent Si, and 96H, with 4.66 per cent. However, it will be noted that for the very low values of H these two bars, and particularly 96H, cross the other curves and have the highest permeability of the series.

Annealing at 675° has raised the maxima of all curves, but the comparative order is about as in the previous condition. Bar 96A, with a low silicon percentage of 0.603, lies very close to the standard throughout the entire length.

Second annealing at the higher temperature of 1000° has resulted in a falling off of the permeability for these bars of lowest silicon content. But for three bars of highest percentage of silicon, viz., 109A (2.83 per cent, 113V (3.33 per cent) and 96H (4.66 per cent), there has been a considerable improvement of quality at the lower magnetizing forces, with but little change for the upper ranges. Here the curves fall below those of lower silicon, and all are well below the standard. The effect is most striking for bar 96H, with the highest silicon of the series

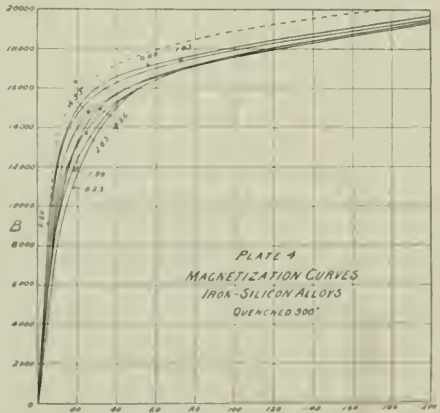


FIG. 4.—MAGNETIZATION CURVES, IRON-SILICON ALLOYS. QUENCHED 900°.

(4.66 per cent); its permeability is very high below H = 10. At this point the curve has a very sharp bend, and thereafter falls below the others. Quenching at 900° has resulted in a deterioration of quality, particularly in the lower ranges, with but little change at higher magnetizing forces

4.66 per cent) after annealing at 1000°, confirming the previous deductions of high permeability for this alloy at low magnetizing forces.

Summarizing, the results of these tests do not show a general improvement of permeability due to the presence of silicon

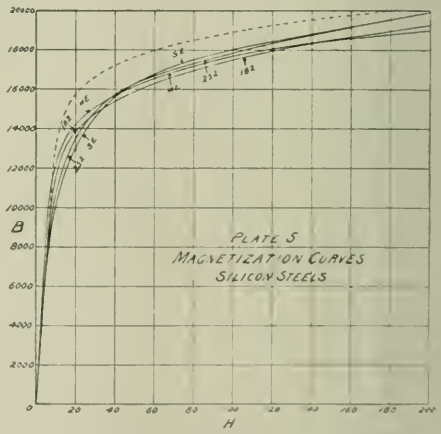


FIG. 5.—MAGNETIZATION CURVES, SILICON STEELS.

in electrolytic iron, but the experience of practice is confirmed in that a high-silicon alloy after annealing at high temperature is of high permeability for very low magnetizing forces and

TABLE II.—COERCIVE FORCE AND RETENTIVITY.

		H (Max.) = 200 Coercive force.				H (Max.) = 200 Retentivity.			
Bar.	Per cent Si.	Unann.	Ann. 675°.	Ann. 1000°.	Quench 900°.	Unann.	Ann. 675°.	Ann. 1000°.	Quench 900°.
117A	0	5.5	6.2	3.1	2.5	12300	13800	10000	8000
96F	0.23	8.6	9.4	5.2	4.3	11700	13800	9200	8000
96A	0.60	5.2	4.5	3.5	3.5	11500	13200	9000	10100
96B	1.03	7.5	6.8	4.1	4.0	12400	14200	9400	10000
96C	1.90	7.7	6.5	4.0	4.3	11800	14100	9300	9100
109A	2.83	8.5	8.2	4.7	5.1	12900	14300	10000	9900
113V	3.33	5.5	5.5	4.5	4.3	8700	11300	8500	9100
96H	4.66	4.5	4.9	2.7	3.3	12000	12800	8000	9600
M E.	Com.			2.1				9700	
S E	Com.			2.5				6100	
252	Com			1.0				8700	
182	Com			3.5				10600	

densities such as are used in transformer design. The improvement is not so great in our tests as it is in commercial materials; it may be that the effect of the silicon is more pronounced in the presence of certain impurities or after special heat treatment.

Commercial Steels.

For the purpose of comparison there is appended in Tables I and II, and plotted in Plate 5, results of tests on commercial silicon steels, the samples being obtained from steel and electrical manufacturers. The tests were made upon the same apparatus as those discussed above, and the results are thus strictly comparable. The dotted curve of Plate 5 is that of the electrolytic iron standard.

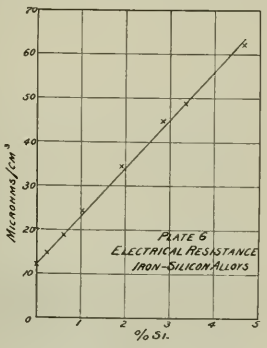


FIG. 6.—ELECTRICAL RESISTANCE IRON-SILICON ALLOYS.

The general character of the silicon steel is well shown and needs no special comment. Curves SE and 252 are of the same brand of material from a large sheet-steel manufacturer, but from samples obtained from different electrical concerns at different periods of time. The uniformity of the material is very striking; this is brought out with great clearness below, where the hysteresis tests data are tabulated.

All inductions have been carried to a maximum of B = 12,000.

Bar.	Per cent Si.	Loop area.	Relative area.	H (Max.)
117A	0	8.65	1.	8.2
96F	0.23	12.58	1.45	30.0
96A	0.60	7.05	0.81	12.5
96B	1.03	8.72	1.01	13.3
96C	1.90	9.53	1.10	1.80
109A	2.83	10.96	1.27	18.0
113V	3.33	8.77	1.02	13.4
96H	4.65	4.39	0.51	7.5
M E	Coml.	5.20	0.60	8.7
SE	Coml.	5.77	0.67	14.7
252	Coml.	6.10	0.71	14.1

These results are hardly a fair criterion of the quality of the electrolytic samples, since all were tested after quenching at 900°.

Electrical Resistance.

A feature of steels for transformer working, second only to the magnetic quality, is a high electrical resistance, to reduce the eddy-current loss. In this respect the silicon steels are an additional advantage.

Measurements of the series discussed in this paper are given in the following table:

Bar.	percent Si.	Microhms per cm³.	Relative resistance.
117A	0	12.1	1.0
96F	0.233	14.7	1.22
96A	0.603	18.7	1.55
96B	1.033	24.3	2.01
96C	1.897	34.4	2.84
109A	2.826	44.6	3.68
113V	3.334	48.6	4.02
96H	4.655	62.2	5.13
182	Coml. silicon	59.5	4.92

The above data are plotted in Plate 6, where the effect of increasing addition of silicon to the iron is indicated as an maximum of 4.65 per cent.

At this content (and this is the alloy with the minimum hysteresis loss) the resistance has risen to upward five times that of the electrolytic iron standard.

Chemical Engineering Laboratories,
University of Wisconsin.
Madison, Wis.

Manufacture and Transport of Ferrosilicon.

The British Local Government Board has issued a "Yellow Book," containing a report on the manufacture, uses, and transport of ferrosilicon. The authors are Dr. S. Monckton Copeman, F.R.S., medical inspector of the Local Government Board; Mr. Samuel R. Bennett, one of the Inspectors of Factories, and Dr. H. Wilson Hake, F.I.C., chemist.

The report was occasioned by certain accidents during the transport of ferro-silicon, which have been noted several times last year in our columns. The most interesting part of this Yellow Book, however, is probably the description of some details of operation in various European ferrosilicon works which were visited by Dr. Copeman. The chemical tests of Dr. Hake, with respect to the gases evolved, and the various researches on the properties of different ferrosilicons by Mr. Bennett and Dr. Hake also contain much of interest.

Composition of Charge.

Originally the charge of ferrosilicon furnaces was composed of a mixture of (1) iron-pyrites or other form of iron ore; (2) siliceous material in the form of quartzite or sand; (3) carbon in the form of charcoal, coal, or coke, together with (4) some lime as flux. Owing, however, to the obvious impurity (especially high phosphorus and sulphur content) of earlier samples of electric-furnace ferrosilicon, scrap-iron and steel shavings are now preferred to iron ore, and quartzite to sand, as being less productive of obstructions in the furnace. Further, the purer the starting material, the less slag will be formed. The addition of lime seems to have been entirely abandoned as unnecessary; this is of importance, in connection with the problem of the generation of dangerous gases from ferrosilicon (see below).

As the properties of the ferrosilicon produced depend to some extent on the raw materials used, the following details from Dr. Copeman's report on the raw materials used in various European plants will not be wholly without interest.

At the **Bozel** works of La Compagnie Generale d'Electrochemie de Bozel (one of the plants of the old ferrosilicon syndicate) the materials employed are as follows:

Quartz (or rather quartzite) obtained locally, containing in the average 95 per cent silicon, 4 per cent aluminium, with small amounts of calcium and phosphorus, the latter averaging from 0.3 to 0.8 per cent.

Steel turnings, containing not more than 0.3 per cent of phosphorus.

Anthracite coal, which is obtained locally and is cheap, but which yields as high a quantity as 20 per cent of ash.

The grades of ferrosilicon produced at the Bozel works represent percentages of silicon of 25, 50 and 80, respectively. In the manufacture of the 80 per cent grade, however, charcoal, although much more expensive, is employed instead of anthracite coal, on account of its great freedom from impurities.[1]

At the Keller-Leleux works at **Livet** (Isere) there are five 1200-hp Keller furnaces, making a melt of 500 kilograms of ferro-alloy every two hours, and three 500-kw furnaces, with a daily output of 12 melts, each of 2 tons. Not all these furnaces are running.

Actual working results have demonstrated that the energy required for the production of 1 ton 30 per cent ferrosilicon in the electric furnace is 3500 kilowatt hours, and at the Livet works it has been found practicable, with an expenditure of 4000 hp to turn out 20 tons 30 per cent ferrosilicon per day.

The materials employed in the Livet works in the manufacture of ferrosilicon are as follows:

Iron and steel shavings,

[1] As the cost of transport, especially in the mountain districts, constitutes an important factor, the particular form of carbon used depends on the facility with which one or the other kind is obtainable within reasonable distance from the works. In most instances anthracite coal is employed, of which large deposits are worked in the mountainous regions of the southeast of France. But this anthracite is often quite impure. Hence the substitution of charcoal in the manufacture of the higher grades of ferrosilicon.

Quartzite, containing 94 to 96 per cent silica.[2]

Anthracite coal (averaging ash 7 per cent; 0.013 per cent P., 0.4t per cent S, no arsenic).

"Iron and steel shavings, although more expensive, are exclusively used by this firm rather than 'tournures de fonte,' generally employed by other manufacturers for the reason that they contain much less phosphorus—a maximum of 0.10 as against an average phosphorus content of 0.50 to 0.80 for iron turnings."

At the **Giffre** works, St. Jeoire (Haute-Savoie), of the Société Electro-Chimique du Giffre) the constituents of the furnace consist of quartz-rock, anthracite and steel turnings, of which the quartz-rock contains about 96 per cent of silica. The anthracite of good quality from the point of view of small phosphorus content is obtained locally.

At the **Ugine** (Savoie) works of the Société Anonyme Electrometallurgique Procédés Paul Girod, the constituents of the charge are:

Quartzite, containing about 92 per cent of pure silica, with traces only of phosphorus and sulphur.

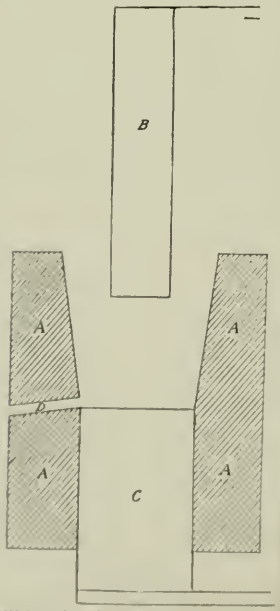
Coal (forging) obtained from St. Etienne—of best quality, and guaranteed not to average more than 8 per cent of ash, or more than .005 per cent of phosphorus or sulphur.

Steel and iron scrap.

At the **St. Marcel** (Savoie) works of the Société d'Industrie Electro-Chimiques La Volta, the charge constituents are:

Quartzite, obtained in the neighborhood of the works, with an average content of 98 per cent of silica, together with traces of magnesia, but no lime.

Gas coke, for production of the 50 per cent grade of ferro-silicon, while in the manufacture of 25 per cent alloy or other comparatively low grades this is replaced by anthracite coal, which is cheaper, but not so free from impurities.



A—Walls. BC—Electrodes. D—Tapping Hole

FIG. 1.—FERROSILICON FURNACE. DRAWN TO SCALE.

Iron-ore (for 50 per cent ferrosilicon) or steel shavings (for lower grades). The iron ore is stated to contain 95 per cent of iron oxide (Fe_2O_3), but no trace of aluminium, manganese or calcium.

The **Wednesfield** experimental station of the British Coalite Company, Ltd., use as constituents of the charge:

(1) Natural sandstone rock.

(2) Pitch, with addition of 10 per cent anthracite coal.

(3) Iron oxide.

FIG. 2.—FERROSILICON FURNACE AT GIFFRE WORKS.

The sandstone rock, utilized as the source of silica, is of a greyish color, owing to the fact that it is impregnated with about 8.5 per cent of bitumen; in addition to which it contains small quantities of other volatile constituents, including traces of nitrogenous matter. On being heated, it yields a whiteish grey, extremely friable material, small pieces of which can be readily broken down in the fingers to a fine powder, which consists of pure silica to the extent of 99 per cent.

The pitch is obtained as a by-product in the manufacture of coalite. This is used with addition of 10 per cent of its weight of anthracite coal. They would prefer to employ pitch-coke, but this from the electrical point of view would form too good a conductor.

The iron oxide consists of Fe_2O_3 precipitated in the form of a fine powder, which owing to its purplish color is useless as a paint, and so is obtainable at a much lower price than the red oxide. The average of several analyses indicates the presence of 95.4 per cent of pure iron oxide.

In making up the charge, the proper proportions of silica and iron oxide are mixed with the anthracite coal; and the whole bound together into a mass with pitch. After mixing, the volatile hydro-carbons in the pitch are driven off by heating, as the result of which the charge is left in a compact mass, which is broken up into small lumps of a suitable size for use in the furnace.

No flux is employed. The object of using pitch is the double one of ensuring that the carbon should be intimately mixed with the iron-oxide and silica, and of making the whole mass a conductor of high resistance

[2] There seems to be little difficulty in the French plants in obtaining, at a reasonable cost, quartzite or other comparatively pure form of silica, but it generally contains an appreciable amount of calcium phosphate as impurity. This is of importance in connection with the production of dangerous gases from ferro-silicon, as will be shown later on.

Furnace Construction.

Resistance furnaces are used throughout for the manufacture of ferrosilicon.

The furnaces which Dr. Copeman saw in operation at the different works visited were, for the most part, of very similar character and size, the main differences being in the external shape—usually circular (as are all internally), but in one or two instances square in section—in the size and shape of the upper electrode and in the mechanical means adopted for raising and lowering and usually for water-cooling this electrode. The furnaces at the Bozel works are provided with two tapping holes, from which the molten ferrosilicon is run alternately, whereas in the case of the furnaces at other works, one tap hole only is provided.

The furnaces are built up of fire-brick, sometimes lined with "a composition of graphite somewhat similar to that employed in the manufacture of the electrodes," and the exterior of the furnace is usually braced with iron "stays" or completely enclosed in a circular or square iron casing made up of sections bolted together.

Figs. 1, 2 and 3 show different designs of ferrosilicon furnaces. Fig. 1 is a drawing to scale of the usual type of furnace, Fig. 2 is a view of the furnace used at the Giffre works in St. Jeoire, Harte-Savoie, and Fig. 3 shows the design of the Keller furnace.

Except in the case of the furnace specially designed by Keller, of which an outline illustration is given in Fig. 3, the second lower electrode is placed beneath the floor of the furnace, its upper surface being flush with the level of the furnace hearth. At the Giffre works, for instance, this lower electrode is formed in situ by ramming into an open space in the floor of the furnace a mixture formed by heating together powdered retort graphite and coal tar.

The power employed in each furnace ranges from 250 or 300 hp to as much, in some cases, as 750 or 800 hp.

The type of furnace used in the Keller-Leleux works is such as to obviate the use of the furnace hearth as a conductor. Further to ensure the continuity of working, several electrodes are placed in parallel in the system, any one of which is renewable without the necessity of stopping or varying the working of the furnace as a whole. Keller's furnace, constructed on these principles, contains at least two groups of two electrodes each, the two latter being arranged in parallel, and the two groups themselves in series. These four electrodes of equal capacity project through refractory walls, and each electrode is provided with independent mechanism by which it can be raised or lowered at will. The intensity is equalized in each focus of the same group by means of readings taken with an ammeter, and the pressure of the group is controlled by means of two voltmeters inserted respectively between the furnace hearth and each group.

The voltage and amperage employed (the former especially) varied somewhat widely at the different establishments inspected, ranging from 40 to 75 volts, and from 10,500 to 15,000 amperes, respectively.

When once charged and started, the furnaces are run continuously for an average period of a couple of years, although, as stated at the Keller-Leleux works, the period may be extended to as much as four years. While running, the furnaces are tapped at intervals of 1 to 2 hours, the men working in shifts so that the furnaces can be attended to by night as well as by day, fresh amounts of charge being shoveled on the top of the furnace around the upper electrode as often as is rendered necessary by the melting and consequent shrinkage of the underlying mass.

Proportioning of Charge.

The charge is generally made up in proportions according to the formula

$$SiO_2 + 2C = Si + 2Co.$$

so that for every 60 parts by weight of pure silica 24 parts of pure carbon are used. But the making up of a charge is complicated by the fact that each of the three constituents, iron, quartzite and anthracite, notably the anthracite, will certainly contain impurities, the amount of which in each case has, therefore, to be determined and allowed for in the calculation.

This is illustrated by a concrete example, given by M. Barut, administrator of the Giffre Works at St. Jeoire, Haute-Savoie.

The composition of the anthracite coal in use at the time of Dr. Copeman's visit was "roughly" as follows:

Carbon ..... 82 per cent.
Ashes ..... 10 per cent.
Moisture and gases..... 8 per cent.

while analysis of the quartzite showed that it contained about 96 per cent SiO_2.

"Taking, then, the combining weights represented by the equation already referred to, viz:

$$SiO_2 + 2C = Si + 2CO$$
$$60 \quad 24 \quad 28 \quad 56$$

and noting that 24 kilograms of carbon will be represented by 30 kilograms of 80 per cent anthracite, the required proportion of silica to coal will be the simple one of 2:1, on which, however, allowance has to be made for the fact that 100 parts by

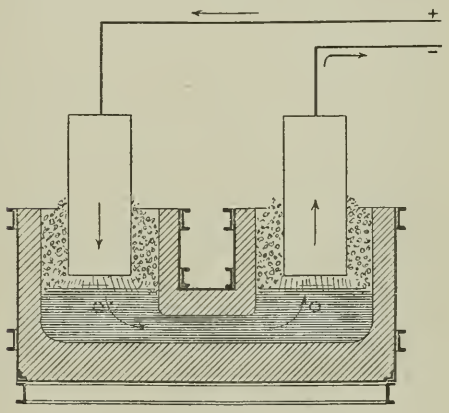


FIG. 3.—KELLER FURNACE.

weight of quartzite contain 96 parts only of silica, or about 144 kg of silica in 150 kg of quartz.

"As regards the source of the iron also, some allowance is of course necessary, seeing that the steel turnings employed will contain a certain amount of iron oxide in addition to small amounts of impurities.

"Taking these different factors into consideration, the respective weights of the several ingredients required to make up a charge for the production of, say, a 50 per cent ferrosilicon, will be as follows:

Quartz..... 150 kg (= about 144 kgms SiO_2).
Anthracite..... 72 " (= about 66 kgms Si).
Steel turnings..... 55 "

277 kg

"Dealing with these quantities:

"Sixty-six kilos silicon + 55 kilos steel turnings should theoretically yield 121 kg. all the carbon of the anthracite being given of as CO (carbon monoxide gas). But, as a matter of fact, in addition to the total disappearance of the carbon, a certain proportion of the silicon and iron (about 11 kg on the quantities here dealt with) is also not accounted for in the final product, being presumably lost through incorporation with the slag. Consequently, the amount of alloy actually obtained from the above charge of 277 kg will be 110 kg only of 50 per cent ferrosilicon.

"In the event of a lesser or greater silicon percentage being required, the weight of steel turnings in the charge must be increased or diminished accordingly."

Process of Manufacture.

In practice the requisite amounts of the three ingredients of the charge are weighed out separately, the quartz and anthracite coal having previously been reduced to small fragments of a size varying roughly from that of a filbert to that of a walnut. After weighing, the different materials are thrown together into a heap on the floor of the furnace house where they are mixed as thoroughly as possible with a shovel. The resulting rough mixture is then thrown onto the top of the furnace a shovelful at a time, care being taken to heap it up around the vertical electrode or electrodes, as the case may be.

To receive the molten material when the furnace is tapped, a shallow bed of sand is prepared, either on the floor or, as was the case at one of the manufactories visited, in an iron box on wheels, which accordingly can be removed as soon as the tappings shall have to come to an end. Usually, however, immediately the flow of white-hot liquid has ceased, the surface of the resulting pool of molten alloy is skimmed to remove the layer of slag on the surface, a short, thick log of green wood fixed crosswise on the end of a long iron bar being employed for this purpose at the Keller-Leleux works.

As soon as the cake or slab of ferrosilicon has solidified, it is broken up with heavy iron hammers into comparatively small pieces, which in the case of the 50 per cent grade, at any rate, it has recently become the custom, at several of the works Dr. Copeman visited, to treat, while still hot, by immersing them in a bath of kerosene oil or melted crude paraffin.

This "pickling" treatment has been adopted on the assumption that by thus preventing contact with the air by means of a coating of oil or paraffin, it would prove possible to obviate the well-known tendency of certain samples of 50 per cent ferrosilicon to disintegrate spontaneously, and thus, incidentally, to prevent evolution of noxious gases, mainly phosphoretted and arseniuretted hydrogen, which under certain circumstances appear to be associated in specially large quantity with this particular grade of electrically produced ferrosilicon.

But as regards the prevention of the process of disintegration, the precise causation of which, by the way, is still to seek, the kerosene or paraffin bath treatment has not proved successful; while the fumes given off when ferrosilicon, which has been "coated" after this fashion, is subsequently heated, as appears to be considered before being utilized for its specific action on addition to a bath of molten steel or iron, are most objectionable, owing to the crude material usually employed. In addition, the heating of the alloy thus rendered necessary before use at ironworks must obviously promote in special degree the evolution of poisonous gases, and thus prove a special source of danger to workmen engaged in handling the material, and to whom this preliminary treating of the alloy is entrusted

In view, therefore, of the complete futility of this process of "pickling" for the intended purpose, coupled with the fact that it may actually involve, later on, an added source of danger to workers in the iron foundries, it is certainly desirable that the method should be abandoned.

Evolution of Gases from Ferrosilicons.

Concerning the evolution of obnoxious or dangerous gases from ferrosilicon under the influence of moist air, especially when accompanied by friction, Dr. H. Wilson Hake states in his report that the following gases have been alleged to be evolved: Phosphoretted hydrogen (PH_3), arseniuretted hydrogen (AsH_3), siliciuretted hydrogen (SiH_4), acetylene (C_2H_2), and hydrogen. In the numerous samples which he has examined, he has not found, however, any acetylene or siliciuretted hydrogen, and he does not believe they are evolved. All his evidence seems to show that the poisonous emanations evolved from ferrosilicon by the action of water consist mainly of phosphoretted hydrogen; sometimes alone, but for the most part accompanied by varying proportions of arseniuretted hydrogen.

The formation of these gases is due to the presence of small quantities of certain impurities in the coal, iron and quartz, used as starting materials; and in this respect the most important impurities are calcium phosphate and arsenic.

Calcium phosphate $Ca_3(PO_4)_2$ one of the impurities present in coal and in quartz, which in itself is a perfectly harmless salt, insoluble in water and widely diffused in nature, is responsible for the production of a dangerous compound by reduction in the electric furnace, in the presence of carbon, to calcium phosphide (Ca_3P_2).

This calcium phosphide remains in the ferrosilicon, and in contact with water or moist air is decomposed with evolution of phosphoretted hydrogen (PH_3), with an intensely poisonous character.

Arsenic again, an element closely allied with phosphorus in its properties, is another impurity liable to be present in various combinations in coal and in iron, and this element also finds its way into the ferrosilicon apparently as calcium arsenide. Calcium arsenide is also decomposed by water or moist air, evolving arseniuretted hydrogen (AsH_3), a gas scarcely, if at all, less poisonous than phosphoretted hydrogen.

Specific Gravity of Ferrosilicons.

The variations of hardness and of specific gravity of alloys containing different percentages of silicon are quite characteristic and may be used as a rough indication of the approximate composition.

The specific gravity decreases as the percentage of silicon increases, which obviously follows from the fact that the specific gravity of pure iron is 7.8 while that of pure silicon is 2.49.

Fig. 4 gives the results of Dr. Hake's determinations of the specific gravity of various samples of commercial ferrosilicon in comparison with the theoretical curve (the thick line in

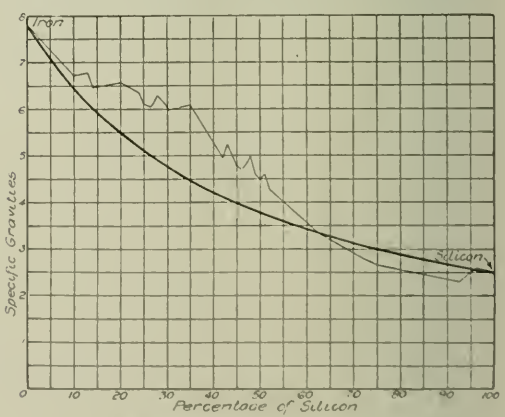


FIG. 4.—SPECIFIC GRAVITY OF FERROSILICON.

Fig. 4) representing the calculated specific gravities for all percentages on the assumption that mixtures of iron and silicon are formed without change of volume.

A comparison of the two curves demonstrates, what is indeed already well known, that such is not the case and the specific gravities are partly above and partly below the theoretical curve, showing that contraction of volume has occurred in the alloys formed up to 60 per cent silicon and expansion of volume from this to 96 per cent silicon, presumably owing to the presence of definite silicides of iron in the alloys.

Microstructure.

Mr. Samuel R. Bennett's report gives further confirmation of the general character of the specific gravity curve and also discusses the microstructure of ferrosilicons. Fig. 5 is Guertler and Tammann's diagram. Mr. Bennett gives some interesting microphotographs of various ferrosilicons and then sums up his conclusions as follows:

The various percentage grades of ferrosilicon are substantially binary alloys.

Two compounds, Fe_2Si and FeSi, exist; the former corresponding to 20 per cent Si, and the latter to 33.3 per cent Si, by weight. Microstructure also indicates the existence of two

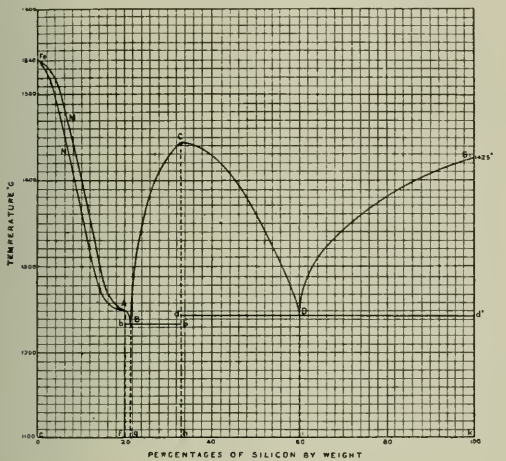


FIG. 5.—FERROSILICON DIAGRAM.

eutectics, one consisting of Fe Si and FeSi, corresponding to 21.6 per cent silicon approximately, and the other consisting of FeSi and Si at 60 per cent silicon.

Investigation of the specific volumes of various grades of alloys does not indicate the presence of other compounds.

Up to 20 per cent Si the alloys consist of solid solutions of Fe and Fe_2Si which are hard, firm masses giving off little or no gas. Alloys from 20 per cent Si to 21.6 per cent Si consist of primary crystals of Fe_2Si in ground of eutectic composed of Fe_2Si and FeSi; these alloys begin to get more brittle than the lower grades. From 21.6 per cent Si to 33.3 per cent Si the structure shows FeSi surrounded by eutectic Fe_2S and FeSi; from 33.3 per cent to 60 per cent Si there are crystals of FeSi in eutectic FeSi + Si; and above 60 per cent Si crystals of Si in field of eutectic FeSi and Si.

The amount of gases given off by different grades of ferrosilicon depends not only on the impurities present in the materials used in the electric furnaces, but also in the physical properties of the alloys, in consequence of which, in certain cases, access of moisture is afforded to a much larger extent of surface.

Hardness.

By the term "hardness" Dr. Hake implies the resistance offered to mechanical disintegration, or in other words, the varying difficulty experienced in breaking up and powdering the samples. To give two extreme instances, a sample with 26.5 per cent Si required a steel ore-crusher to break it up before it could be powdered, while a sample with 50 per cent Si could be crushed between the fingers.

Up to 30 per cent silicon content, the samples are more or less refractory without exception. There is an absence of any tendency on the part of these grades of ferrosilicon to disintegrate spontaneously.

Similarly, from 70 to 96 per cent where the silicon is in excess, although the samples are more or less brittle, they are not so easy to break up and to reduce to powder as those containing between 42 and 60 per cent silicon, and they also do not show a tendency to disintegrate spontaneously.

On the other hand, the samples containing from 42 to 60 per cent show a distinct tendency to spontaneous disintegration. They crumble on keeping, and some of them actually fall to powder after the lapse of a few weeks or months. This spontaneous disintegration is usually accompanied by a very noticeable evolution of evil-smelling and poisonous gases.

It has been alleged that the same phenomenon is apt to occur with 33 to 35 per cent grades, but Dr. Hake has no personal experience of the fact.

Transport and Storage of Ferrosilicon.

In view of the accidents which had happened, four of the large ferro-alloy manufacturers of France (Giffre, Girod, Keller-Leleux, La Volta) formed a special committee which came to the conclusion to propose "to give up making dangerous grades completely, which now are known to be those ranging from 30 to 40 per cent, and from 47 to 65 per cent."

"By discarding these specially dangerous grades and ceasing the manufacture of 30 to 40 per cent and 47 to 65 per cent ferrosilicon, there would remain nothing but perfectly compact alloys as free from danger to transport and to warehouse as ordinary pieces of cast iron."

These recommendations have been accepted by eight other French manufacturers, so that in all 12 French firms have agreed upon the above recommendations.

Dr. Copeman for the British Government Board goes even further in his recommendations. He is of the opinion that "the conclusion is justified that the manufacture of ferrosilicon should, at present, be restricted to the production of percentage grades of 70 per cent and upward at one end of the scale; and, at the opposite extremity, up to and including 30 per cent, on the supposition that under certain circumstances the employment of a comparatively low grade may be desirable or necessary. . . .

"But as, pending international agreement on the question, intermediate percentages of ferrosilicon will doubtless continue to be manufactured and sold, the issue, by the Board of Trade, of special regulations will be necessary in order to obviate, so far as may be possible, chance of further accidents during the transport of this substance."

The following regulations are suggested by Dr. Copeman:

"1. Ferrosilicon should not be sent out from the works immediately after manufacture, but, after being broken up into pieces of the size in which it is usually sold, should be stored under cover, but exposed to the air as completely as possible for at least a month before being dispatched from the works.

"2. Manufacturers should be required to mark in bold letters each barrel or other parcel of ferrosilicon with the name and percentage grade (certified by chemical analysis) of the material; the name of the works where it is produced; the date of manufacture, and date of despatch.

"3. The carriage of ferrosilicon on vessels carrying passengers should be prohibited. When carried on cargo boats, it should, if circumstances permit, be stored on deck. If it be considered necessary to store it elsewhere, the place of storage should be capable of being adequately ventilated, and such place of storage should be cut off by airtight bulkheads from the quarters occupied by the crew of the vessel.

"4. This regulation should apply to the transport of ferrosilicon on river or canal barges as well as on sea-going vessels.

"5. Storage places at docks or at works where ferrosilicon is used should have provision for free access of air, and should be situated at a distance from workrooms, messrooms, offices, etc.

Elsewhere in his report, Dr. Copeman makes the following remarks, which may be mentioned in this connection: "So far

as I am aware ferrosilicon is no longer packed in closed iron drums, so that, to this extent, likelihood of explosion is avoided, but as in transport, even when packed in wooden barrels, a considerable amount of friction must inevitably occur between the rough lumps, in which form the ferrosilicon is ordinarily sent out from the works, I would offer the suggestion that, in the course of packing the alloy into barrels, all interstices might usefully be filled in with shavings, wisps of hay, or other similarly cheap material."

A brief comment on the situation of the ferrosilicon industry in this country will be found on our editorial pages.

On the Preponderance of Nullovalent Matter.

By Dr. Gustavus Hinrichs.

For more than a thousand years chemists of all schools and creeds have experimented on all sorts of ponderable matter under all conditions conceivable, without ever meeting any form of matter which resisted all means of producing chemical changes.

Chemists have, therefore, concluded that all ponderable matter is chemically active, and can, with greater or less readiness, be subjected to entire cycles of chemical transformations, producing more and more complex substances in ascending cycles and obtaining finally, by descending cycles, ponderable matter in the simplest form known, namely, the chemical elements.

Hence chemists have, during the last two centuries at least, held that all ponderable matter is chemically active, and that the chemical elements are simple and irreducible substances.

Two New Fundamental Conclusions.—For more than half a century I have recognized the latter conclusion to be erroneous and for more than a third of a century I have found that the first conclusion cannot be true, and that on the contrary we shall have to admit:

I. Ponderable matter is not chemically active except under very special forms, affecting at most a hundredth of all matter.

II. Matter is one; hence the chemical elements are not simple substances, but combinations of several degrees of one single substance, the Pantogen.

This latter conclusion I have tried to demonstrate in my publications for now more than half a century.

As to the first conclusion, it is known since 1894 that there exist small amounts of a few nullovalent chemical elements. This is the argon group, or argonoids, comprising: Helium, neon, argon, krypton and xenon. The so-called emanation of radium is also nullovalent; but is of an entirely different order, as we expect to show shortly.

Still it remains generally understood that these nullovalent elements are exceedingly rare, though they seem to be widely diffused.

Accordingly, the exception from the old rule, that all ponderable matter is chemically active, appears to be quantitatively of little importance, though sufficient to prove that the old conclusion is not absolutely correct.

So far as we ourselves are concerned, the discovery of nullovalent elements had been expected for more than twenty years; for we have published it implicitly in our Molecular Mechanics, Davenport and New York, 1874; see section XVIII: The Element Atom an Atom Crystal, pp. 178-182.

The Nullovalent Elements in 1874.—In the work of 1874 specified, the four elements Fl, O, N, C are (p. 180-181) compared with the simple organic radicals, CH_3, CH_2, CH, and the tetravalent C, as derived from the saturated (nullovalent) methane CH_4, according to their atomic weight a and their valence n

"If we generalize this, we can express the atomic weight a of these elements by the formula

$$a = 2(10-n) = 20-2n$$

Valence	$n =$	1	2	3	4
Calc. at weight	a	18	16	14	12
Obs. at weight		Fl—1	O	N	C"

"502.—Now, this corresponds exactly to the organic radicals—namely, if we start from the saturated methane, $CH_4 = 16$, we have $a = 16 - n$, namely,

Atomicity	$n =$	1	2	3	4
Atomic weight	$a =$	15	14	13	12
Radicals		CH_3	CH_2	CH	C

"It is evident that the graphical symbols (Fig. 1, line F, Plate I) might completely represent the above elements, if the *dot* stands for two—i. e., four pantogen atoms * * * while the *cross* stands for twelve, some aggregation of pantogen atoms in the shape of a square prism for one end of which one of the four corners has been removed. It will be seen that this not only accounts for the atomic weight, but also for the atomicity of these most important elements."

In paragraph 503 of the same work, the probable structure of the sulphoids (O, S, Se, Te) is given; in 506 that of the Kaloids (Li, Na, Ka, Rb, Cs), while in 504 the increase 16 is defined, and 505 gives the general law of the addition by 3 times, 3 times, 2 times, 3 times, in the three highest members of each genus of elements and the explanation of the fact exactly as we have given it in our latest publication on this subject: The Proximate Constituents of the Chemical Elements, 1904. These references will be sufficient at this time and in this brief note.

The parallel between the elements and the organic radicals being complete, it is implied that a nullovalent element must correspond to the saturated CH_4, especially as the formula includes the elements of opposite (negative) valence, namely:

Valence	$n =$	—1	—2	—3	—4
Calc. atomic weight	$a =$	22	24	26	28
Observed		Na—1	Mg	Al—1	Si

These two groups of four each of chemical elements must therefore be connected by a nullovalent element of the atomic weight of 20. This element is known as Neon.

The case is much stronger still. For if we subtract from the constant 20 the common rise of 16 in the atomic weight of the consecutive groups, we obtain the corresponding formula

$$a = 4 - 2n$$

giving the values,

Valence	$n =$	—1	—2	—3	—4
At. wght. calc.	$a =$	6	8	10	12
At. wgt. observed		Li—1	Be—1	Bo—1	C

and requiring a nullovalent element for $n = 0$ of the atomic weight 4. This evidently is our helium.

These relations contained in my formulæ and figures printed in the work of 1874, as quoted, were quite familiar to me and their bearing was well understood; but it was absolutely impossible, at that time, to state my conclusion. In fact, I shall mention, my experience in 1908 proved that I was too hasty even then, when I was ready to furnish positive proof, as we shall see soon.

What I dared to proclaim in 1874 was more than the chemical public could take; only Berthelot requested me to furnish him certain specified portions of that work for publication in the *Annales de Chimie et Physique*. This was not done, because my time was in 1875 so fully taken up by the collection and study of the Amana Meteorites.

I will mention just one striking experience, which made me hesitate to give further early details of my investigations. My Atommechanik of 1867 was denounced in the *English Weekly Chemical Journal* by the editor, who had never seen the work. It is evident from his statements, he had only the English Resumé, and devoted most of his attention to the historic introduction which is no part of the work itself. Some critics are in every way more free to move the less they are loaded with knowledge of their own and the less they have seen of the work they want to condemn.

It is precisely this condition of things that has now come home to their readers and subscribers, who have been kept in the dark about what has been done, greatly to their disadvantage, even in their own investigations.

Upon the publication of the discovery of Argon in 1894, I

exposed the above relations in a special public lecture, and predicted that it would be but the first of that entire group of nullovalent chemical elements implicitly contained in my Molecular Mechanics of 1874.

The first time I explicitly mentioned my principle of the high preponderance of nullovalent matter over chemically active matter in Nature was in a letter of April 15, 1908, to a prominent chemist of Paris. That chemist never inquired further about it.

The Nullovalent Matter of the Chemically Active Elements.—The formula and figures published by me during the past half century show plainly that the element atoms of a group or Genus have a common active head, and differ only in the growth attained in their bodies, which growth goes by regular additions of very nearly $a:3a:3a:6a$, if the element belongs to what I term heavy or common elements. In the lighter elements, the additions are $a:a:3a:3a:6a$. In all cases, a is nearly 16.

These statements may be verified by an examination of my Principles of Chemistry and Molecular Mechanics of 1874; the Pli-Cacheté of 1891 (published, 1907); True Atomic Weights of 1894, and The Proximate Constituents of the Chemical Elements, of 1904. All my deductions of the physical, chemical and crystallographical properties of the elements are based upon this general structure. We can not go into further details here.

Since the chemical character of an element depends on the head only, all these additions are chemically inert or nullovalent. That is: in the Kaloids, the chemically active matter can not be more than what there is of it in an atom of Lithium. But by no means is all the weight of a lithium atom chemically active.

By my recent researches I have been able to prove that, in a hydrogen atom, the chemically active part is less than 1 per cent (*Comptes Rendus*, T. 147, pp. 797-800, 1908).

This percentage is not likely to be greater in an atom of lithium (as a matter of fact, it is demonstrated now to be much less than 1 per cent). Since the chemically active part in all kaloid atoms is the same, it cannot be more than 7/23, say 1/3 per cent in Na; 7/39 or 1/6 per cent in Ka; and not more than 1/20 per cent or half a pro mille in Cesium.

In other words: the total amount of chemically active matter is much less than 1 per cent of the total ponderable matter in existence. That is: chemically nullovalent matter is not an insignificant small fraction of the ponderable matter of the universe, but constitutes more than 99 per cent of all matter.

After waiting more than thirty years before speaking of this decidedly strange result, I found that I had been altogether too hasty. The details of this, my experience, are quite interesting to me and are likely to be so to others in a near future; but we have no place for them here and now.

Right here and just now the reader will possibly say that I have not proved my statements about the great predominance of chemically inert matter over the chemically active ponderable matter. I fully sympathize with the reader; for I know exactly what is the difficulty in his way, and will do all I can to clear the road.

First, the reader is in the position of a student listening to a demonstration of a geometrical proposition while not familiar with geometry; he cannot see the force of the demonstration.

Now, in that sort of a case, I have found it quite useful to show by some simple measurements or weighings, on models, that the proposition holds good in such special cases.

I shall do that very thing in a concluding article next month, in which I shall determine the weight of the amount of active matter in the atom of silver and of other chemical elements, which alone is characteristic of its chemical activity.

In the meantime I would respectfully request the reader of the present note to lay this paper aside, and to read it very carefully once every week till the next issue comes to hand; I dare say that the reader will gradually feel at least some of the force of the reasoning here employed.

By this process the reader will also have prepared himself for the more ready understanding of the mode of calculation that will be used in the closing paper.

Just think for a moment on the difficulties in the way of understanding the form of the earth and the position of the antipodes—a thousand years ago. In science we have to wrestle with such apparent difficulties at every big step we take forward into greater light.

St. Louis, Mo.

Pulp Classification and Tube Mill Efficiency.

In an interesting paper presented to the Mexican Institute of Mining and Metallurgy (Informes y Memorias del Instituto Mexicano de Minas y Metalurgia, Vol. 1, No. 3, December, 1909), Mr. V. B. Sherrod reports on some features of the extended research work which has been done at the Guerrero Mill, Pachuca, Mexico, particularly in the direction of pulp classification and in ascertaining the efficiency of tube-mill work under varying conditions.

The flow-sheet on page 140 illustrates clearly the sequence of the various operations and is self-explanatory.

Table I, "Screen Analysis of Samples," shows the average screen, or grading analyses of several sets of samples taken during the month of July, 1909, at the Guerrero Mill, the various products at each step being carefully sampled. A study of the figures will show some very interesting features.

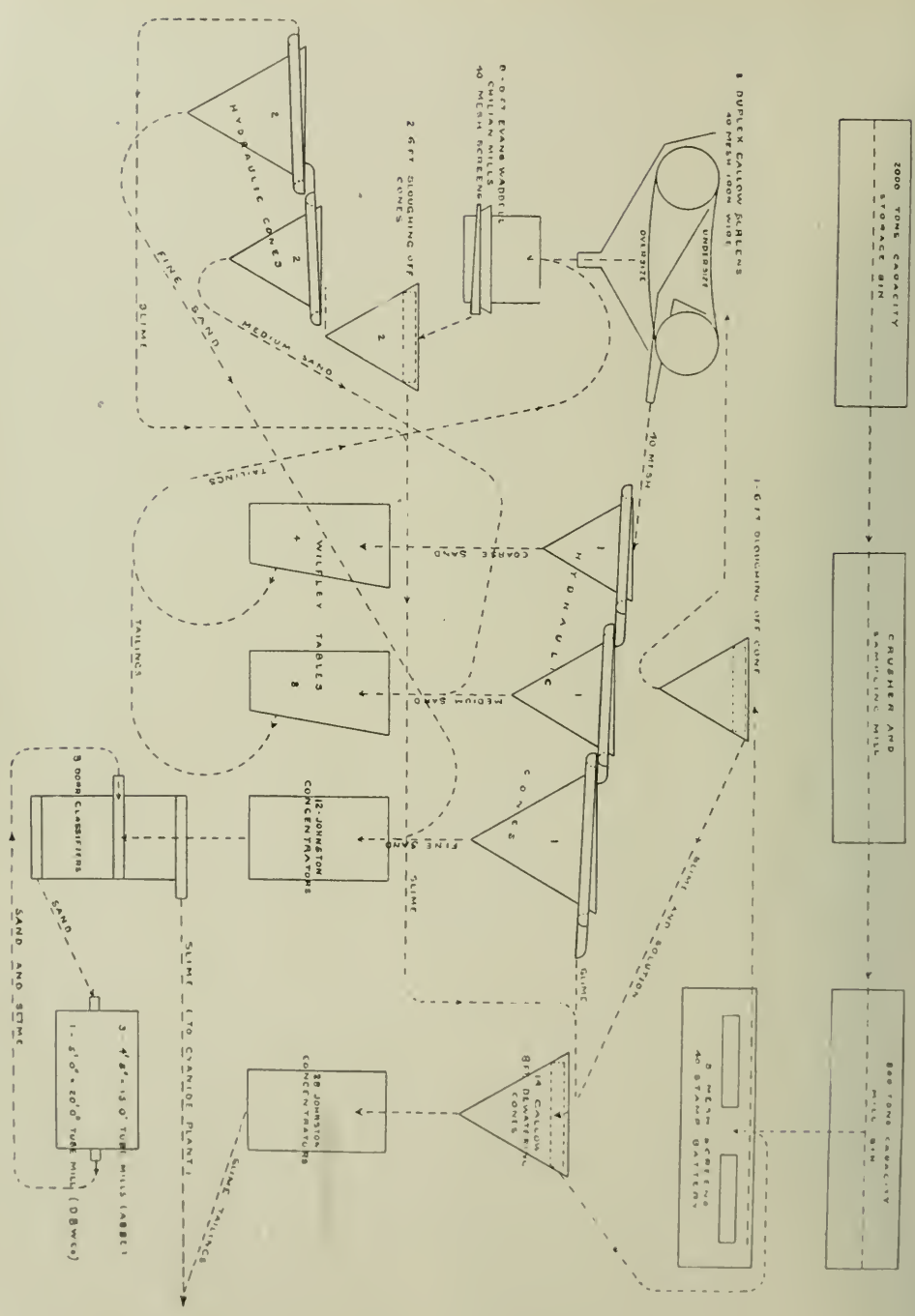
Special attention is called by Mr. Sherrod to the work of the Callow screens, which at 100 ft. belt travel per minute (a very high speed) when equipped with 30-mesh, No. 33 steel wire (the purpose being to separate as undersize all material that can be made to pass a 40-mesh laboratory screen) gives an oversize 92 per cent of which remains on a 40-mesh screen and 96.5 per cent remains on 60-mesh screen. At the same time an undersize is sent to the hydraulic cones, 9½ per cent of which passes a 40-mesh, said product containing practically all of the liberated mineral. The effect of this system is seen in the enormously wide streak of mineral on the first three Wilfley tables, which really receive as feed a product that has once been concentrated by the Callow screens and once by the hydraulic cones following these.

Another feature of interest to concentrator men is the analysis of the product of the Chilean mills, 4 per cent of which remains on a 40-mesh screen and 32.4 per cent of which passes a 200-mesh screen notwithstanding that a 30-mesh "ton-cap" screen is used in the mills. The discharge through these screens is kept up to the maximum, a heavy splash being used, but all material coarser than 60-mesh is, so far as possible, cut out in the first hydraulic classifiers following these mills and sent back to the Wilfley tables, where it joins the oversize from the Callow screens and returns again to the Chilean mills.

Another feature of the grinding at Guerrero Mill is the fineness of the tube-mill feed, only 7 per cent of which remains on a 60 screen. This is, of course, absolutely against high tube-mill tonnage, but is justified by the necessity for the most careful possible concentration, which means on these ores pretty fine grinding (in stages) and careful elimination after each step and before the tube mills get a chance to slime the mineral. The apparently low efficiency of the tube mills receiving such feed (in comparison with, for instance, the Loreto Mill of the same company, in which a much simpler flow-sheet is in use) is principally due to the fact that all of the easily slimed portion of the ore has been made to pass a 200-mesh screen during its repeated passage through the Waddell mills, leaving for the tube mills nothing but the most refractory portion to grind.

Tube Mill Operations, Screen Analysis of Feed and Discharge.

The Table II is made up from sampling done at Loreto Mill on various dates, and does not furnish absolute comparison with the performance of the Waddell mills above mentioned, on account of the difference in grinding systems employed and also


2000 TONS CAPACITY STORAGE BIN
CRUSHER AND SAMPLING MILL
800 TONS CAPACITY MILL BIN
8 DUPLEX CALLOW & ALLENS 40 MESH 100M WIDE
OVERSIZE
UNDERSIZE
6'-0" EVANS WARFIELD CHILIAN MILLS 40 MESH SCREENS
2 6FT SLOUGHING OFF CONES
HYDRAULIC CONES
FINE SAND
MEDIUM SAND
SLIME
TAILINGS
WILFLEY TABLES
12-JOHNSTON CONCENTRATORS
5 DORR CLASSIFIERS
SAND
SLIME TO CYANIDE PLANT
SAND AND SLIME
3 4'6"-15'0" TUBE MILLS (ABBE)
1-5'0"-20'0" TUBE MILL (DORR)
1-6 FT SLOUGHING OFF CONE
10 MESH
HYDRAULIC CONES
SLIME
SLIME TO VATS
8 MESH SCREENS 40 STAMP BATTERY
14 CALLOW BIN DEWATERING CONES
28 JOHNSTON CONCENTRATORS
SLIME TAILINGS

FLOW-SHEET.

TABLE I—SCREEN ANALYSIS OF SAMPLES

		Per Cent. Solids	Percentage on Screens									
			+10	+20	+40	+60	+80	+100	+120	+150	+200	−200
Battery discharge		20.74	36.0	20.0	16.4	6.8	2.6	3.6	.5	2.0	1.8	10.3
Elevator No. 1, Sloughing-off-cone	Overflow	3.59	....	....	....	....	....	.5	.1	2.5	4.9	92.0
	Spigot	30.85	35.8	20.1	17.0	7.5	3.0	3.9	.8	2.5	2.0	7.4
Callow-Screens	Oversize	35.50	43.0	27.0	22.0	4.5	1.5	.5	....	.3	.1	1.1
	Undersize	11.92	....	....	8.8	23.7	12.0	14.7	2.0	7.0	6.0	25.8
Wilfleys, pulp from Callows	Feed	18.00	....	....	8.7	28.0	14.8	16.3	2.9	7.8	5.5	16.0
	Discharge	9.33	....	....	8.6	28.1	15.0	16.2	2.8	7.5	5.9	15.9
Wilfleys, Chilean Return	Feed	20.35	....	....	11.5	44.0	16.3	12.3	2.5	4.2	2.5	6.7
	Discharge	11.37	....	....	12.0	44.0	16.0	12.0	2.5	4.1	2.6	6.8
Wilfley Tables, General	Feed	18.84	....	....	11.6	37.0	15.0	14.7	1.6	5.1	3.9	11.0
	Discharge	11.87	....	....	11.7	36.9	15.2	14.8	1.5	5.0	4.0	10.9
Waddell Mills	Feed	22.46	19.5	11.4	16.0	24.5	9.6	8.2	1.0	4.0	2.2	3.6
	Discharge	22.60	....	....	4.0	23.8	12.7	14.5	1.2	6.8	4.6	32.4
Elevator No. 3, Sloughing-off-cone	Overflow	4.41	....	....	....	.2	.4	.5	.1	.8	1.5	96.5
	Spigot	25.93	....	....	6.4	28.5	13.2	14.5	2.0	8.2	4.5	22.7
Elevator No. 4, Sloughing-off-cone	Overflow	4.49	....	....	....	.3	.5	.4	.1	1.5	3.7	93.5
	Spigot	27.88	....	....	7.5	29.6	13.0	15.9	2.5	8.8	3.5	19.2
60-inch, No. 1 Classifier	Overflow	5.59	....	....	....	.8	.9	1.0	.5	2.6	3.3	90.9
	Spigot	17.90	....	....	.6	12.0	10.5	18.7	4.8	10.0	8.0	35.4
60-inch, No. 2 Classifier	Overflow	6.09	....	....	....	.9	1.0	1.1	.5	2.5	4.0	90.0
	Spigot	24.02	....	....	.5	14.3	12.9	21.4	2.7	12.9	9.8	25.5
Johnstone Slime Tables	Feed	13.52	....	....	....	.1	.2	.4	.1	2.5	3.2	93.5
	Discharge	9.53	....	....	....	.1	.3	.3	.1	2.6	3.2	93.4
Johnstone Fine Sand Tables	Feed	60.00	....	....	.4	7.7	10.7	27.8	5.0	20.8	12.8	14.8
	Discharge	14.21	....	....	....	2.3	5.5	20.6	4.7	20.0	15.0	31.9
Dorr Classifiers	Feed	58.64	....	....	.5	7.5	11.9	26.5	4.4	22.0	12.5	14.7
	Discharge	59.34	....	....	....	3.7	6.7	23.2	3.1	21.2	14.2	27.9
Tubemill No. 1, 4'6,, by 15' Abbé	Feed	59.84	....	....	.5	8.5	10.0	26.0	4.5	24.1	12.8	13.6
	Discharge	60.87	....	....	....	3.1	7.2	22.7	4.7	23.0	12.5	26.8
Tubemill No. 2, 4'6,, by 15' Abbé	Slime to Tanks	10.60	....	....	....	....	....	2.0	.2	6.6	8.1	83.1
	Overflow	4.49	....	....	....	.2	.4	.5	.1	.8	1.5	96.5
	Spigot	25.93	....	....	6.4	28.5	13.2	14.5	2.0	8.2	4.5	22.7
Tubemill No. 3, 4'6,, by 15' Abbé	Overflow	4.49	....	....	....	.3	.5	.4	.1	1.5	3.7	93.5
	Spigot	27.88	....	....	7.5	29.6	13.0	15.9	2.5	8.8	3.5	19.2
Tubemill No. 4, 5' by,, 20' D. E. W. C.	Overflow	5.59	....	....	....	.8	.9	1.0	.5	2.6	3.3	90.9
	Spigot	17.90	....	....	.6	12.0	10.5	18.7	4.8	10.0	8.0	35.4
General	Slime to Tanks	10.60	....	....	....	....	....	2.0	.2	6.6	8.1	83.1

because the ores fed to these mills are much more friable than those ground in the Guerrero mill. The object of the test was to determine the most efficient feed for the tube mills and to furnish comparison between the tube mills differing from each other in make and leading dimensions, especially diameter.

The samples from which the analyses were obtained were taken in sets, and for obvious reasons, such as daily variations in the character of the ore, etc., too close comparisons should not be made between analyses of samples of different sets. Some sets of samples must also be eliminated from any comparisons whatever, because conditions changed so frequently during the two or three hours consumed by the tests that determination of kilos per minute of solids fed to mill, percentage of solids, etc., are not accurate. Such tests are indi-

TABLE III.—SCREEN TESTS OF WADDELL CHILIAN MILLS

OBJECT OF TEST—TO DETERMINE EFFECT ON EFFICIENCY OF VARIOUS FEEDS

Test	Date	Solids in Feed	Dry Ore Per Minute	Water Per Minute	P	+10	+20	+40	Ground Through 40		+60	Ground Through 60		+80	Ground Through 80	
		Per Ct.	Kilos	Kilos		Per Ct	Per Ct.	Per Ct.	Per Ct.	Kilos	Per Ct.	Per Ct.	Kilos	Per Ct.	Per Ct.	Kilos
1	7/3	13 11	24	159	35 ..	39	11 7	12 5	....	....	16 4	....	....	9 0	....	....
		....	....	....	....	....	....	7 0	56 2	13 5	18 6	54	12 9	13 5	49 5	11 8
2	"	27 7	56	146	41	35	14 0	19 0	....	....	14 6	....	....	7 0	....	....
		....	....	....	....	....	....	5 0	63 0	35 3	19 5	58 1	32 5	15 5	49 6	20 8
3	"	21 5	70	256	48	37 0	13 0	15 6	....	....	15 8	....	....	8 5	....	....
		....	....	....	....	....	....	6 0	59 6	41 7	19 0	56 4	39 48	14 0	50 9	35 6
4	"	24 4	62	192	44	36 0	12 9	18 8	....	....	15 6	....	....	8 3	....	....
		....	....	....	....	....	....	8 8	61 9	38 4	19 2	58 3	36 1	14 5	52 10	32 30
5	"	22 3	80	278	50	30 0	16 0	16 0	....	....	17 2	....	....	9 0	....	....
		....	....	....	....	....	....	8 6	53 4	42 7	18 4	52 2	41 8	14 2	47 2	37 8
6	"	9 9	34	310	35	35 3	17 5	15 0	....	....	13 1	....	....	7 0	....	....
		....	....	....	....	....	....	5 0	62 8	21 35	18 7	57 2	19 45	14 2	50 0	17 0
7a	"	13 6	40	254	38	32 0	10 0	15 6	....	....	20 2	....	....	9 8	....	....
		....	....	....	....	....	....	6 2	51 4	20 6	21 0	50 6	20 2	16 2	45 8	18 3

Test	+100	Ground Through 100		+120	Ground Through 120		+150	Ground Through 150		+200	Ground Through 200		−200	Ground Through 200		Kilos Through 200 Mesh O/ H.P. O/ Min
	Per Ct.	Per Ct.	Kilos	Per Ct.	Per Ct.	Kilos	Per Ct.	Per Ct.	Kilos	Per Ct.	Per Ct.	Kilos	Per Ct.	Per Ct.	Kilos	
1	4 4	....	....	0 5	....	....	2 0	....	....	1 4	....	....	2 9	....	....	
	12 2	41 7	10 00	2 2	40 0	9 6	7 10	34 9	8 37	8 3	28 0	6 7	30 8	27 9	6 69	0 19
	4 0	....	....	0 2	....	....	1 6	....	....	1 0	....	....	3 3	....	....	
2	11 5	42 1	23 6	2 0	40 3	22 6	6 9	35 0	19 6	7 2	28 8	16 1	32 0	28 7	16 1	0 39
	4 7	....	....	0 1	....	....	1 9	....	....	1 8	....	....	1 4	....	....	
3	12 5	43 1	30 2	2 1	41 1	28 8	7 5	35 5	24 8	7 9	29 4	20 6	30 6	29 2	20 4	0 42
	5 0	....	....	0 3	....	....	1 2	....	....	0 8	....	....	1 0	....	....	
4	12 0	45 1	27 9	2 4	43 0	26 7	7 0	37 2	23 1	7 8	30 2	18 7	31 0	30 0	18 6	0 42
	5 1	....	....	0 2	....	....	2 0	....	....	1 9	....	....	2 5	....	....	
5	13 2	39 1	31 3	2 6	36 7	29 4	7 0	31 7	25 4	6 6	27 0	21 6	28 9	26 4	21 1	0 42
	4 9	....	....	0 4	....	....	2 1	....	....	1 0	....	....	3 1	....	....	
6	12 0	43 9	14 9	2 3	42 0	14 2	7 0	37 1	12 6	6 8	31 3	10 6	33 2	30 1	10 2	0 29
	12 5	....	....	0 5	....	....	2 2	....	....	1 6	....	....	1 6	....	....	
7a	14 1	37 7	15 1	2 1	36 1	14 4	7 1	31 1	12 5	6 5	26 3	9 5	26 5	24 9	10 0	0 26

a This mill was equipped with wornout rings from other mills.

cated by marginal notes. Despite these disabilities, however, it was possible to collect much valuable data.

The determination of the most favorable feed for horse-power efficiency was naturally the first consideration, with special reference to the percentage of solids in the feed and the kilos of dry feed per minute for each different mill. A little study of the table leaves no doubt as to these points, except in the case of No. 4 mill (D. E. W. Company, 5 ft. x 20 ft.) wherein the value of these tests was practically destroyed by the variations of feed above referred to. Therefore, the tests on this mill are given for what they are worth, demonstrating as they do the horse-power efficiency of the mill does not increase with the quantity of dry feed up to an undetermined point.

Briefly, the conclusions which may be safely drawn from the series of tests are as follows:

1. That the grinding efficiency increases with the percentage of solids in the feed up to approximately 55 or 60 per cent.

2. That the grinding efficiency increases with the quantity of ore fed per minute to a point which varies with the dimensions of the mill, and that beyond this point capacity is not gained, and within reasonable limits there is no loss except that due to the return to the mill of an excess of oversize.

The limit of profitable feed varies with the character of the pulp, and it is only possible to determine this limit for any mill by special experiment with the ores for which the mill is intended.

At the Loreto mill the profitable feed for the 4-ft. x 18-ft. mill was apparently 80 kilos of solids per minute, or at the rate of 115 metrical tons per 24 hours, although heavier feed did not reduce the efficiency of the mill. The limit for the two 4-ft. 6-in. x 15-ft. mills was not proved, but is apparently somewhat lower than the correct feed for No. 1 and is probably due to the

TABLE II—SCREEN ANALYSES OF TUBE MILL FEED AND DISCHARGE

Mill Material		1909	Per Cent. Solids	K's Per Minute		Screen Analyses								K's per Min. Through 200 Screen	H. P.	Kilos per H. P. per Min. Through 200
				Solid	Solution	+40	+60	+80	+100	+120	+150	+200	−200			
1 Krupp: 4′ × 18′–31 R. P. M.	Feed	3 23	55.0			0 2	5.0	5 7	14 2	2 3	20 0	19 8	33 5			
	Discharge		55 3			0 2	0 6	5.5	2 0	11 7	15	6 3	63 1			
2 Abbé: 5′×15′–29 R. P. M.	Feed	"	60.3			0 3	5.2	6 6	14.0	2 0	20 7	19 1	30 3			
	Discharge		60.8				0.7	2.4	9.7	2.1	16 7	17 3	49.9			
3 Abbé: 4′×18′–31 R. P. M.	Feed	"	60.6			0.3	5.3	6.4	13.3	2 8	20 5	19 0	32 0			
	Discharge		60.8				0 6	2.1	8.3	2 0	16 0	16 6	52 5			
4 D. E. W. Co.: 5′×20′–26½ R. P. M.	Feed	"	58.9			0.3	5.2	6 2	14.4	2 5	18 5	20 0	32.5			
	Discharge		62.7				0 2	0 9	5 6	2 2	10 9	14 3	64 6			
1 El Oro Liner	Feed	3/27	56 7	65.9	50 4	0.2	5.0	7.5	19.0	4.0	20 8	19 8	23 5			
	Discharge		56.9				0 2	2.0	10 5	3 0	16 0	15 2	52 2	18 8	32 8	0 58
2 Silex Liner	Feed	"	64 8	100 0	54 4	0.2	5 5	7 5	18 8	5 0	20 0	18 0	25 0			
	Discharge		65 9				1 0	4 2	15.5	3.0	18 0	17 8	39 5	14 5	27 0	0 54
3 aSilex Liner	Feed	"	64 9	70.3	37 0	0.2	4 4	7 4	18 2	4 5	21 6	16 8	26 6			
	Discharge		65 0				1.0	3 2	14.0	4 0	18 0	14 5	44 5	12 6	27 0	0 47
4 bEl Oro Liner	Feed	"	64 9	64.9	35.1	0.3	4 5	7.0	17.8	4.0	21 5	17 0	27 6			
	Discharge		66.7				0 2	1.6	9 0	3 0	15 6	15 7	54 0	17 1	51 0	0 34
1	Feed	4/13	56 7	102.9	77.7	0.4	5 0	8 8	21 0	4.5	20 0	18 2	22 5			
	Discharge		58.9				0 8	4 0	14 0	4.3	18 5	15 5	42 7	22 8		0 70
2 c	Feed	"	62.8	115.5	68.3	0.4	4 8	10 0	21 2	4.2	21 3	19 0	19 0			
	Discharge		63.4				1 8	5 5	17 0	4.4	21 0	18 5	31 7	14 7		0 55
3	Feed	"	62.7	113 6	67 6	0 3	4 9	9 9	21.0	4 5	20 5	19 0	19 1			
	Discharge		63 0				1.5	4.5	16.5	4 4	20 0	18 0	34 0	16 9		0 62
4	Feed		62.4	103 8	62 6	0.4	5 0	9 9	21.1	4 4	20 5	18 9	19 7			
	Discharge		63 4				0 7	3 8	13.5	4.2	18.4	15 5	44 0	25 2		0 50
1	Feed	5/20	52 1	46 0	42 0	1 2	12 0	11.0	18.5	3.8	16 8	12 8	23 0			
	Discharge		52 8				0 2	1 5	8.2	2.0	14 8	13 8	58 0	16 1		0 49
1d	Feed	"	46 4	50 8	59 6	1 0	11 0	11.5	20.0	4 0	15 5	14 5	21 8			
	Discharge		47 8				0 3	2.0	10.2	2 0	15 0	14.0	55 0	16 9		0 52
1	Feed	"	37.1	50.4	83.6	1.0	10 9	12 5	20.4	3.8	17 0	13 2	20 4			
	Discharge		37.3				0 5	2 8	11.5	2.5	16 0	14 8	50 5	15 4		0 47
1	Feed	"	31 4	52.4	104 0	1.0	10 5	12 4	20.0	3.0	16 0	14 0	21 6			
	Discharge		31 5				0 6	3 0	11.8	2.6	16 8	15.0	48 7	14 2		0 43
2	Feed	4/17	63 9	82.8	46 8	0.5	7 4	10.8	20.0	3.7	20 5	14.6	22 0			
	Discharge		64 9				2 0	6 5	16.4	3 7	18 6	16 5	35 6	13 6		0 51
1 e	Feed	"	56 6	50 0	38 3	0.4	5 3	11.0	21.0	3.6	20 7	16 3	21 5			
	Discharge		56 9				0 2	2.0	8.7	1.7	17 0	16 5	53.0	15 7		0 48
1 f	Feed	"	49 3	32 0	33 0	0.6	6 5	10.2	20 5	3.2	20 7	16 0	22 0			
	Discharge		52 4				0.1	0 3	2.8	1.2	10 2	16 3	68 6	14 9		0 45
1	Feed	"	58 8	83 0	58 1	0.2	6 2	9 0	18.8	4 0	20 0	16 2	24 0			
	Discharge		59 7				0 4	3 0	11 5	3 0	17 0	18 2	46 0	19 9		0 61
1	Feed	"	61 9	131.6	81 0	0.2	5 3	8 5	19.5	4.4	19.2	19 0	23 0			
	Discharge		62 0				1 0	4 0	14 1	3 8	21 0	17 0	38 0	19 7		0 60
2	Feed	4 15	63 7	87 6	49 9	0.4	5 2	9 0	19 6	5.0	21 5	17 3	21 1			
	Discharge		64 6				1 6	5 8	15 0	2 4	21 8	17 5	35 0	12 2		0 45
4	Feed	"	64 6	68 4	37 5	0.2	0 0	3 8	19 0	4.7	20 0	18 8	22 0			
	Discharge		65 9				0 1	2 5	9 9	2 1	15 4	16 5	54 0	21 0		0 41
4 g	Feed	"	64 7	38 8	21 1	0 2	5 2	9 9	20 7	3 0	22 2	17 0	21 5			
	Discharge		65 2				0 2	1 2	7 8	1 5	15 0	14 5	59 5	14 7		0 2
4	Feed		64 9	110 8	59 9	0.3	4 5	7 5	18 2	4 5	20 5	17 5	26 8			
	Discharge		65 5				0 6	2 8	12 0	2 0	20 8	16 1	45 2	20 4		0 4
4	Feed		65 3	128 4	68 2	0 2	4 5	8 0	18 7	3 9	22 0	16 2	26 4			
	Discharge		66 5				0.6	3 1	3 2	3 4	20 0	17 2	42 0	20 0		0 4
1 h	Feed	9 16	60 5	141 0	92 0	0 4	10 5	15 0	22 0	4 4	20 0	12 0	12 6			
	Discharge		61 1				6 0	11 0	19 0	6 2	18 0	15 5	24 3	10 8		0 43
2	Feed		67 9	127 0	60 0	0 3	9 3	15 8	25 4	4 0	18 8	13 2	13 2			
	Discharge		67 6				8 0	12 5	20 9	5 0	19 2	12 5	21 9	11 0		0 41
3	Feed		67 1	128 4	61 0	0 5	12 0	14 0	21 5	5 0	20 2	14 4	13 4			
	Discharge		67 3				7 5	12 6	21 8	4 0	18 5	14 0	21 6	11 8		0 44
4	Feed		67 0	130 0	64 0	0 4	10 2	14 5	24 8	4 0	20 8	12 8	12 5			
	Discharge		68 2				5 0	11 5	19 3	2 0	20 0	11 5	31 0	22 8		0 45
4	Feed	9 21	61 3	130 0	82 0	0 5	10 5	17 0	22 0	5 2	18 0	14 0	12 8			
	Discharge						6 0	12 0	21 0	5 0	17 5	13 2	25 3	16 2		0 3
4 i	Feed		56 1	140 8	110 0	0 5	10 0	16 0	22 5	5 0	19 0	15 5	11 5			
	Discharge						6 5	13 0	18 3	5 1	18 0	14 2	24 9	17 9		0 35
4	Feed		55 6	162 0	130 0	0 7	13 0	17 1	22 0	5 0	20 0	13 0	10 2			
	Discharge						7 0	11 0	21 0	5 5	19 4	13 1	23 0	20 8		0 41
4	Feed		66 1	157 2	80 0	0 6	11 6	16 2	23 2	4 4	20 8	12 0	11 8			
	Discharge						6 5	11 9	20 0	4 2	17 2	13 5	11 7	23 4		0 4
4 j	Feed	3/25	67 1	79 0	41 0	1 0	13 5	15 3	19 1	5 0	20 5	12 0	13 6			
	Discharge		68 1				5 0	10 6	17 0	4 0	16 5	13 2	13 2	15 9		0 31
4	Feed		58 1	98 0	71 6	1 2	12 0	16 5	20 0	5 1	20 0	13 0	12 2			
	Discharge		58 9				5 3	11 0	18 2	4 0	17 3	13 0	31 2	18 6		0 365
4	Feed		52 8	113 1	95 0	1 0	13 0	15 5	20 5	4 9	19 4	13 9	12 7			
	Discharge		51 5				6 5	11 2	17 5	5 0	18 0	12 8	30 0	19 1		0 375

a Fed at less than capacity to determine the efficiency of mill. *b* Same as *a*. *c* Same as *a*. *d* Fed at less than capacity to determine effect of per cent. of solids in feed. *e* Fed at less than capacity to determine effect of various feeds. *f* Same as *e*. *g* Fed at less than capacity for same purpose as *e*. *h* Determinations of feed per minute vitiated by power troubles during test. *i* Power troubles during test. *j* Determination of feed not correct, power troubles

fact that they are 3 ft. shorter, which reduces the capacity somewhat, and they have a greater diameter, which tends toward a reduction of their horse-power efficiency.

This statement as to horse-power efficiency is not absolutely proven by the above tests, but in every case the efficiency has been lower in mills of greater diameter, except one in which the small mill did not have more than about 60 per cent of its most economical feed.

The 5-ft. x 20-ft. mill (which is mechanically the best built one) has shown in every comparative test a lower grinding efficiency than the smaller mills, but it would not be fair to look upon this deduction as final, owing to reasonable doubt as to the accuracy of determination of quantities.

Screen Tests on Waddell Chilean Mills.

These tests, shown in Table III, were made to determine the best feed for these mills, and it requires but a superficial examination to decide that the heaviest feed which can be taken care of by the discharge screen in use is the most economical one. In fact, based on the results of these tests, the entire launder system for distributing the feed for these mills is being replaced by a "distributing tub" system designed to do away with the inequalities of the launder system and to permit every mill to receive at all times as near as possible its maximum feed.

The mill efficiency in the above tests was computed in kilos per minute, ground to pass a 200-mesh wire screen, not as a definite standard, but as a convenient means of comparison for the different mills.

A Reichsgericht Decision on the Scope of the Patent Treaty Between the United States and Germany.

In our correspondence columns in this issue "Manufacturer" refers to a recent Reichsgericht decision of considerable importance with respect to the patent treaty between this country and Germany. The Reichsgericht is the German Supreme Court, and the decision in question was rendered in the patent litigation of the National Cash Register Company, of Berlin, defendants, now the National Cash Register Company, of Dayton, Ohio, appellants, against Schubert and Salzer, complainants and appellees, regarding withdrawing the German patents 119,260, and 119,155.

These patents, which refer to improvements in cash registers, were taken out in the name of the National Cash Register Company, a limited company, the place of business of which was Berlin.

On July 16, 1907, the complainant (Schubert and Salzer) moved that the patents should be withdrawn according to § 11, No. 1, of the German patent law, because at that time (more than three years after the publication of the granting of the patents) the inventions were not worked in Germany to a sufficient extent. All the essential parts of the cash registers protected by the patents were manufactured by the American parent company, the National Cash Register Company, of Dayton, Ohio, and were imported to Germany from this country.

The German Patent Office, in a decision of May 11, 1908, ordered the withdrawal of the patents according to the motion of the complainants, Schubert and Salzer.

Against this decision of the German Patent Office the National Cash Register Company, of Berlin, appealed. After the papers had been submitted to the Reichsgericht and after a date for the oral hearing had been arranged, the name of the firm owning the patents underwent a change. The firm name is now the National Cash Register Company, Ltd. This change was registered in the Patent Office and in the papers in this suit. Further, all four patents were assigned on May 25, 1909, by the National Cash Register Company, Ltd., in Berlin, to the National Cash Register Company, of Dayton, Ohio, which is the American parent company of the Berlin firm. Accordingly the ownership of the patents was also changed in the registers of the Patent Office to this American firm, and the assignment of these patents was published in the *German Imperial Gazette* of July 29, 1909.

Further, on Aug. 1, 1909, in No. 47 of the *Imperial Gazette*, the patent treaty between the German Empire and the United States concerning mutual industrial protection, which took effect on Feb. 23, 1909, was published.

The oral arguments of the attorneys, in accordance with the direction of the Reichsgericht, were restricted to the question whether this patent treaty should influence the decision in the suit concerning these two patents, and, if so, what kind of an influence the patent treaty should exercise.

The German Reichsgericht, in its decision rendered on Nov. 21, 1909, reversed the decision of the German Patent Office and denied the motion to withdraw patents 119,155 and 119,260.

In the decision of the Reichsgericht the question is first discussed whether the American firm should be admitted as the principal party in this suit for the reason that the American firm is now registered as owners of the patents in the Patent Office; or whether, in view of the opposition of the complainant, this should not be done. The Reichsgericht decided on the basis of a lengthy argument on the applicability of certain paragraphs in the German patent law and the German civil law that "the opposition of the complainant against the admission of the American company as defendant in this suit is to be disregarded, and it is decided that this latter firm is the main party entitled to defend the patents."

"As far as the subject-matter itself is concerned, the withdrawal of the patents, even after it had been justly decided by the Patent Office, based on § 11 of the patent law, cannot be affirmed by the court, after the treaty between the German Empire and the United States of America referring to the mutual industrial protection, dated Feb. 23, 1909, has taken effect on Aug. 1, 1909, and after the patents have been assigned to the present owner.

"The treaty which has been passed by the Federal Council and by the Diet, and which has been published in the *Imperial Gazette*, is legally binding. It is provided in article 1, sentence 1, that those laws of the one country of the high contracting parties which decree the withdrawal or other restrictions in case of non-working of a patent or other protective rights, should apply to patents belong to citizens of the other high contracting party, only as far as patents belonging to citizens of this country are affected by their own laws. The effect of this provision for the patents which are here under discussion becomes evident from the circumstance that according to § 11 of the German patent law the owner of the patent is obliged to work his invention to an adequate extent, if he does not want to lose his patent rights, but that, on the other hand, there is no working clause in the United States. As appears also from the memorandum with which the Imperial Chancellor submitted the treaty to the Diet, this treaty sets forth that American citizens are freed from the provisions of the German law regarding the working clause. As it is there further stated, 'they are hereby placed on an equality with the German citizens who obtain patent protection in the United States without any obligation to work their patents,' therefore the treaty provides that the working clause which is decreed by the German law does not apply to patents belonging to a citizen of the United States. Because in his home country the working clause does not exist, he is not subjected to the German provisions for a working clause on account of his nationality.

"The question whether the provisions of this treaty should affect this suit which has been started before the ratification of the treaty will be discussed later on. If we at present disregard this point, it would follow that for the reason that the patents in question now belong to the American firm, the motion for the withdrawal of the patents would not succeed. The complainant opposes this and asks for a different interpretation of the treaty from two points of view.

"First, the complainant contends that the treaty only speaks of patents which have been 'granted' to citizens of the other country. This means to the applicant for the patent or to that person to whom the patent has originally been granted, but it seems to exclude later assignees, such as the present complainant. This point is not well taken. It is contradicted by the English text, which is also published in the *Imperial Gazette*. This reads: 'Patents enjoyed by the citizens of the other contracting party.' That means patents the ownership of which is enjoyed by the citizens who possess them. * * *

"The complainant has further argued that in the treaty the expression 'citizens' has been used and that citizenship is a quality which is inherent only to physical individuals and since the present defendant is a corporation doing business in the State of Ohio under the American law, it cannot be a 'citizen.' But this argument must also necessarily fail. The point whether the terms 'citizens,' 'subjects,' used in treaties apply to physical persons only has been already adjudicated and denied by the Imperial Court. * * * It must be decided that any corporation or company doing business within the territory of any two countries and incorporated under the laws of the country must be considered 'citizens' within the meaning of the treaty.

"A third restriction of the treaty, which, however, has not been argued by the complainant, has been mentioned in a publication of the *Zeitschrift für Industrierecht* (1909, page 229). It is there contended that an American is only freed from the working of his German patent if he owns at the same time a corresponding home patent. The testimony in this suit differs as to the question whether such is the case with the patents in litigation, so that further testimony would have to be taken if this point were argued. But the point is not well taken. * * *

"Finally, the question must be considered whether the provisions of the treaty should apply in a suit which has been brought before the treaty took effect, and in which suit the courts of the first instance has already rendered an unfavorable decision. In this respect it must at once be denied that new laws enacted in favor of the citizens of the United States are only effective for patents which are applied for or granted after the treaty took effect. Neither the context nor the purpose of the treaty allows such a restricting interpretation. Evidently the purpose of the treaty was to make the freedom from the working clause unrestricted, at least from now on, and to enforce this freedom also in case of older protective rights. It is only debatable how the treaty affects patents which have existed beyond the limit defined in § 11 of the patent law and which, within these limits, have not been worked, so that according to the present law a motion for their withdrawal would be allowed, as has been the case with the patents here in suit.

"In answering this question, the general principle must be considered that new laws must not affect well-acquired private rights. This would apply, if complainant after the lapse of three years, during which the patent had not been worked, would have acquired a private right regarding the withdrawal of the patent. But this is not so. A private right for the withdrawal of a patent does not exist. The withdrawal possesses the nature of a popular complaint which the individual only exercises for the benefit of the general and public interest. A patent 'may' in case of certain legal presumptions be withdrawn, but it must not be done necessarily. The court possessing jurisdiction may not insist on withdrawing, if considering the individual case it concludes that the public interest of the home country is not injured or if it decides that the owner of the patent alleges good reasons for non-working. A decision lies within the free judicial discretion and it must be rendered strictly in accordance with the facts prevailing at the time of rendering the decision. Especially for the higher instances of the court, the circumstances prevailing at the time when the decision is rendered, are binding.

"The question what is to be considered as public interest might be differently answered. It may be held of primary importance that the patent law should serve the national industry and that it must help to increase domestic welfare by increased opportunity for labor; or it may be held that the free international exchange of all technical progress and the new products which are obtained in this way may be the first consideration. In this regard the courts must judge in accordance with the existing laws. The patent law in § 11 sanctions the protection of the national industry. Nobody must be permitted to utilize a patent to the exclusion of others. The inventor must within a certain time work the invention at home. The Paris International Convention for the Protection of Industrial Property and the Brussels additional treaty confirmed this principle. The treaty with the United States, however, supports the second point of view. By granting to the American industry which by their domestic laws is freed from the working clause certain privileges in Germany which the national industry does not enjoy and permitting the American to utilize his German patents to the exclusion of the German industry, the treaty brings out other and new viewpoints for the definition of the public interest in relation to the industry of this country. After the treaty came ino force and during its duration, the German courts cannot enforce any more upon a citizen of the United States the protection of the German national industry as a public interest.

"For these reasons the motion to withdraw the two patents in suit must be denied."

This decision of the German Reichsgericht is a model of impartiality. The definition of the scope of "public interest" in the concluding part of the decision shows a spirit of enlightened self-interest and broad-mindedness which is admirable.

Notes on Chemistry and Metallurgy in Great Britain.

(*From our Special Correspondent.*)

The Institute of Metals.

The annual general meeting was held on Jan. 18 and 19, at the Institution of Mechanical Engineers, when the new president, Sir Gerard Muntz, delivered his inaugural address, in the course of which he said that the absence of a general knowledge of the non-ferrous metals was simply appalling; and a large field existed for young metallurgists in such metals. There was probably a larger extent of investigation before the Institute than was open to any kindred association.

In opening the discussion on the paper on "The Properties and Constitution of the Copper-Arsenic Alloys," by Mr J. D. Bengough and Mr. B. P. Hill (abstracted on page 95 of our February issue), Professor Huntington said he had a notion he had done some work on the subject some years ago. The first point attracting his attention was the statement that arsenic gave copper good working properties at high temperatures. His experience was that arsenic had a very bad effect on copper at high temperatures.

Mr. W. Rosenhain, with reference to the formation of cracks, said Professor Huntington's theory appealed to him rather than the authors' deduction. Raising the specified limit of arsenic could not be recommended without further research; and it would be essential to learn something more about fatigue and resistance to shock.

Mr. Rhead said that two or three years ago he was working on alloys of electro-copper containing from 0.5 to 1.5 per cent of arsenic. With 0.5 per cent distinct separation of arsenic compounds, on the borders of the crystals, took place. A similar segregation occurred when antimony was used. On two occasions he had had evidence of serious pollution of arsenic alloys by gases containing chlorine. At their first Birmingham meeting he had pointed out the serious effects produced by heating copper for long periods in reducing gases. The production of oxide accounted for the cracked appearances.

Mr. Bengough replied, as to the reaction between the oxide

and reducing gases, that there was a critical temperature at which the reaction took place. It did not occur below 600° C. He proposed to make his general reply in a written communication.

Mr. Ernest A. Smith's paper on **"The Assay of Industrial Gold Alloys"** followed. It was abstracted on page 92 of our last issue.

Professor Gowland attached a high value to this paper by reason of its containing much original matter hitherto unpublished. Mr. Smith was especially fitted for writing such a paper. The old Japanese 9-carat alloy contained gold and silver only, and the silver could be removed from the surface by pickling. The Japanese alloy now contains silver and copper. There was difficulty in taking a fair sample from a gold-silver-copper alloy; in fact, a dip sample was necessary. When silver only was alloyed with the gold the old touchstone test gave reliable indications for the assay.

Professor Turner concurred as to the great value of the paper, but regretted that any reference to cadmium in the alloys had been omitted.

The next paper read was by Dr. R. Seligman and Mr. F. J. Willott on **"The Analysis of Aluminium and its Alloys,"** in which the authors strongly urged the desirability of standardization of methods, and fully described the procedure they had found consistently reliable. (An abstract of the paper appeared on page 94 of our last issue.) The discussion was adjourned.

On the second day, Jan. 19, the first paper taken was on **"A Contribution to the Study of Phosphor Bronze,"** by Mr. O. F. Hudson and Mr. E. F. Law, who said that the object of their research was to investigate the relation between the constitution of the phosphor-bronze alloys and their mechanical properties. (See the abstract on page 95 of our last issue.)

Mr. A. Philip fully recognized the great value of the work carried out by the authors, but thought that the evidence they had produced of the existence of a triple alloy of copper, phosphorus, and tin, was not quite conclusive.

Dr. Rosenhain was also of opinion that the evidence adduced by the authors did not adequately support their conclusions. When the additional facts embodied in the full details of their work were placed before the Institute the authors would have made a most valuable contribution to the subject which would be of permanent interest.

Mr. Spence Thomas regarded the matter from the point of view of a user of phosphor-bronze. In his experience the life of bearings in hot mills, under exactly similar conditions, varied from 1 month to 15 months. He urged co-operation between producers and users in the hope of securing greater durability.

Professor Turner remarked that the differences in mechanical properties between the various grades of phosphor-bronze were as great as those between cast iron, wrought iron and steel. Whatever opinions might be entertained as to the existence of the triple alloy described in the paper, the fact remained that there was some constituent of phosphor bronze which disappeared and reappeared under varying conditions of heat treatment, and this fact was most important in connection with the subject.

Mr. Law replied that the matters referred to by Mr. Spence Thomas might be explained by the low melting point of the triple eutectic, viz., 620° C., the consequence being that bearings in which it occurred were not able to withstand the high temperature encountered in hot mills.

Mr. Spence Thomas said that when such bearings were used in cold mills their life also varied, and had been found both longer and shorter than in hot mills.

Mr. Gaywood attributed the trouble with the bearings to an excess of lead in the phosphor-bronze.

Mr. Law had arranged to illustrate the paper by lantern projections from colored photomicrographs on Lumière plates; but, unfortunately, the impracticability of securing sufficient exclusion of daylight, or a suitable disposition of the apparatus, did not permit of the desired degree of distinctness being attained.

Mr. T. Vaughan Hughes read a paper on **"Failure in Practice of Non-ferrous Metals and Alloys,** with Particular Reference to Brass Loco-Tubes." (Abstracted on page 94 of our last issue.)

Dr. Rosenhain, in the discussion, said he found difficulty in acquiescing in the view that a film of non-conducting substance one-tenth of a millimeter in thickness could so prevent the transmission of heat as to raise the temperature to 800° C. Had the author further examined the tubes from which the deposit had been scraped? If the theory were true, the effect of over-heating would be found at the firebox ends of nearly all the tubes, and such evidence, if forthcoming, would be almost conclusive. He did not concede that there was justification for assuming that overheating occurred only after the tubes came from the makers.

Professor Turner said the thinning of the tubes might be attributed to various causes. Abrasion by particles of clinker was one; and for another there might be chlorine in the gases. Examination of the films on the surfaces of tubes undergoing corrosion would show the presence of a considerable quantity of chlorine. If lead were present it certainly caused failure; but corrosion was known to occur in tubes which were commercially pure. Tubes were cold-drawn, the ends were expanded afterwards, and the damage was often caused by the final annealing.

Professor Huntington agreed that the annealing was probably responsible for troubles. The usual difficulty was in getting a continuous history of the material from its manufacture to its use.

Mr. Heathcote remarked that, from a paper recently read before the Society of Chemical Industry, it appeared that oil in the feed-water had caused the tubes to attain a temperature so high that globules of molten copper were found on their fire-sides. This represented 1000° C.

Mr. Bengough said that overheating, doubtless, occurred during the expanding. The strength of the material was not very much affected at ordinary temperatures; but under stress at high temperatures it was seriously deteriorated. He considered that prolonged heating at a moderate temperature would produce the same structure as heating for a short period at 800° C.

The president observed that he had experienced corrosion from the outside of the tubes; he had known tubes to have been corroded quite through where alkaline water was used. The Midland Railway had had much difficulty in that respect. As to the effect of chlorine, the old tubes returned had chlorides as well as sulphides in the tube.

Mr. Vaughan Hughes explained that the fuel used was coke. No chlorides of copper or zinc were found, but sulphate and chloride of calcium.

Institution of Mechanical Engineers.

Alloys Research Committee.

Mr. E. B. Ellington presided at a meeting of this Institution on Jan. 21, when the ninth report of the Alloys Research Committee was presented. It deals with the properties of various alloys of copper, aluminium and manganese, and corrosion of copper-aluminium alloys by sea-water.

Sir W. H. White, chairman of the committee, said the work of that committee was to so conduct the research that the chief results would be of practical utility to mechanical engineers. Some of the earlier reports had, he thought, been unjustly criticized in that respect. The results of the work done were so stated as to be capable of being applied commercially in the manufacture of non-ferrous metals.

Sir Gerard Muntz stated that his practical experience enabled him to confirm some of the results arrived at in the report. His firm was making aluminium castings for motor cars, and

great difficulty had been encountered in the heat treatment. The cutting qualities of some of the alloys had considerable interest for him; and that was a point which, among others, he should try to follow up as a manufacturer.

Prof. J. O. Arnold remarked that the work which had been done was in a direction new to engineers.

It appeared that from a mechanical point of view they must not expect too much from micrographical analysis. Molecular changes, both in ferrous and non-ferrous metals, demanded investigation in the interests of public safety. Such changes were often not detected by the usual tests, but had a serious effect on the properties of metals.

The discussion stands adjourned to Feb. 18.

Market Prices.

Copper.—Starting at close on £62 per ton, the price of standard fluctuated frequently, though not to great extent, up to Jan. 12, it then commenced to fall, but picking up slightly on the 18th, recovered ultimately to £60 17 6.

Tin.—Dropped smartly from the opening price of £153 to £148 10 0 during the first three days; then recovered to £151 7 0, after which fairly steady fall to £147 10 0.

English Lead.—Flat throughout the month at £14 to £14 3 0, closing £13 7 6.

Iron, Cleveland Warrants.—Rose fairly steadily from 51/- to 52/3.

Scotch Pig.—Followed much the same course as Cleveland, i.e., during the month (6th to 13th and 20th) carrying the price from 62 6 to 65/-; closing at 67/-, but little doing at this price.

Aluminium ingots, in ton weights	£75	0	0
Antimony, black sulphide powder, per ton	20	0	0
Sulphate of ammonia, f.o.b. Liverpool, per ton	11	12	6
Copper sulphate, per ton	19	15	0
Caustic soda, white, 77 per cent, per ton	11	2	6
Bleaching powder, 35 per cent, per ton	4	5	0
Shellac, standard T. N., orange spots, cwt	3	8	6
Carbolic acid, liquid, 97 to 99 per cent, gal			11
Creosote, ordinary good liquid, gal			2⅜
Naphtha solvent 90 per cent at 160° C., gal		1	3½
Rubber, Para, fine, lb		7	8
Platinum, per ounce troy, nominal	5	15	0
Zinc (Vicille Montague bud), per ton	27	10	0

London, England.

Recent Metallurgical and Chemical Patents

Iron and Steel.

Basic Open-Hearth Process.—Mr. Louis M. Atha, of Newark, N. J., remarks that the ordinary basic open-hearth process of making steel from a mixture of pig iron, steel and iron scrap is expensive in certain localities owing to the high price of pig and the large amount of limestone necessary to add to the charge. The pig iron or cast iron are employed for the carbon they contain, to melt down the steel. Steel scrap could be substituted if carbon was introduced in some form separately. Mr. Atha states that while pulverized coke, charcoal or anthracite will not do, because they are carried off with the vapors, oil retort carbon may be advantageously employed. He uses 75 per cent steel scrap, 18 cast iron or pig iron, 2 carbon, and 5 limestone. "These proportions give a saving of about 30 per cent of pig and 50 per cent of limestone. By using steel scrap, which is low in phosphorus, it is not necessary to use as large a quantity of limestone as usual." (943,171, Dec. 14, 1909.)

Tool Steel.—Mr. John W. Evans, of Belleville, Ont., Can., patents, a tool steel, containing 92 per cent or more of iron, 3 per cent or less of titanium, 2 per cent or less of vanadium, 3 per cent of nickel or less, and 1 per cent or less of carbon. It has "the characteristics of being tough and hard to the end that it will retain an edge when working hard metals at high speed." (948,166, Feb. 1, 1910.)

Furnace Construction.—Details of construction of a special furnace for the heating of the ends of blanks from which bolts are to be manufactured are described in a patent of Mr. Edwin Alls. (943,763, Dec. 21, 1909; assigned to Monarch Engineering and Mfg. Co.)

Electric Steel.—Three new modifications of induction furnace designs for steel refining, due to Gronwall, Lindblad, and Stalhane, Unger and Grunwald are described below under Electrical Furnaces. At the same place will be found a method of Queneau of using the pinch effect for mixing an electric furnace charge, and a method of Strauss of making calcium silicide for steel refining.

Copper.

Fractional Roasting and Flotation Process.—To separate the valuable metals from such ores as chalcopyrite $CuFeS_2$, bornite or cruhesite Cu_3FeS_3 and mixtures of the same with pyrites in which copper is chemically combined with iron, Mr. A. S. Ramage, of Newark, N. J., subjects them to a fractional roasting process, with subsequent separation by flotation. For instance, in case of an ore containing iron pyrites and chalcopyrite, with a content of about 5 per cent of copper and from 30 to 40 per cent of sulphur, the ore is roasted at about a red heat in a furnace capable of utilizing the exothermic reaction between the ingredients of the ore. The roasting is conducted at a temperature sufficient to decompose only the iron sulphide, and leave the copper in the form of copper sulphide. The roasted ore is then crushed to at least fifteen mesh and passed through a solution of acid sulphate of soda and free nitric acid (the solution being formed by adding nitric acid to sulphate of soda), which solution is kept near the boiling point. The copper sulphide immediately rises to the top of the bath and can be floated off. Any copper oxide which may have been formed in the roasting operation, at once dissolves and can be recovered from the solution by any of the well-known ways.

Where the ore contains besides iron pyrites and chalcopyrite, zinc blende in quantity, the ore is roasted at a temperature of not over 600° C, with the result that the iron sulphide only is decomposed. The roasted ore is then subjected to the acid sulphate of soda solution in the manner before described, the solution being kept at nearly the boiling point, when the unchanged sulphide of zinc and copper rise to the top of the bath and are floated off, thus being together separated from the iron oxide. The copper and zinc sulphides are then roasted at about 700° C, until all the zinc sulphide is decomposed, the burnt mixture, after cooling, being treated with dilute sulphuric acid, whereby the zinc is completely dissolved, and the copper sulphide is left unchanged. The zinc may then be recovered from the solution by electrolysis in any preferred manner.

In the case of the ores of the Cobalt district, Dominion of Canada, which are principally cobalite, nicolite, and mixtures of chalcopyrite and pyrites containing silver in the metallic state and also as sulphide and sulpharsenide, these minerals are "floated" from the gangue, the metallic silver remaining with it, and from which it is mechanically separated. The sulphides and sulpharsenides of cobalt, nickel, iron and copper are then roasted at about 800° C, the roasted ore is "floated" as above and the copper sulphide floated off. The iron, nickel, and cobalt are reduced together as an alloy, electrolytically. (949,002, Feb. 15, 1910, assigned to Chemical Development Co.)

Reduction of Copper Sulphide.—The nearly pure sulphide

of copper obtained in the above process is placed by Mr. Ramage in a cylindrical furnace, where it is reduced to the metallic state by mixture of hydrogen and water gas at a low red heat. At this temperature the hydrogen reduces the copper sulphide to metallic copper in the form of a powder, while the carbonic oxide and sulphuretted hydrogen gases pass off and are led under and around the cylindrical furnace and burned to supply the heat necessary for reduction. The reduced metallic powder is delivered from the furnace into air-tight chambers and allowed to cool. The ore could be reduced by means of carbon, but some undecomposed carbon always remains, and in the subsequent electrolysis contaminates the electrolytic copper, so that the gaseous reduction is preferable. The cold metallic powder is preferably then used as an anode in a suitable electrolyte. The cathodes can be sheets of copper or revolving tubes. (949,003, Feb. 15, 1910.)

Wet Process for Pyritic Copper and Copper-Nickel Ores.—Mr. J. T. Carrick, of Johannesburgh, Transvaal, digests the hot desulphurized crushed ore with an excess of hydrochloric acid, in the absence of oxygen, until the iron and also the nickel are dissolved out. (By treatment of the digester liquor with ammonia and aspiration of air through it, the iron is precipitated, and after removal of the iron precipitate, the nickel is thrown down by the addition of lime.) The ore remaining in the treatment vat after removal of the iron and nickel consists of a porous mass of gangue and sulphuric acid and by injection of compressed air the sulphide is oxidized to cupric sulphate, which is leached out in about 24 hours. (947,789, Feb. 1.)

Separation of Copper from Other Metals.—See below the abstracts of the patent of Chalas on copper-nickel separation under Electrolytic Processes and of the patent of Chance on treating copper-iron matte under Electrolytic Furnaces.

Gold and Silver.

Agitating Ore Pulp.—In the treatment of ore pulp there is a difficulty in preventing the heavier particles from settling to the bottom of the tank, where they would be practically withdrawn from the effective action of the cyanide solution. To prevent this, Mr. Bernard MacDonald, of Guanajuato, Mexico, employs the apparatus shown in Fig. 1, which makes use principally of compressed air, without the employment of mechanical stirrers and agitators. The charge is supplied to the tank from the overhead main *a* into the funnel *d* at the top of the charge pipe *e'*. The funnel *d* has a perforated bottom *d'* upon which rests the cyanide to be brought into solution. At its lower end the tank is subdivided into several compartments by transverse partitioning with inclined sides. In the illustration (in which the lower diagram is a horizontal section along the line 2-2 of the upper diagram), this partitioning is in form of a single ridge-like gable *i*, so that in this case there are only two compartments. In this case the charge pipe *e'* rests upon the summit of the gable *i*, and at one end thereof, and from this point it extends downwards and forwards, terminating in branch jets *h* directed toward the valve casings *j*. The charge pipe *e'* has a height of 40 to 47 feet, so that the hydrostatic pressure is strong enough to insure a vigorous exit of the charge from the jet openings. After the charge of ore pulp or slime has been supplied to the tank, or at intervals during such supply, the valve *f* at the top of the charge pipe is closed so as to cut off further supply of pulp, and through the pipe *g* (right below this valve) compressed air is admitted into the charge pipe *e'* to agitate the charge and prevent settling. Air-injector jets extend upwardly through the bottom of the tank into the valve casings *j*; they are connected with the supply of compressed air through the piping *p*. Right above the valve casings *j* are provided vertical transfer pipes *k*. By the action of the compressed air the heavy particles of pulp which tend to settle down in the pockets at the bottom, are driven up into the transfer pipes *k*, and are discharged again into the tank at the top through the T-connections *o*. In this way the whole pulp is continually stirred up and is being kept in a practically homogeneous consistency. After the treatment is completed, the pulp is discharged through the down-takes *g'* and pipe *g* at the bottom to the filter press. (948,766, Feb. 8, 1910.)

Electrolytic Amalgamation.—See the abstract of the patent of Alling below, under Electrolytic Processes.

Zinc.

Reduction by Carbon Monoxide.—A process for reducing zinc oxide to metallic zinc by carbon monoxide, practically pure and free from nitrogen, is patented by Mr. Oscar Loiseau, of Selaigneaux, Belgium. The carbon monoxide gas is produced at a temperature somewhat above the reduction

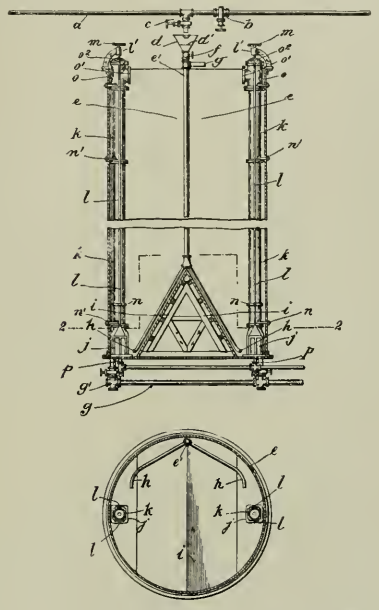


FIG. 1.—AGITATION OF PULP IN CYANIDE TANK.

temperature of zinc oxide (1400° C). The simplest way to produce the carbon monoxide gas is to have an incandescent mixture of coke and zinc oxide in a large closed reduction chamber and to blow pure oxygen, mixed with pure carbonic acid (free from nitrogen), over this mixture. The less the proportion of carbonic oxide compared with the oxygen, the higher is the temperature of the carbon monoxide gas produced. The reduction of the ore occurs according to the usual reactions; the role of the reducing carbon in the mixture is to transform immediately into carbon monoxide the carbonic acid derived from the reduction of the ore. The metallic zinc is obtained in separate condensers. The chief advantage of such a process would be the employment of a large-unit reduction chamber instead of the many retorts in present zinc smelting practice. The method is especially interesting as another indication of the great interest now being taken by various metallurgists in Belgium in the use of liquid air for the separation of oxygen and nitrogen, and the uses of pure oxygen for various metallurgical purposes. (947,831, Feb. 1, 1910.)

Electric Zinc Smelting.—A note in the abstract of a patent of Queneau on using the pinch effect for agitating an electric furnace charge, found below under Electric Furnace, refers to electric zinc smelting.

Various Metallurgical Patents.

Combined Roasting and Smelting Furnace.—Much has been heard last year of the Fink furnace, on account of the trial which it has undergone at Garfield, Utah, as described in

our Vol. 7, page 287. A patent of the inventor, Dr. Edward Fink, of Milwaukee, Wis., describes the furnace as follows: It is a combination of a heating or roasting barrel *a* and a smelting barrel *b*, which are arranged one above the other. Both barrels are charged with pulverized ore and fluxes and are rotated, the openings *i* of the barrel *a* and the tap holes 3 and 5 of the barrel *b* being closed. Fuel and air are supplied to the burner nozzles *q*, while air is blown through the pipe *s* so as to direct the hot blast over the ore in the smelting barrel *b*. By the rotation of the two barrels, the ore and the fluxes are thoroughly mixed and under the action of the hot blast in the lower barrel *b* the ore in it gradually begins to melt; the gaseous products of combustion escape through the flue *f* into the barrel *a*, where they heat the ore contained in it and finally escape through the flue *g*. The ore in the barrel *a* is thus gradually heated, and if it contains considerable sulphur this is ignited and the ore begins to roast; air being supplied thereto if necessary through the connection 6. The air blown through the tube *s* is heated so that a true hot blast is issued from nozzle *q*. When the smelting operation in barrel *b* is concluded the slag and the molten metal are withdrawn and by means of the connection *o* and the screw conveyor *n* the roasted ore from the barrel *a* is now brought into the melting barrel *b*. Then the conveyor is again removed from the hole *m* and the latter is closed. Fresh ore is filled into *a* and the operation starts again. For converting or refining operations, as, for example, for converting matte produced from copper ores containing considerable sulphur into an enriched matte or blister copper, one or more twyers *w* are introduced into the barrel *b* and are connected with the blast pipe *z* of the blower *z*. Each twyer is adjusted so that its inner end will project slightly below the surface of the molten

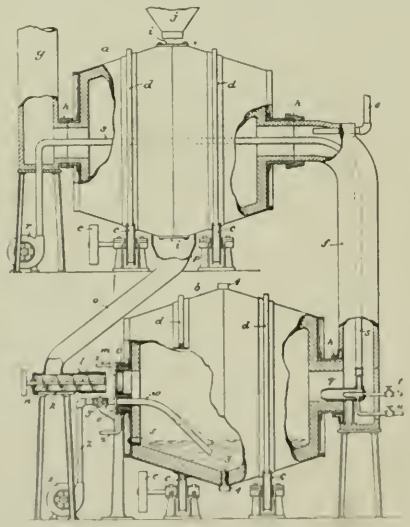


FIG. 2.—COMBINED ROASTING AND SMELTING FURNACE.

matte. Air is then blown and the barrel is rotated and a strong oxidizing flame is projected from the nozzle *q* over the matte. (948,468, Feb. 8, 1910.)

Nickel-Cobalt-Silver-Arsenic Ores.—For the smelting of ores which contain, besides silver, considerable quantities of nickel, cobalt, and especially arsenic (for instance, 300 to 10,000 ounces of silver per ton, to to 30 per cent nickel and cobalt, 5 to 40 per cent arsenic, no lead). Mr. C. C. Cito, of Irvington, N. J., proposes to add to the ore a percentage of copper, preferably equal to two-thirds of that of nickel and cobalt, and at least the same as that of arsenic, and not less in amount than the silver. The smelting operation, yields, firstly, an alloy containing besides copper, all the silver, nickel, and cobalt, and nearly all the arsenic in the ore treated, and secondly, a slag containing a very low percentage of silver, nickel, and cobalt. The alloy is subsequently separated by electrolysis, while the slag is used as flux in the blast furnace to recover the balance of the metals. (949,059 and 949,261, Feb. 15, 1910.)

Gas-Fired Vertical Smelting Furnace.—Mr. John S. Loder, of Reno, Nevada, builds the upper section of his smelting furnace in the usual way with vertical refractory walls. Below the straight vertical upper section is placed a funnel-shaped lower section made of metal and water-cooled, and this section is provided with a projecting shoulder or bulge. The ore charged into the upper section strikes the inclined lower section and the projecting shoulder, and the intention is to support it in this way in form of an arch by the high pressure produced below the shoulder by the gaseous fuel and air charged into the furnace through burners. This is intended to provide a free combustion space in the bottom of the furnace immediately beneath the ore, so that the flame plays on the bottom of the ore, gradually melting the same, which falls to the floor. (948,098, Feb. 1, 1910.)

Utilization of Flue Dust from Smelters.—Mr. N. H. Livington, of Globe, Arizona, proposes to utilize the collected flue dust from smelters directly in a converter without first briquetting it. The dust delivered from a hopper is blown into a pipe connected to a high-pressure blower and is then blown into a large pipe connected to a low-pressure blower and discharging into the converter, where the dust mixes with the molten metal. (943,752, Dec. 21, 1909.)

Electric Furnaces.

Induction Furnace.—To reduce or eliminate the primary stray flux (which has such a detrimental effect on the power-factor of an induction furnace), Messrs. E. A. Grönwall, A. R. Lindblad and O. Stalhane, of Ludvika, Sweden, place auxiliary coils in the path of the leakage flux in such a way that the magnetic flux produced by these coils counteracts the leakage flux. Fig. 3 shows one possible method. The coil 5 with the terminals 6 and 7 is the primary. The channel 4 containing the fused bath is the secondary. The coils 8 and 9 are the auxiliary coils; in the upper diagram one terminal 10 of one coil is connected to one terminal 11 of the other; the two other terminals 12 and 13 are connected to an outside source of current. If there exists primary leakage a greater number of lines of magnetic flux pass through leg 1 than through leg 2; the e.m.f. is induced in coil 8 is greater than the e.m.f. induced in coil 9. The difference of these two e.m.fs. is proportioned to the primary leakage flux. If a suitable current is passed through the coils 8 and 9 so that the m.m.f. of the coils will equal the m.m.f. with which the primary leakage flux tends to issue from core, the primary leakage is thereby overcome. Only a small voltage is then required at the terminals 12 and 13 of the coils 8 and 9, sufficient to overcome their ohmic resistance. The current for the auxiliary coils may be taken from the primary mains, and for this purpose the "Hick transformer coupling" may be employed; this is illustrated in the lower diagram. (942,961, Dec. 14, 1909.)

Silicon Carbide, Artificial Gems.—The theoretical reaction on which the manufacture of carborundum is usually based is $SiO_2 + 3C = SiC + 2CO$, and the following proportions by weight (affording a slight excess of carbon over that theoretically required) are used in practice: 35.1 per cent ground coke, 54.4 sand, 7 sawdust, 3.5 common salt. The silicon carbide thus produced occurs in the form of opaque crystals of various colors, the black predominating. Mr. Frank J. Tone, superintendent of the Carborundum Company, has discovered another reaction $3Si + 2CO = 2SiC + SiO_2$. This

is a gaseous reaction, gaseous silicon combining with carbon monoxide at a temperature below the temperature of dissociation of silicon carbide. The silicon carbide thus produced builds up in large crystals, which start upon crystals, initially

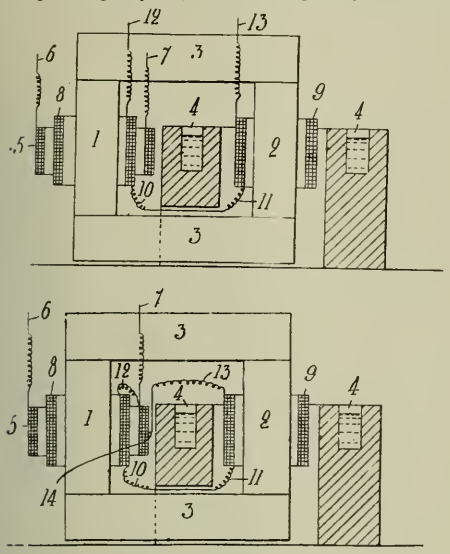


FIG. 3.—INDUCTION FURNACE.

formed by the direct union of silica and carbon of the charge, and by prolonging the operation of the furnace these crystals can be made to attain a large size. For this purpose the charge mixture should be constituted so as to afford spaces for the mingling and combining of the gases. This space is afforded by using sawdust in the charge mixture, preferably in a larger percentage than heretofore employed. Mr. Tone uses a carborundum furnace with a core of carbon, preferably in the form of carbon granules, free from contamination of metallic oxides (for instance, a carbon core taken from previous runs of a furnace). This he surrounds with a mixture of pure silica, petroleum coke (or purified ground coke), and sawdust. These are mixed in a proportion so that the silica will be somewhat in excess of the amount theoretically required by the formula

$$SiO_2 + 3\,C = SiC + 2\,CO.$$

These proportions by weight are attained by the following mixture: ground carbon, 30 per cent; silica, 57 per cent; sawdust, 9 per cent; common salt, 4 per cent. The furnace is then started and the operation is carried on for such a length of time that the zone of carborundum crystals adjacent to the core is decomposed to a considerable thickness, preferably to an extent of 2 in. or 3 in. The decomposition of these crystals results in the liberation of silicon vapor, which combines with the carbon monoxide gases in the furnace and builds up large crystals of carborundum just outside the zone of graphite which is formed by the decomposition. The crystals thus produced are clear and transparent, being free from coloration, and can be made of a large size. It is important that the carbon of the charge should not be in excess, since otherwise the crystals produced are apt to be black. These crystals have a very high index of refraction (exceeding that of crown glass and the diamond) and possess double refraction and are stated to be of great value as gems, since they can be cut with regular facets and polished. By adding to the furnace charge a small amount (not more than 0.5 per cent) of a coloring material, like oxide of chromium or oxide of manganese, the crystals can be tinted without destroying their transparency. (949,386, Feb. 15, 1910; assigned to Carborundum Company.)

Induction Furnace.—A modified form of induction furnace has been devised by Mr. MAGNUS UNGER, of Pittsfield, Mass. Fig. 4 shows it in perspective, with a plan view with connections to a polyphase system. It is a tilting furnace with two sets of primary windings 3 and 4 wound upon cores 5 and 6. Surrounding the primary windings is the furnace chamber 7. The annular furnace chambers interlink to form a central space which is used as a convenient treating chamber. As the annular channels surrounding the primary cores may be made of comparatively small cross-section without sacrificing the capacity of the furnace, the resistance at these regions will be correspondingly high, thus keeping a high power-factor and at the same time making a furnace of larger capacity than it is otherwise possible to operate with commercial frequencies. As there will be practically no current in the center of the furnace, the furnace chamber is made shallower at this point by the projection 9. However, the central chamber holds a considerable amount of the charge, which is protected in this way from excessive radiation, and the furnace has, therefore, a high thermal efficiency. The central space may be used for the introduction of comparatively large blocks of raw material, either ore or flux. The furnace may obviously be operated also by single-phase current. The design reminds one somewhat of the general arrangement of the Roechling-Rodenhauser furnace, although the pole plates characteristic for the latter are absent. (941,435, Nov. 30, 1909; assigned to General Electric Company.)

Induction Furnace.—To prevent the radiation of heat from the open top of an induction furnace and to permit the charging of the furnace when closed while facilitating full inspection of its interior, Mr. CARL GRUNWALD, of Essen, Germany, builds the secondary melting channel of oval or rhombic rather than circular form and provides it with lateral charging doors so that it is possible to inspect the entire surface of the

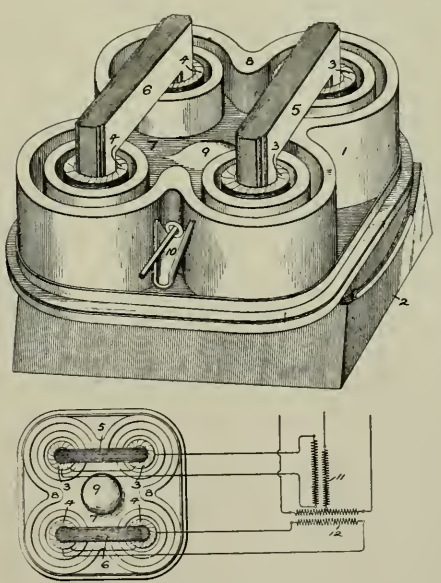


FIG. 4.—INDUCTION FURNACE.

molten charge from these two ends. (943,403, Dec. 14, 1909.)

Mixing of an Electric Furnace Charge by Pinch Effect.—Mr. A. L. QUENEAU, of Philadelphia, Pa., makes use of the "pinch effect" to produce regular pulsations in an electric furnace charge in order to thoroughly mix it. His arrangement, as applied to a three-phase furnace, is illustrated in Fig. 5, only two of the electrodes *i* being shown. The description relates

to steel refining. In this case the electrodes *i* are made of steel and are inserted in the lower ends of three narrow channels which converge upward and connect with the molten charge in the hearth *c*. With a sufficient current the pinch effect manifests itself in the channels *g*, near the upper end, a short distance below the entrance to the hearth, and will thereby cause an interruption of the molten-metal column within the channel. But the hydrostatic pressure of the heavy metal bath in the hearth will immediately close again this disruption, then it is broken again, and so on. It is emphasized that the interruptions occur at places where reducing conditions prevail so that there can be no oxidation of the disrupted surfaces. These pulsations are transferred to the molten metal in the hearth which is thereby held in constant agitation and thoroughly mixed. While the description in the patent specification relates primarily to steel refining, Mr. Queneau, who is a well-known authority on zinc metallurgy, thinks that the furnace would also be suitable with a molten metal as resistor, which is not to be refined itself, but is used as a source of heat to transmit the

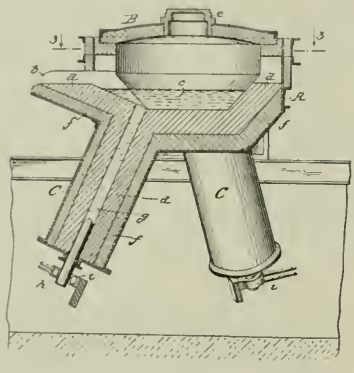


FIG. 5.—MIXING CHARGE BY PINCH EFFECT.

heating effect "to a charge of some other material superposed upon it, as, for instance, a zinc ore charge made up of the usual mixture of zinc ore and coke for the production of spelter." (948,343, Feb. 8, 1910.)

Electric Furnace for Reduction of Scruff and Dross.—Mr. Raymond S. Wile, of Pittsburgh, Pa., uses the furnace shown in Fig. 6 for the reduction of scruff and dross from tinning, galvanizing and similar operations. The metal shell 2 is lined with the refractory 3. Of the two carbon electrodes 6 and 7 the upper is movable. The dross or scruff, mixed with coke dust as reducing agent, is fed into the upper part of the furnace and rests on a bath of neutral melted material, such as glass. The upper electrode is so suspended that it just dips into the surface of the melted glass bath. The object of the melted glass bath is to collect in it the reduced metal, which is thereby protected against oxidation. The metal collects around the lower electrode and is tapped out from time to time. One chief advantage of this furnace is that the loss of metal which occurs in present methods is practically eliminated. Any metal or metallic oxides which may be volatilized in the lower portions of the charge is condensed again in the upper portions. The molten bath of glass must, of course, be of lower specific gravity than the reduced metal. (947,723, Jan. 25, 1910, assigned to American Sheet & Tin Plate Company.)

Calcium Silicide for Steel Refining.—The production of calcium silicide, which is now finding more and more extended use in steel works, is the subject of a patent of Mr Georges Strauss (assigned to Societe Anonyme La Compagnie Generale d'Electro-Chimie de Bozel, of Paris, France). By heating a mixture, in definite proportions, of lime, silica and carbon in an electric furnace, calcium silicide is obtained according to the equation

$$CaO + 2\,SiO_2 + 5\,C = CaSi_2 + 5\,CO.$$

But the direct formation of silicates is a troublesome element in this process. To avoid this, Strauss uses calcium carbide (instead of lime) and metallic silicon (instead of silica) as

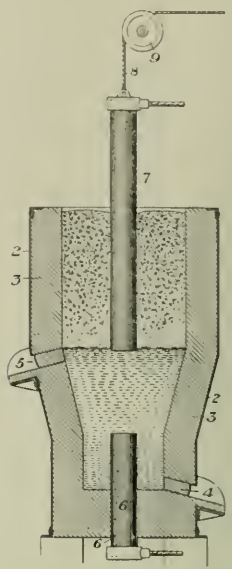


FIG. 6.—FURNACE FOR REDUCTION OF DROSS.

starting materials. He gives the following equations for the reactions:

$$CaC_2 + 2\,SiO_2 + 2\,C = CaSi_2 + 4\,CO.$$
$$2\,Si + CaO + C = CaSi_2 + CO.$$

"Experiments made with calcium silicide have shown that this metallic silicide is capable of replacing with advantage the aluminium usually employed in steel works for the purpose of preventing the formation of blow holes in steel ingots. The advantages ensuing from such application are, in addition to the economy secured by the very low price of calcium silicide, very great rapidity of reaction and the suppression of the undesirable 'shortness' which aluminium communicates to ingots of steel." (948,190, Feb. 1, 1910.)

Refining Tantalum.—While heretofore tantalum (for the manufacture of lamp filaments) has been chiefly refined by electric furnace methods, Dr. E. Weintraub, of the General Electric Company, has devised an electrolytic process for refining tantalum. A molten solution of potassium-tantalum fluoride is electrolyzed between an anode of impure tantalum and a cathode of pure tantalum or of other conductive and inert material. The impurities in the anode (carbon, tantalum, oxide, etc.) fall to the bottom of the cell. (947,983, Feb. 1, 1910.)

Electrolytic Furnaces.

Separation of Metals From Mattes or Speisses.—Messrs. E. M. Chance, of Philadelphia, and E. Kent, of Lansdowne, Pa., use the matte or speisse infused form as anode in an electrolytic furnace. The metal to be separated passes into the fused electrolyte and is deposited on the cathode. For instance, if lead, containing 2 per cent to 5 per cent of silver, is to be treated, it is molten and used as anode with a fused electrolyte of double silicate of lead and sodium. The lead passes into the electrolyte and is recovered at the cathode, while the silver re-

mains back at the anode. A second example is the treatment of copper-iron matte with fused sodium and potassium silicate as electrolyte (which may with advantage contain about 2 per cent of manganese sesquioxide to increase fluidity). If the e.m.f. is maintained below a certain critical value, only the copper is dissolved in the electrolyte and deposited on the cathode. (948,681, Feb. 8, 1910.)

Electrolytic Processes.

Electrolytic Amalgamation.—Gold in such a finely divided state that it cannot be recovered by ordinary methods is recovered by an electrolytic amalgamation process of Mr. James H. Alling, of San Francisco, in the apparatus shown in Fig. 7 in plan view, the lower diagram being a broken side view of part of it. 1 is the sluice box, which is sloped at a greater angle than usual sluice boxes. Its bottom contains a continuous amalgamating plate 2, formed at intervals into pockets or troughs 3. Each pocket has a rear wall 4 with considerable slope, a front wall 5 sloping slightly backward, and a bottom substantially horizontal. In each pocket is a quantity of mercury 7. The sides of the sluice box are formed with oblique grooves 8 inclined rearwardly upward, and across the sluice box extend amalgamating plates 9 with their edges in said grooves. These are made anodes. In addition to them there are provided swinging carbon anodes 16 suspended from bars 17 extending across between the tops of the sides of the sluice box, substantially midway between the anode plates 9. The amalgamating plate 2 in the bottom is made the cathode. The material, having been first finely pulverized is fed by a chute 23 into the top

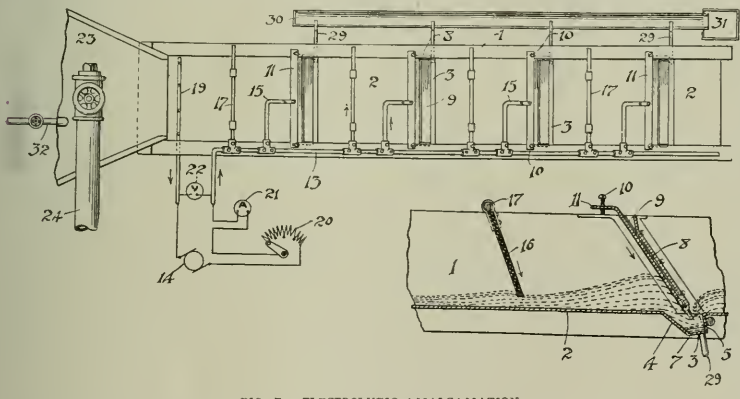


FIG. 7.—ELECTROLYTIC AMALGAMATION.

of the sluice box. At the same time there is fed, by a pipe 24, a sufficient quantity of water, about four tons to 1 ton of material, and through a pipe 32 is fed a small proportion, about 1 to 1000, 2000 or more parts of a mixture consisting of aqueous solutions of potassic cyanides and mercuric chloride. The proportion of the mixture to the water will vary according to the richness of the material being treated, the rule being to supply in the mercuric chloride 1 oz. of mercury to each ounce of gold in the material. In treating certain ores the inventor has found satisfactory proportions by weight to be ½ lb. to 1 lb. of potassic cyanide to 2 oz. mercuric chloride and 5 gal. of the solutions. The addition of the cyanide facilitates the electrolysis and cleans the gold. The density of the current used is comparatively low, being from 1/10 amp to 1 amp for each square foot of cathode surface according to the richness of the material. A tension of about 5 volts is used. It is in order to replenish the aqueous solution of mercuric chloride with mercury that the surface of the anode is amalgamated and since the mercury is continually being removed from the rear surface of the anode plates by electrolysis, as well as by the impact of the particles of gold, it is desirable to replace this mercury as fast as removed, so that there will always be a sufficient coating of mercury upon the anode plates to obtain a proper action of the apparatus. The inventor states that he has been able to save "over 95 per cent of the values of material containing finely divided gold, which cannot be saved by ordinary means." The principal advantages of the system of electrolytic amalgamation over the ordinary cyanide treatment are stated to be the saving in cost of plant and time and labor required in the process. (947,957 and 947,958, Feb. 1, 1910.)

Nickel-Copper Separation.—The old problem of separating the nickel and copper from the nickel-copper-iron sulphide ores of the Sudbury district, which has attracted so many inventors, is the subject of a patent of Dr. Adolphe Chalas, of Philadelphia, Pa. The nickel-copper matte from pyrrhotite smelting is bessemerized to slag off the iron and to concentrate the nickel-copper contents to about 80 per cent. The resulting rich matte is then roasted to convert it into oxides of nickel and copper, which are finally reduced, with production of an alloy of nickel and copper. The resulting nickel-copper alloy is then cast into anodes, which are placed in an electrolytic cell, No. 1, having sheet cathodes of nickel, copper, or iron, and containing as an electrolyte an aqueous solution of an alkali-metal salt, for example, sodium chloride. The anodic and cathodic solutions are separated by a diaphragm. As electrolysis proceeds, nickel and copper are dissolved from the anodes as chlorides, while a sodium hydroxide solution is produced at the cathode. During electrolysis the solution is continuously circulated from the anodic compartment of the cell through a filter, No. 1, containing a body of precipitated nickelous (cuprous) hydroxide, which reacts upon the dissolved copper chloride with production of dissolved nickel chloride and precipitation of copper hydroxide.

The solution thus freed from copper and containing an additional amount of nickel equivalent to the removed copper is mixed with the alkali solution which continuously flows from the cathodic compartment of the cell. These two solutions when mixed react with the production of pure nickelous hydroxide, which precipitates, and sodium chloride brine. The brine, carrying the precipitate in suspension, is circulated through a filter, No. 2, which retains the pure nickelous hydroxide, the clear solution being returned to the cell to regenerate the electrolyte. When the nickel hydroxide in filter No. 1 has been completely dissolved and replaced by precipitated copper hydroxide, the stream of electrolyte from the anodic compartment of cell No. 1 is diverted to a filter, No. 3, which has been filled with fresh precipitated nickelous (cuprous) hydroxide, and the first filter is emptied, cleaned and refilled. When filter No. 2 has also been filled with pure nickel hydroxide it is also replaced by another and is emptied and cleaned for subsequent use. The pure copper hydroxide from filter No. 1 and the pure nickel hydroxide from filter No. 2 may be merely dried or calcined to give pure copper and nickel oxides. The nickelous (cuprous) hydroxide used in filter No. 1 for the precipitation of copper is produced by directly mixing the anodic and cathodic solutions flowing from an electrolytic cell, No. 2, identical with cell No. 1, so that nickel and copper hydroxides precipitate together. The brine, carrying the precipitate in suspension, is circulated through a filter, No. 3, which retains the mixed hydroxides, the clear solution being returned to cell No. 2 to regenerate the electrolyte. The nickelous hydroxide alone reacts with the anodic solution from cell 1.

Commercially five filters or sets of filters are preferably employed, which are worked in rotation; filter No. 1 containing nickel-copper hydroxide is used in the second stage of the process, while filter No. 2 is being filled with nickel hydroxide free from copper, and filter No. 3 is being filled with nickel-copper hydroxide. Filter No. 3 is then substituted for filter No. 1, an empty filter; No. 4 is used to replace No. 2, and the last filter, No. 5, is filled with nickel-copper hydroxide in sequence with No. 3, the process being cyclic and the reagents continuously regenerated. (947,791, Feb. 1, 1910.)

Electrode Construction.—Mr. John William Stubbs of Middlewich, England, patents the electrode construction of Fig. 8, showing sectional views at right angles to each other. Between a pair of copper bars *a* are disposed at intervals the cross-bars or nipples *c* of carbon. Lead *b* is then cast around the carbon nipples and between them and the copper bars *a*, thus attaching the carbon nipples firmly to the conductor bars, both mechanically and electrically. The copper bars are then coated with tar, pitch, paraffin wax, asbestos impregnated with oil, or some other impervious insulating material to prevent corrosion of the copper by the electrolyte, and are finally covered with Portland cement *g*. The ends of the carbon nipples are furnished with carbon blocks *h*. The electrodes are evidently intended for sodium chloride electrolysis. (947,358, Jan. 25, 1910.)

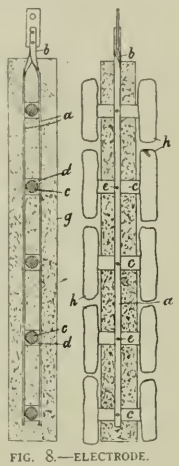


FIG. 8.—ELECTRODE.

Charging Device for Electric Furnaces.—The construction described and illustrated by Taussig on page 46 of our January issue, is covered by a recent patent of A. Helfenstein. It relates especially to the broad openings or slots which are provided in the roof of the furnace around the charging hopper. These openings are usually tightly closed by sand-sealed covers, but they permit access to and guarding of the heating zone and entrance of the mixture in the electric melting hearth also during the charging period. They are useful for the removal of slag and crusts, for ascertaining any breaking of the electrode and for removal of the broken part. The openings permit also to introduce ingredients to correct the mixture, when disturbances occur which require an interruption of the working. They also act as safety valves in cases of too high gas pressure, avoiding so an escaping of gas through the high mixture pile in the charging pit and the charging tubes. By these slots the closed furnace possesses all advantages of an open furnace without the disadvantages of the same. For further details see Taussig's paper. (942,068, Dec. 7, 1909.)

Electric Discharges Through Gases.

Electric Furnace for Producing Nitric Acid from Air.—A peculiar furnace construction which has some points in common with that of Birkeland and Eyde, and also with that of the Badische Company, but differs essentially from both of them, has been patented by Mr. F. I. duPont (E. I. duPont de Nemours Powder Company). He uses a long tube of quartz with electrodes in both ends, between which the arc is formed. But this arc is not straight as in the furnace of the Badische Company, but is deflected and kept in rotation by means of various rotary magnetic fields. For this purpose a number of magnet systems are placed one above the other in such a way that in any system the polarities are reversed as compared with the adjacent magnet systems above and below. In this way the arc is caused to rotate and to be blown from side to side of the tube. The air, while flowing through the tube, has to pass through the arc as many times as the arc crosses over the tube, and between passages it is cooled by contact with the quartz tube, which is cooled from the outside. (943,661, Dec. 21, 1909.)

Synopsis of Metallurgical and Chemical Literature.

Sintering of Iron Ores.—Dr. J. Savelsberg, in German patent 210,742, makes an interesting proposition for the sinter-roasting or plain sintering of fine iron ores, manganese ores, pyrites, cinders, flue dust and other metallurgical materials. He mixes them with a small amount of waste fuel, and "hessemer" roasts them in the well-known Huntington-Heberlein converter, used so successfully for self-roasting of lead sulphide ores. The fuel may be coal dust, coke breeze, lignite, peat, waste wood, etc., and 8 to 15 per cent of the weight of the ore is sufficient for the sintering. The sinter roasting burns out the fuel and sinters the material together to cakes which may be broken into any desired size, and which are very well suited for blast furnace reduction. (*Métallurgie*, Oct. 8, 1909.)

Blast Furnace Slag for Cement Making.—In the Lond. *Elec. Review* of Feb. 11, a description is given of the use of the Colloseus process for utilizing the slag of the blast furnace in cement making at the Coltness Iron Company works at Newmains, Lanarkshire. This is the first plant of its kind in Great Britain. The blast-furnace gases are utilized for operating three gas engines, two of them being for blowing purposes and the third for generating electric power for operating the ball and tube mills, conveyors, etc., in the cement mill.

Rotary Extractor for Precious Metals.—In the December, 1909, issue of the *Journal* of the Chemical, Metallurgical and Mining Society of South Africa, W. D'Arcy Lloyd and Ernest T. Rand point out that from time to time attempts have been made to precipitate gold and silver from their cyanide solutions while the precipitating agent was in motion, with more or less success. For example, zinc shavings have frequently been subjected to mechanical motion while in contact with the gold and silver cyanide solutions and in most cases as far as actual precipitation of metals was concerned, some success has been achieved, but mechanical difficulties have developed due to the method of applying motion by attrition, etc. The main idea underlying all the attempts to perform the operation of precipitation with the precipitating agent in motion is to avoid the fouling of the zinc, which is so painfully evident in the first compartments of all extractor boxes after a three days' run from time of dressing. It is well known that after thorough washing and redressing, the zinc will go on precipitating as briskly as ever until fouling starts again. The natural inference from this is that if the zinc could be kept continuously in motion, thus preventing fouling, the highest efficiency would be maintained. With this point in view, the authors devised a rotary extractor machine, which will be described in greater detail in a later issue, and for which on the basis of the experiments the following advantages are claimed: (1) Ready access by authorized persons to the gold slimes, without stoppage of precipitation, enabling a daily clean up to be made if required. (2) With clippings and coarsely cut filiform zinc, absence of breaking down and readiness of removal of adhering gold in an acid bath without consumption of all the zinc, with a consequent saving of zinc and acid. (3) Clean-up quick and cleanly with a consequent advantage of a minimum of loss. (4) Shortness of time rotor is out of action during dressing. (5) No fouling of zinc. (6) While in operation no danger of theft of gold slimes, as all valves would be locked. In the discussion which followed Mr. A. F. Crosse pointed out another advantage, namely, the mechanical driving off of the hydrogen bubbles from the zinc, since this removal of hydrogen helps precipitation.

An Electrolytic Zinc Process.—For some time reports have

been received on a new electrolytic process of producing zinc from ores, in which the Siemens & Halske Company is interested. This has now been described by V. ENGELHARDT and M. HUTH in *Metallurgie* of Jan. 8. The authors emphasize that if an electrolytic zinc process shall be successful the process must be very simple and the apparatus cheap and durable. The main difficulty is to maintain the zinc salt solution always in good, pure condition, since when it is fouled the deposit of the zinc becomes spongy, and besides the zinc deposited on the cathode is redissolved by the free acid. Only as long as the zinc salt solution is kept in great purity, the redissolving of the deposited zinc by the solution can be prevented. The chief difficulty of keeping the solution pure when using a sulphate solution is the selection of a proper anode material. Platinum is out of the question on account of the cost. Carbon anodes are quickly destroyed, while from lead anodes or hard lead anodes such considerable quantities of lead pass into solution that the electrolytic zinc deposited on the cathodes becomes impure and loses its resistivity against the acid. The iron oxide solutions which have been found successful in sodium chloride electrolysis are also attacked by the sulphate solution and the deposition of pure zinc is no longer possible. The first anodes that were found successful for zinc sulphate electrolysis were the lead peroxide anodes of Ferchland. By the use of these anodes it is possible to obtain a pure zinc which is not attacked by the solution even if it contains as much as 10 per cent of free sulphuric acid. In an experimental plant in Upper Silesia sulphate electrolysis with Ferchland lead peroxide anodes was carried out on a larger scale for about a year. The process was found to be technically successful. The losses of zinc are three times greater with zinc smelting than with the electrolytic process. Further, the operation showed that the purification of the solution to the desired high degree could be carried out easily and without great cost if certain precautions were taken in the making up of the solutions. With the exception of some small troubles in the beginning, the electrolysis itself was quite successful. The electrolytic zinc has a purity of 99.98 per cent to 99.99 per cent. The energy consumption was 3.9 kw-hours per kilogram of zinc. This relatively high power consumption was due to the small mechanical durability of the anodes, causing both an increase of voltage and a reduction of amp-hour efficiency. With a more durable anode material the authors figure that the energy consumption would be 3.3 kw-hours to 3.5 kw-hours per kilogram of zinc. The cost of operation per 1000 kg of electrolytic zinc was $39.25. This figure includes cost of power (0.625 cent per kw-hour), wages, material for purification of solution, remelting the electrolytic zinc, general cost, and license. But the figure does not include amortization or cost of ore. Two-thirds of the figure of $39.25 is the cost of electrical energy, so that the authors think that with very cheap power the production of 1000 kg of electrolytic zinc could be reduced to about $19.25. (Under the condition of power cost in this country this would hardly be possible, since a cost of 0.625 cent per kw-hour is already very low.) An equally excellent material for anodes for sulphate electrolysis, to be used as a substitute for the lead peroxide anodes of Ferchland, are the manganese peroxide anodes made by the process of Siemens and Halske. This material used as an anode in a sulphate solution is absolutely insoluble and is far more durable than lead peroxide anodes. Experiments made with manganese peroxide anodes for over a year have shown that these anodes will last for several years. While the Ferchland lead peroxide anodes can be made only in stick form, the manganese peroxide anodes can also be made in plate form. In spite of the comparatively successful results obtained in the plant in Upper Silesia, the process has not been introduced on a large scale. The reason is stated to be that the electrolytic process when carried out with steam power and when treating expensive high-percentage ores is economical only if the very pure electrolytic zinc will demand a considerably higher price than the commercial spelter. At present there is a margin of $50 per ton for very pure zinc. But whether this margin could be maintained for very pure zinc produced on a large scale is doubtful and it would be injudicious to build a plant on the basis of such expectations. A small margin will, however, always be maintained since the cost of rolling is considerably lower with the pure zinc on account of its great softness. The authors expect, however, that the electrolytic process can be used safely with an expectation of profit in all cases in which poorer and cheaper ores or zinc scrap are to be treated

Protective Lime Diaphragm On Cathode; Production of Hypochlorites and Chlorates.—In the electrolysis of aqueous solutions for the reduction and oxidation of chemical compounds it is important to prevent the cathodic reduction products from being oxidized again by contact with the anode and to prevent the anodic oxidation products from being reduced again at the cathode. The old ordinary diaphragm cell has distinct disadvantages in many cases. A modification—in which a diaphragm close to and adhering to the cathode is produced automatically during electrolysis in the cell—has been found useful for the production of hypochlorite and chlorate. Thus H. Bischoff and F. Foerster found in the electrolysis of alkali chlorides for hypochlorite production that an addition of calcium chloride greatly diminishes the cathodic reduction of the hypochlorite formed; this is due to the formation of a layer of lime on the cathode which prevents direct contact between the hypochlorite and the cathode. E. Mueller has later shown that an addition of potassium chromate is even more effective; it results in the formation of a chromium oxide diaphragm which prevents any cathodic reduction whatever. The use of an addition of calcium chloride in the commercial manufacture of hypochlorite and chlorate has failed because the diaphragm formed at the cathode would not stick to it with sufficient rigidity. E. MÜLLER and M. BUCHNER have now found (*Zeit. f. Elektrochemie*, Feb. 1) that they can produce a firmly adhering lime diaphragm on the cathode by adding a very small quantity of a colloid (gelatine, etc.) to the solution to which calcium chloride or chromate has been added. This effect of the colloid seems to be the same as in the Betts lead refining process, where the application of a very small amount of a colloid results in a firm adhesive deposit of lead.

Ammonia-Soda Process.—A recent lecture by Prof. A. COLSON, of Paris, before the Society of Chem. Ind. in London is noticed in Lond. *Eng'ing*, Feb. 11. "Prof. Colson is manager of the St. Cobain Alkali Works, in which the ammonia-soda process is used. Professor Colson dealt in his lecture with the present state and possible modifications of the ammonia-soda process. The lecturer stated that the value of the soda produced by the process was about £2,000,000 annually: owing to the salt duty, France produced only $1,500,000 worth—less than England and Germany. Four operations might be distinguished: The decomposition of limestone into quicklime and carbon dioxide; the precipitation of sodium carbonate from ordinary salt solution by means of ammonia and carbonic acid; the calculation of the sodium bicarbonic to carbonate (soda): and the regeneration of ammonia from the chloride formed by means of caustic lime. The ammonia played an intermediate, though most important part. The total heat balance of the whole reactions was the evolution of 5200 calories, but 44,500 calories were required for decomposition of the limestone and coal was hardly required for the other reactions." There were many difficulties in the different operations more perhaps for the engineer than for the chemist, for the chemistry was well understood. For instance, the pumps drawing off the carbon dioxide were rapidly destroyed by the gas and the flue-dust. The precipitation of the bicarbonate had to be rapid lest a double carbonate of ammonium and sodium be formed or some ammonium chloride be precipitated with the carbonate; the latter would be destructive to the cal-

ening plant. The regeneration of the ammonia was now effected by quicklime, not by milk of lime, and the ammonia was absorbed by water. The operation took place in column apparatus, such as were used for the rectification of alcohol, but they were much larger, as volumes of 15 and 20 cubic meters had to be dealt with per hour. In one case gas discharge pipes of insufficient cross-section had caused very serious trouble. It was also most essential to secure actual contact between the lime and the ammonium chloride. In some French works solid salt was the raw material of the soda; in others brine was used, which was exclusively applied in England. The process as worked in various Solvay works was now considered almost perfect, especially after the introduction of machinery to replace manual labor; that, however, required large capital. More than 2.5 kilogrammes of ammonia should not be lost in the cycle of operations if the process were to pay. Professor Colson then explained that his various experiments had been conducted not so much with the object of improving the various operations as with the object of suppressing some of them. He proceeded to show how this could be done when sodium nitrate was substituted as raw material for the chloride (ordinary salt). The by-product would then be a mixture of sodium and calcium nitrates, which was as valuable almost to agriculture as the original Chili saltpeter. "The discussion of the paper was disappointing. One gentleman employed in the ammonia-soda industry said that he had almost been afraid that Professor Colson might give away some of the secret manipulations by which the troubles were overcome. One cannot help feeling that the discussions in our engineering and electrical societies, in which trade secrets used to be so jealously guarded, have of late years risen to a higher level."

Copper-Aluminium-Manganese Alloys.—The 8th report to the alloys research committee of the (Brit.) Inst. Mech. Eng., by Carpenter and Edwards, dealt with copper-aluminium alloys. This has now been followed by the ninth report, by W. Rosenhain and F. C. A. H. Lantsberry, of the Nat. Phys. Laboratory, on the effect which an addition of manganese has on the properties of copper-aluminium alloys. Broadly speaking, the ternary alloys, of which the principal ingredient is copper, when compared with the best binary alloys of copper and aluminium, present certain advantages, having a higher yield-point, a slightly higher ultimate stress, and undiminished ductility. The alloys were tested in the form of sand castings, chill castings and rolled bars. One alloy proved best in one form, while others surpassed it under other conditions. These alloys showed improvement in the rolled form over the chill-castings, especially in elongation. Certain alloys examined by dynamic test prove these particular ternary alloys to be practically equal to the binary alloy containing 10 per cent Al, but they show a very slight inferiority as regards the safe range of alternating stresses. On the other hand, under impact test some of the ternary alloys gave remarkable results—very remarkable ones, indeed—under the abrasion test in Dr. Stanton's machine. As regards corrosion in salt and fresh water, the ternary alloys are equal, and in some cases rather superior to the copper-aluminium series. With regard to the alloys at the light end of the series the results of the present investigation are not so favorable. In the case of one of the series the tests on chill-castings gave good tensile strength and elongation, but rolled bars of the same alloy showed no better results than those obtained with copper-aluminium alone. The temperature at which the metal is poured greatly influences the properties of these alloys. An almost complete reprint of this very long, illustrated report begins in Lond. *Engineering* Jan. 28.

A New Plating Method.—A new method of plating was described by Mr. A. Rosenberg in a recent lecture before the Society of Arts in London. (Lond. *Electric Review*, Feb. 11.) The desired metallic coating is produced by simply rubbing the surface with a paste made of a prepared powder. While the process is therefore simply one of rubbing the metal in, its principle is electrochemical. The rationale of Mr. Rosenberg's process is this: His plating powders are compounded of: First, some form (whether elementary or combined) of the metal to be deposited; secondly, an uncombined electropositive metal, usually magnesium, but sometimes aluminium, zinc or cadmium, in a highly divided state of division; thirdly, a substance capable of producing an aqueous electrolyte when brought into contact with moisture (this ingredient is only necessary when the metal to be deposited is present in the metallic state); and lastly, certain inert substances such as chalk, soapstone, kieselguhr, boracic acid, dextrine, etc., whose function is to act partly as diluents, to prevent premature too rapid action, and partly as polishing materials. It is clear that if a mixture of the above ingredients is moistened and rubbed over a clean metallic surface, an innumerable number of infinitesimal voltaic cells will be produced, and while the electropositive metal present—the magnesium, for example—will go into solution as anode, the metal of the salt present will be deposited on the surface under treatment, which acts as cathode, and as the paste is rapidly rubbed over the whole surface, a continuous metallic coating will be electrodeposited in the literal sense of the term. The following simple experiment may be made as an illustration. Rub a mixture of magnesium power (2 parts), and powdered nickel ammonium sulphate (20 parts) with a moist rag over a copper penny. This will immediately become coated with a bright film of nickel.

Further, to illustrate the reactions, take the mixture recommended for the deposition of zinc: Zinc dust, 15 parts; ammonium sulphate, 5 parts; magnesium, 1 part; chalk, 10 parts; soapstone, 2.5 parts. In this case it is advisable to use the zinc in the metallic form, and both magnesium and zinc will go into solution electrolytically forming their respective sulphates, while zinc will immediately be thrown out of solution again, as the action proceeds, and deposited on the cathode, namely, the object under treatment. The function of the soapstone in this instance is to prevent the mixture from absorbing moisture if exposed to the air. The chalk is useful both as diluent and as a polishing agent. It is further useful as a source of alkalinity, which, just as in ordinary plating, may be valuable in improving the color and quality of the deposit. The object to be plated has to be reasonably clean and free from grease, but the operation itself involves both mechanical and chemical cleaning, and hence the great precautions taken to clean the work in ordinary electroplating can be dispensed with in the new process. At the same time, the brighter and more highly polished the surface to be plated the brighter will be the deposit thrown down. The plating powders are being placed on the market under the name of "galvanit." At present only silver, nickel and tin mixtures can be obtained; but the list is to be extended, and will include alloys like brass and gun-metal, which, it is stated, can be plated with ease. It appears that the mixtures if kept in a fairly dry place do not deteriorate with time on account of local chemical action being set up; this is prevented partly by the presence of the inert substances referred to previously; and partly by coating the magnesium powder with a thin film of paraffin or resin, by immersing it in a volatile solution of these substances. The friction involved in the plating process is sufficient to remove the insulating films from the magnesium when that is no longer required.

Electrode Losses in Electric Furnaces.—In an appendix to his paper on electrode losses, just published in Vol. XVI. of the *Transactions* of the American Electrochemical Society, Mr Carl Hering gives the results of an experimental determination of his mean conductivities and resistivities between 100° C (cold electrode end temperature) and 900° C. (furnace temperature). In the inch units specified in his article in our last October issue they are: graphite $r = 0.000323$, $k = 0.740$; carbon $r = 0.00148$ and $k = 0.350$.

The Wood-Distilling Industry.

By Charles L. Campbell.

The rapid growth of the wood-distilling industry in this country from its start up to within the last two years, and recent developments in this industry in affecting it, taken together with the fact that there is no literature in the English language covering the subject to any extent, would seem to warrant at this time a brief review of the subject and a look at its future.

Historical.—The wood-distilling industry is a very old one, the subject being mentioned in the writings of the old Romans. The distillation was very crude, however, being carried on for the purpose of obtaining charcoal alone, and it was not until 1835 that any considerable advance was made in the industry. At this time Dumas and Poligot, in Europe, investigated the constitution of the vapors given off from the wood during distillation, and recognized the three principal constituents: acetone, methyl alcohol and acetic acid.

Other investigators examined the tar and gases formed, the gas, which usually contains about 30 per cent to 35 per cent of methane, CO, etc., being used by Lebon for lighting in some German towns. These investigations mark the beginning of the modern wood-distilling industry, although the first wood-distilling plant was started at Hausach, in Germany, in 1819, and a man named Adams established a plant at North Adams, Mass., in 1830, where the vapors constituting the pyroligneous acid were recovered.

The coal-tar industry in Europe, which began in 1856, at the time of Perkin's discoveries, marked the beginning of the modern epoch in wood distillation, since the production of coal-tar colors required the use of methyl alcohol and acetic acid and brought about the extensive refining of the pyroligneous acid after its recovery. This industry was, however, confined mostly to Europe, and the subsequent developments in organic chemistry, in which Germany was the leading factor, requiring the refined products of wood distillation, made her the center of this refining industry.

The industry in the United States, however, developed much more slowly, the first wood-alcohol refinery being founded in 1874 at Binghamton, N. Y., by G. C. Edwards. The industry in this country at that time was mostly for the purpose of producing charcoal for fuel or for making charcoal iron, in contrast to the German industry, where the refined products were the objective. It was, therefore, natural that the raw products of the industry in this country should be shipped to Germany, and such was the case, the record of import showing that in 1897 Germany imported an amount of raw materials from the wood-distilling industries equal to twice her production. This state of the two industries, American and German, has continued up to the present time, America exporting in 1908 one-third of its production of raw wood alcohol and large amounts of acetate of lime, but a change has been coming about and indications point to decreased exportation from and increased refining in this country.

Present Methods.—In order to understand where the industry in this country stands at present and the opportunities open to it in the future, a brief outline of the process used in the great majority of American plants will be given. In this country at the present time there are probably about 150 plants, ranging in size from 500 to 250,000 cords per day. In 1907 the following plants reported their yearly consumption and output to the Department of Agriculture:

States.	Number of plants.	Cords per year
Michigan	13	602,000 cords.
New York	28	127,000 "
Pennsylvania	53	358,000 "

However, while these three States support the majority of plants in the wood-distilling industry, yet there are many plants in other States which were not reported to the Department of Agriculture. This list also does not include the plants distilling pine woods for the purpose of obtaining oil of turpentine.

Of the processes used in these plants there are a great many modifications, and it would take a good-sized volume to discuss them all, but a thorough investigation has shown that of these special processes a majority have either failed entirely or worked unsatisfactorily.

There are two distinct processes in use at the present time, i.e., destructive and steam distillation, but the latter is a small industry as compared with the former and cannot be considered at this time. The destructive process in general use consists in heating the wood either by external heat or by partial combustion so that the wood-fiber material is broken down, the products yielded being:

(1) Charcoal.
(2) Tar.
(3) Watery distillate, pyroligneous acid.
(4) Uncondensable gases.

The object of the distillation is the recovery of one or all of these products, the amounts of same being dependent on the wood and the process used.

The destructive distillation process for which the principal kinds of wood used are beech, birch, maple and oak, and may be divided, in turn, into two distinct parts, the "kiln" and the "retort" processes. In the "kiln" process the primary object of the distillation is the production of charcoal, usually for making iron, and the center of this branch of the industry is in Michigan. The kiln is a bee-hive shaped brick retort, holding anywhere from 15 to 120 cords of wood, the kiln being packed by hand and heated by burning the wood with an insufficient supply of air, or by burning the uncondensable gases formed during the distillation, the kiln in this case being fitted with heating pipes in its interior and sometimes lined with metal. The time of distillation may be as much as three weeks, and the vapors which are given off are usually recovered.

In the "retort" process the object of the distillation is usually the recovery of charcoal and pyroligneous acid, as well as tar and gases. The retorts are made of iron, and are set in brickwork. They are of all shapes and sizes, the main types being:

(1) The small, upright, removable retort, packed by hand.

(2) The small, horizontal, cylindrical retort, fixed in position, and filled and emptied by hand.

(3) The large, horizontal, square or cylindrical retort or oven, fixed in position, and filled and emptied mechanically, the wood being in most cases packed on cars and these cars being wheeled into and out of these retorts.

Of these retorts type No. 1, the small, upright oven, is probably most used. These retorts may be heated by coal, wood or tar, or by the gases obtained in the distillation.

The vapors given off during the distillation are condensed by devices of some sort, various arrangements being made, but the main types are the coil, box and upright tubular condensers. The uncondensed gases are led by suitable pipes to be burned under the retorts without further treatment. The crude pyroligneous acid obtained contains approximately the following constituents:

Tar and tar oils	9.8 per cent.
Acetic acid	10.0 "
Methyl alcohol	3.0 "
Acetone	.2 "
Water	77.0 "

also propionic, valerianic and butyric acids, allyl alcohol, aldehyde, furfurol, ammonia, methyl acetate, methyl-ethyl ketone, amines, and pyridine bases, etc.

Further processes have to do with the removal of the various products from each other, and conversion into a commercial form. The tar and oils are removed for the most part by allowing the crude pyroligneous acid to settle, by which operation three distinct layers are formed, the lower one con-

sisting of tar with creosote oil and some pyroligneous acid; above that the pyroligneous acid containing considerable amounts of dissolved tar; and on top the lighter hydro-carbon oils.

By a proper arrangement of taps these different layers can be separated from each other and collected in separate tanks ready for subsequent operations.

The tar-containing pyroligneous acid is re-distilled in a steam-jacketed or coil-heated pan, by which operation the tar is left as a residue and vapors containing acetone, wood alcohol, acetic acid, water, and slight amounts of other impurities are formed. These vapors are passed into a milk of lime solution containing about 20 per cent CaO, by which process acetic acid is removed from these vapors and a weak solution of calcium acetate with some tarry impurities is obtained.

The unabsorbed vapors, consisting mainly of alcohol, water and acetone, may be condensed to give a weak alcoholic solution, or may be passed through a concentrating apparatus, consisting usually of water-cooled bonnets, to give a 35 per cent to 40 per cent solution of wood alcohol. The solution of acetate is heated to remove the last traces of alcohol, the alcoholic vapors so formed being united with the others, and the impure solution is then filter-pressed to remove dirt, excess lime, etc., and finally evaporated, first in comparatively small, open, steam-jacketed pans until a thick paste or moist cake is obtained, then in open, shallow iron pans set in brickwork over the retorts until an acetate of lime of 80 per cent to 82 per cent, containing a small percentage of moisture, is obtained.

The manufacture of brown acetate by direct neutralization of impure, settled, pyroligneous acid, with milk of lime, is unimportant, the amount of this acetate manufactured being equal to only 1 per cent of the amount of gray acetate.

The 35 per cent to 40 per cent impure acetone-containing wood alcohol obtained as described is further concentrated in a column still heated by steam coils to give an impure 80 per cent by volume solution, which is not refined at the distilling plant, but in special refiners. The charcoal, which is left in the retort, is removed and cooled in a different manner according to the character of the oven used. With the small, removable retort the whole apparatus is lifted out of its setting and the charcoal either left to cool in the retort itself or dumped into a cooling receptacle. With the small, horizontal fixed retorts the charcoal must be removed by hand from these retorts and placed in a closed receiver to cool. From the large horizontal retorts, however, the charcoal may be removed on the cars into cooling receptacles of suitable size, and when cool may be dumped from these cars where required.

The tar, whether settled out of the pyroligneous acid or removed from it by distillation is in the majority of plants burned under the retorts by suitable arrangements.

This outlines the procedure required for the production of the principal products—gray acetate of lime of 80 per cent to 82 per cent; crude wood alcohol of 80 per cent; charcoal, tar and oils.

These products are then sold to various consumers, either to chemical manufacturers and refiners, iron men and fuel consumers, or for export. Of these products the acetate of lime is used for the production of acetone, acetic acid and various acetates; the crude wood alcohol for the production of formic acid; the refined wood alcohol, pure methyl alcohol or denaturing spirits, and the tar and oils for various purposes.

The amounts of the various products yielded in the distillation process per cord of wood depends, as was mentioned before, upon the kind of wood and the process. This can be shown very well by comparing the average yields returned by the plants of Pennsylvania, where the "retort" process is used, and the object of the distillation is the recovery of all of the product; with those returned by the plants of Michigan, where the "kiln" process is used, and the object is the production of charcoal for use in making charcoal iron. The following figures show this:

State	Process used.	Products per cord.
Pennsylvania	Retort	200 lb. gray acetate of lime.
		10 gal. crude alcohol (80 per cent)
		50 bu. charcoal.
Michigan	Kiln	50 lb. gray acetate of lime.
		4 gal. crude alcohol (80 per cent).
		40 bu. charcoal.

Financial Situation.—The previous outline has given the main details of the process used in distilling wood in America to-day, and in order to understand the present situation of this industry as an investment, the following figures are given, these figures showing the average cost per cord of wood for the years 1906, 1907 and 1908; also the value of the principal products produced during that time.

	1906	1907	1908
Wood per cord (sawdust included)	$3.24	$3.14	$3.09
Wood per cord (sawdust not included)	3.69	3.62	3.57
Charcoal, per bushel	.065	.076	.071
Crude alcohol, per gallon	.340	.149	.172
Gray acetate, per pound	.021	.019	.015
Brown acetate, per pound	.012	.012	.008
Iron acetate, per gallon	....	....	.095
Oils, per gallon	.080	.024	.066

From this table it can be seen that while the price of wood has decreased a very little, the value of the products produced has decreased a very great deal, notably in the case of the crude wood alcohol.

Now this decrease in the value of the products of the wood-distilling industry is due to two factors:

(1) The financial depression of 1907.

(2) The passing of the denatured alcohol acts of 1906 and 1907.

The price of refined wood alcohol was in 1906 about 75 cents per gallon, while that of grain alcohol was about $2.40 per gallon. There are many chemical processes, such as, for instance, the hat and varnish industries, where either refined wood alcohol or grain alcohol can be used, and the cheaper is, of course, preferred. When the denatured alcohol act was passed the price of grain alcohol denatured for industrial purposes was considerably below 75 cents per gallon, and, of course, replaced wood alcohol where possible. The demand for this wood alcohol having diminished, its price naturally fell and carried with it the price of crude wood alcohol, acetate of lime and other wood products, and drove a number of wood distillers out of the business, at the same time seriously threatening the whole industry in this country. At the present time, however, it has recovered somewhat, and the outlook is bright for the future, as will be shown later.

One, and the principal denaturant, prescribed for grain alcohol is wood alcohol, containing certain impurities, and a large amount of such wood alcohol is consumed in this country. The use of denatured alcohol has not become, however, so general here as was expected, but it is probable that there will be a large increase in consumption in the future and that the consumption of wood alcohol for denaturing will be such as to use up all of the supply available for this purpose, thus restoring the prices to their former level. This situation is exactly like that in Germany 20 years ago, when the denatured alcohol act proved a boon rather than a deathblow to wood distillers and refiners.

This was for the reason that there are many purposes, for instance, in the aniline industries, for which very highly refined wood or methyl alcohol only is suitable, in which industries, therefore, the passing of the denaturing alcohol act had no effect upon the demand for wood alcohol. The use of denaturing spirit which required, as mentioned above, a very impure wood alcohol, containing various impurities, such as acetone, allyl alcohol, pyridine bases, etc., thus permitted the pro-

duction of a very pure spirit by distillation and permitted the utilization of the impure first and last runnings as denaturing spirit.

It can be seen that this was a very economical process, since such pure, refined methyl alcohol commanded a very high price, and the demand for denaturing spirit was such as to cause this to sell at a considerable profit. While industries which demand this pure alcohol have not been developed to a great extent in this country, yet there is every prospect of great development along these lines and this fact, together with the prospect of increased consumption of denatured alcohol and the probability that the amount of raw spirit produced here will not be increased, at least for a good many years, makes the outlook for wood-alcohol producers and refiners very encouraging.

While, therefore, it is thought for this and other reasons, particularly the situation of the acetic-acid industry, that the depression in the wood-distilling industry is but momentary, yet recovery will take several years, and in the meantime it will be necessary for the wood distillers to make their processes profitable. This can be done in various ways, which will be outlined. It is a fact well known that the wood-distilling industry has been conducted, at least in this country, by men unacquainted for the most part with the fundamental principles of chemistry and engineering, who have been unable or reluctant to introduce any considerable improvements into their processes, and as a result have been using up to-day the methods of 30 years ago. With the increased costs of fuel, labor and raw materials it is evident that improvements are necessary to maintain the industry in its proper position, but in the wood-distilling industry it is notably the case that improvements which advances in the chemical industry should have made possible have not been introduced.

Improvements.—The rest of the article will be taken up with an outline of the opportunities open to distillers to so improve their plants or processes as to increase their returns in accordance with the following conditions, or to extend their business to more profitable fields. The conditions which are necessary are that they must either

(1) Increase their yield of products or the quality of the same;

(2) Reduce the costs of production either as regards fuel, labor or apparatus.

Each of these conditions will now be discussed separately.

(1) *Increasing the Yield.*—The yield depends on:

(a) The kind of wood.

(b) Condition of the wood.

(c) The form of retort and setting.

(d) The method of heating.

(e) The methods of recovering and treating the products.

As regards the kinds of wood, this method cannot change, but a proper selection will have much to do with the yield obtained. The condition of the wood, however, is very important. The moisture in the wood not only requires heat for evaporation, but, when the temperature of carbonization is reached, causes side-reactions, which result in the formation of gas instead of liquid vapors, thus decreasing the yield of pyroligneous acid to a considerable extent. In this connection only wood which has been air-dried for a year at least should be used, but a further improvement, recently introduced, consists in artificially drying and forewarming the wood before carbonizing, but after packing on the cars, if the large retort is used, by placing these cars in ovens through which the warm or hot-fire gases are passed, for 24 hours before introduction into the retort. This is done practically free of cost.

The form of retort and the setting is very important. In the first place, only large retorts which can be filled and emptied mechanically should be used. This results in a great saving of time and labor. Again, these retorts should be so placed in the setting that they can be heated uniformly over the whole length, and the heating should be so controlled that the temperature is maintained at a constant point and the heat applied slowly. It has been proven by many experiments that rapid heating results in the formation of larger amounts of gas and smaller amounts of pyroligneous acid, while slower heating decreases the yield of gas and increases the yield of acid.

The methods of recovering the products influence the yield to a considerable extent, and in this connection there should be mentioned the use of an exhauster to create a slight vacuum in the retort, thus preventing leaks, and the use of a scrubber whereby the uncondensed gases are caused to pass through a tower and are deprived of any pyroligneous acid vapor which they may contain. This may seem a small matter, but it has been found that by the use of such apparatus the yields of acetic acid and alcohol can be increased 3 per cent to 4 per cent.

(2) *Decreasing the Costs of Operation.*—To decrease these costs there may be used any appliances which tend to:

(a) Decrease the amount of labor required.

(b) Decrease the amount of fuel required.

(c) Decrease the cost of apparatus and depreciation on the same.

We have already considered these items in connection with the carbonizing itself, recommending large retorts, mechanical loading and unloading, together with the use of artificially dried wood as savers of labor and fuel in this part of the operation. Regarding methods of decreasing the costs of labor, fuel and depreciation in the remainder of the wood-distilling operations it might be well to describe the appliances and methods which can replace those now in use, and to then take up the effect of using this apparatus on the above-mentioned costs.

Practically all of the evaporation required in the operation subsequent to the carbonization is accomplished by the use of high-pressure steam, and of this steam required approximately 40 per cent is needed to re-distill the pyroligneous acid to remove the 8 per cent of tar contained in it. Now if this tar could be separated from the pyroligneous acid vapors as they came from the retort without condensation, and the tar-free vapors then passed into the milk of lime, this 40 per cent of fuel would be saved. Such an apparatus has been devised and has been in operation in many plants abroad and a very few in this country for over two years. It has actually been found to almost completely separate the tar from the vapors.

This apparatus, which operates on practically the same principle as a fractional distilling column, yields liquid tar free from acid and a tar-free pyroligneous acid vapor, and is so constructed that it requires no troublesome hand cleaning. The apparatus is not known to any extent in this country, but is destined to occupy a big field. Because of the economy of steam, it permits the formation of a very pure acetate of lime, does away with the condenser and tar still, with all the trouble of cleaning the same. This apparatus occupies little space, is simple in construction, and yields a tar which can be used for producing creosote and other tar derivatives without further treatment.

Another very important appliance, but very recently introduced, is an apparatus by means of which the acetate of lime solution containing traces of alcohol is de-alcoholized, and at the same time the weak alcohol vapors concentrated to give a product, directly, of 90 per cent to 95 per cent, and requiring only the use of the heat of the alcohol vapors and sometimes exhaust steam to accomplish the evaporation. This apparatus works on the principle of the continuous distilling apparatus used in the grain-alcohol distilleries of to-day to a large extent, the wood alcohol produced being analogous to the high-proof spirit delivered from these continuous distillation columns and the de-alcoholized acetate of lime solution to the "slop" from the same.

Special modifications are required in this apparatus, however, to meet the new conditions, but it has been found to work very satisfactorily indeed. In operating this apparatus, the acetate of lime solution is introduced into the first distilling column

after passing through a "heater," while the weak alcohol vapors are passed into the bottom of the same column, thus furnishing the necessary heat for the distillation. The vapors of wood alcohol and water, passing from the top of this column, then go to a second rectifying column and are concentrated to the required proof.

By means of this apparatus and the tar separator, it is possible to do away with the customary "three-still system," also with the wood alcohol concentrating and rectifying apparatus, and by use of the latent heat of vaporization of the weak alcohol vapors for the de-alcoholization of the acetate of lime solution, a saving of fuel amounting to 20 per cent is effected. Besides the saving in fuel, there is a saving in space and labor required, as well as a decreased cost for the first installation.

In the evaporation of the acetate of lime solution great improvements have also been made. The customary method of carrying out this evaporation, as previously described, in open pans, required high-pressure steam, much hand labor for transferring the material and cleaning the pans from incrusting materials, and gave a poor product if not carefully carried out. To do away with this, an apparatus, essentially a form of triple effect with special devices to meet the requirements, has been introduced, and in consequence of the use of a vacuum, the use of exhaust steam is possible.

This apparatus, working continuously, takes the weak of acetate of lime and delivers it in the form of a thick mush. In consequence of the fact that this apparatus permits the use of exhaust steam from the engines instead of the high-pressure steam required by the former method, there is effected a saving of approximately 100 per cent in the amount of fuel required for this operation, while the labor cost is greatly diminished and a better product is obtained. It is needless to say that this apparatus requires less room than that required by the open-pan method, and it is also a fact that the initial cost and depreciation are less in consequence of the fact that no cleaning and scraping are required on the same.

Further than this, improvements have been made in the method of drying this acetate of lime. Where it was formerly dried in the open pans over the retorts, rotary dryers, especially designed for the purpose, can now be obtained. By the use of a dryer heated by superheated steam the mushy acetate of lime is dried to give a moist cake containing 25 to 30 per cent water, which cake is then thoroughly dried in a second apparatus, also rotary, through which hot air, heated in turn by the hot retort gases, passes, and the dry acetate of 80 to 84 per cent, nearly white in color, is delivered by an elevator to a chute from which the sacks can be filled. The essential saving effected by this last apparatus is in labor and space. The labor required is never over 30 per cent of that required by the open-pan method, while the space occupied by the dryers is very much less.

Summarizing the results which may be obtained by the installation of this improved apparatus, it may be said that that amount of coal required for steam production in the plant will be reduced to 35 or 40 per cent of that required by the present customary process, the labor to one-half, and the initial plant cost for a fifty cord per day plant to two thirds that of plant operating under the present methods, while the cleanliness of the plant and space required for the apparatus is also affected. In every case the improved apparatus operates on principles identical with those employed for different products in other industries, with suitable modifications to meet the conditions.

It is evident that the old method now used to a great extent cannot compete for a minute with this improved process. As regards the yields, these are also increased by approximately 10 and 15 per cent. While these improvements have scarcely been introduced into this country, they have been thoroughly tested out abroad, and several plants, notably one large plant in Michigan, distilling over 200 cords per day, are to be remodeled to employ these processes during the coming year. It is to be hoped that persons engaged in wood-distilling industry in this country will not hesitate to adopt these methods and put themselves in position to compete with those who are awake to their opportunities.

* * * * *

The foregoing has related to the manufacture of the crude products of the wood-distilling industry, but the field in which extensive profits are now being reaped and where more extensive profits will be reaped in the future is in the manufacture of the finished products from these raw materials, that is, more particularly, acetic acid, refined wood alcohol, methyl alcohol, acetone, tar derivatives, and various other products, such as red and black liquors, sugar of lead, sodium acetate, etc.

Acetic Acid.—This acid, manufactured commercially from acetate of lime and a strong acid by various processes, is used in the coal tar color industry, in the dyeing and mordanting industries, in the manufacture of artificial vinegar, etc. The manufacture of this acid under improved and economical processes, yielding acid which gives the highest returns, has been in the hands of five large chemical factories up to this time, and it has been a well-known fact that the business is very profitable.

At the present time wood-distilling people producing sufficient acetate to keep an acetic acid plant of economical size in operation in localities adjacent to the large cities, for instance in Pennsylvania, are offered unexcelled opportunities in this industry. There are a number of small firms who have started plants making acetic acid, but without exception they have installed the process using small fire-heated decomposing retorts, and inefficient methods, which require the use of a large excess of raw materials and produce an inferior product, besides increasing the amount of labor required, etc.

An American firm, E. B. Badger & Sons Co., of Boston, has undertaken the installation of acetic acid plants, employing the most improved and efficient methods, notably the use of very large decomposing retorts and a vacuum system such that the fuel cost is decreased, better regulation made possible, a very much better product produced, and increased yields obtained, these improvements being sufficient to make the difference between small and large profits.

Wood distillers who follow the example of two or three plants in Pennsylvania now installing such apparatus will be well repaid. The above-mentioned firm is also in a position to supply the necessary apparatus for the following industries, in all of which they furnish the most improved type:

Refined Alcohol.—The refining of wood alcohol and the production of pure methyl alcohol is an industry where there is great opportunity for improvement in the methods and apparatus used in the majority of plants, also where wood distillers have a good chance for investment.

The refining of crude wood alcohol in this country has been controlled by a syndicate, and what they did not refine has been shipped abroad. The following table of exports shows that a large amount has been exported, also shows the drop in value of this alcohol within the last two years.

Year.	Quantity Exported.	Value.	Value Per Gal.
1902	627,000 gals.	$338,000	$0.54
1903	834,000 "	453,000	.54
1904	1,194,000 "	585,000	.49
1905	1,097,000 "	603,000	.55
1906	781,000 "	466,000	.60
1907	2,150,000 "	862,000	.40
1908	1,958,000 "	819,000	.42

As may be observed, the higher the price, the less alcohol exported, and vice versa.

With the present low price of crude alcohol and the demand in this country for refined wood alcohol and pure methyl alcohol, and probably soon denaturing spirit, it is absurd for distillers of sufficient size to support a refinery to sell their

crude product. They should install a refinery of their own independent of the trust, as several plants have done to their great advantage. In this industry, however, as in all others, it is necessary in order to conduct it profitably to use improved methods and apparatus. In the manufacture of refined wood alcohol, that grade known as English wood spirit is a spirit obtained by concentrating the crude wood alcohol to such a point that it contains only a small amount of water, not over 5 per cent, but still contains many of the impurities of the crude alcohol.

In the manufacture of this grade large continuous apparatus may be very profitably employed, but it is very seldom that a new plant installs such an apparatus, it being a common occurrence to see plants using small apparatus and running their products through two or three times to get the necessary concentration, thus increasing the fuel, labor, and depreciation costs. The continuous apparatus will take the crude wood spirits and turn out the spirit of concentration desired, all in one operation.

In the production of pure methyl alcohol, it is seldom profitable to employ continuous apparatus, since the distillate must be collected carefully between certain limits determined by boiling point, odor and miscibility with water, and must be treated with various chemical re-agents in order to remove the contaminating products. However, the use of large apparatus effects a considerable saving here. Under present conditions the manufacture of this high-grade methyl alcohol, for which there is considerable demand at high prices, offers very good opportunities, since the impure spirit obtained in the process, which amounts to 60 to 70 per cent of the crude spirit used, is available for denaturing at good prices.

Acetone.—The manufacture of acetone, extensively used as a solvent, especially in the smokeless powder and celluloid industries, is another of these refined products which may be manufactured. Acetone is manufactured from acetate of lime by dry distillation, and is extensively manufactured in this country, although the methods used in many plants are, as in the other industries, inefficient and uneconomical. Process, however, have been devised whereby a product of greater purity may be as readily produced, and offers great opportunities in this industry.

The utilization of the wood tar is a subject that is but little considered, and the majority of plants burn their tar under the retorts. This tar, however, usually contains enough creosote to make it profitable to work. This creosote is a product of indefinite composition, which is much used for producing rosolate of soda, from which sulphuric acid precipitates rosolic acid, a color much used in yellow dyeing or calico printing, and it is also an excellent germicide and disinfectant. It has been profitably extracted from wood tar and wood distilleries in Canada, where it paid to burn coal in the place of tar, and it is evident that in the United States, especially in Pennsylvania, where coal is so cheap, there should be still greater opportunities for working this material and burning coal instead. Plants for refining this tar so as to obtain this creosote, consisting of stills and columns of appropriate design, can be furnished in this country.

From this brief and incomplete outline of the industry in the United States and its possibilities, it is evident that the opportunities open to wood distillers are not as poor as they may have appeared for the last two years, but that rather a large field for profitable investment is open to progressive manufacturers.

Chemistry Exhibits of the American Exposition in Berlin, 1910.

The committee in charge of the American Exposition, to be held in Berlin from July to October, has taken note of the vast importance the chemical and allied industries bear to the trade relations between Germany, commercial Europe, and the United States, and has set aside for the use of these industries some thousand square feet of space for their exhibits.

Mr. E. H. Johnson, a prominent printing-ink manufacturer of Philadelphia; Mr. R. H. Lersner, of the firm of C. M. Childs & Company, New York; Mr. I. F. Stone, president of the Chemists' Club, of New York, and Mr. Eugene A. Widmann, secretary of the H. A. Metz Company, New York, have been asked to serve as a committee on the chemical exhibits and have accepted this invitation. Mr. Widmann, chairman of this committee, can be reached at 30 Church Street, New York City.

There is no need of drawing attention to the immense exchange of chemical and allied products between the United States, Germany, and European as well as Asiatic countries. Berlin, being the commercial center of Continental Europe, offers a very fertile field not only for the manufacturer of the general line of chemicals, but for the maker of specialties. Buyers from Russia, Norway, Sweden, Italy, Turkey and other countries, and even Asiatic countries, as China and Persia, gather in this city to obtain the larger percentage of their wants.

To the American chemical industry a chance is thus given to find new markets by exhibiting the products of their factories at the coming Berlin exposition. By making their wares known in Berlin, their name and brand reach those numerous other countries.

The exposition has the endorsement of both the American and German Governments. Mr. J. Pierpont Morgan is honorary president; Mr. David R. Francis, who was president of the Louisiana Purchase Exposition held at St. Louis, is first honorary vice-president; Mr. John Wanamaker is second vice-president; Dr. George F. Kunz is president of the advisory committee, and Mr. Herman A. Metz is chairman of the executive committee. In Germany, Prince Henry of Prussia is to act as president of the reception committee.

While going to press we are informed that the exhibition will probably be postponed to the year 1911.

A Double-Zone Gas Producer for Bituminous Coal.

At a meeting of the Mechanical Section of the Engineers' Society of Western Pennsylvania, held in Pittsburgh on Dec. 7, 1909, Mr. E. F. Bulmahn, engineer of the gas-producer department of the Westinghouse Machine Company, read a paper on their bituminous power gas producer.

There are two classes of gas producers: pressure producers and suction producers. The question of the relative economy of these two classes of producers is more a matter of design than of theory. However, considering the economy of the two as equal, the sanitary conditions attending the operation of the suction producer is enough argument in its favor.

The two foregoing classes of producers are again divided into three distinct types, viz.: updraft, downdraft and double-zone producers.

The advantages of the updraft producer (fuel charged at top, air introduced at bottom, gas taken off at top above the fuel bed) are the production of a gas of high heat value and complete consumption of the fixed carbon in the fuel. Its disadvantage is that a large percentage of the volatile matter in the fuel is carried off in the gas as tar.

This disadvantage is overcome in the downdraft producer (air introduced at the top of fuel bed and gas taken off at the bottom). By passing the volatile matter, given off during the coking of the coal, through incandescent carbon the tarry vapors are fixed into permanent gases, CH_4, H_2, and CO, producing a fixed gas free from tar. In this type of producer, however, it is impossible to completely consume the fixed carbon. The breaking up of the tarry vapors and the reduction of CO_2 to CO requires from 3 ft. to 4 ft. of incandescent fuel. The oxygen introduced on top of the fuel bed after it has traveled about 2 ft. below the top has combined with carbon to form CO_2 so

that the only carbon recovered in the fixing zone is that taken up by the CO_2 gases in reducting to CO. This results in a high carbon loss in the ash.

The **double-zone** producer is designed to embody the good features of both the updraft and downdraft producers, overcoming the production of tar and completely consuming the fixed carbon. This is accomplished by having two distinct fuel beds, one operated as a downdraft producer, to break up the volatile matter, and the other operated as an updraft producer to consume the fixed carbon, the resultant gas from each zone being taken off at the center of the producer.

The double-zone producer built by the Westinghouse Machine Company was the special subject of Mr. Bulmahn's paper. It is illustrated in Fig. 1. It consists of a top and bottom shell, interposed between which is a circular iron vaporizer, which

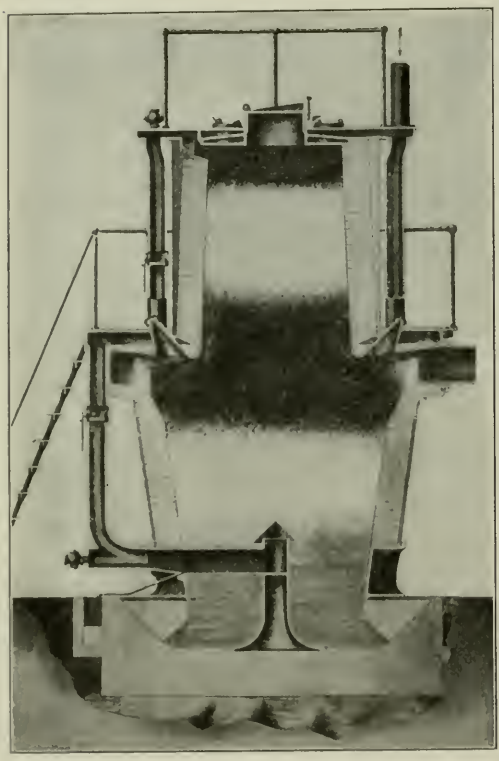


FIG. 1.—BITUMINOUS GAS PRODUCER.

furnishes steam for the producer and also serves to form the gas offtake. Air for combustion is drawn through the vaporizer, being saturated with water vapor, and is led through two pipes to the top and bottom fuel beds, the amount going to each one being regulated by two graduated cocks placed in the pipe line.

The producer is of the water-bottom design, permitting the removal of ash during continuous operation. The top of the producer is provided with a large coal-charging door, which when opened allows the operator to examine the entire top of the fuel bed. Poking is provided for by a series of poke holes through the vaporizer to take care of the bottom fuel bed, the top being poked through holes located on top and through the charging hole.

In the operation of the producer, it is filled completely with incandescent carbon. Coal is charged on top of the fuel bed and air admitted for its partial combustion. The volatile matter given off passes downward through the incandescent carbon, where it is broken up into fixed gases. The coke formed passes down through the producer, being consumed in its travel through the bottom shell, until finally reduced to ash. The gas formed in the bottom fuel bed passes upward, being taken off at the vaporizer with the gas from the top fuel bed.

From the producer the gas passes through a vertical baffled pipe, provided with a water supply for the removal of heavy impurities, to the washer (Fig. 2). The lower end of the pipe is dipped into a water seal and carries away the waste water.

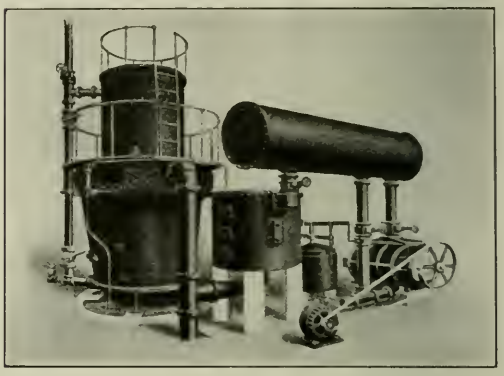


FIG. 2.—GAS PRODUCER PLANT.

The washer is a cast-iron shell, containing a series of three perforated diaphragms, immersed in water. The gas passing through the small perforations deposits its solid impurities, consisting of light ash and lampblack, and is cooled by the incoming water supply. This type of gas washer and the absence of scrubbing material makes possible the continuous cleaning of the gas and displaces the mechanically driven high-speed cleaning apparatus usually employed.

From the washer the gas passes into a cylindrical shell which is used as a mixing tank. The exhauster draws the gas from the mixing tank, maintaining a suction on the producer, and delivers the gas under pressure. When a fuel containing volatile matter is charged on an incandescent fuel bed, the gas given off during the first five minutes is highly enriched, due to the hydrocarbons given off from the volatile matter. The usual method of overcoming this variation in the gas is to provide sufficient holder capacity to equalize the quality. In the Westinghouse producer the exhauster delivers an excess quantity of gas, the excess passing into the mixing tank, where it is mixed with the lower-heat-value gas. The amount is controlled by a gasometer regulator and valve, which also serves to maintain a constant pressure in the gas outlet pipe. This gives a continuous circulation of gas through the mixer and insures an even quality of gas.

The adjoining table gives the results of a 51 weeks' continuous test of a 175-hp double-zone producer and a 15 by 14.3-cylinder gas engine. The plant was operated both 24 and 10 hours a day, the tests covering periods of from 10 to 30 days. Seven different fuels were tested as shown in the table, which gives the composition of the fuels and the economy of operation of producer and gas engine when operated with these fuels.

These tests demonstrate the adaptability of this type of producer to handle fuels, covering a wide range of heat values, and particularly the low-grade fuels. The capacity of the producer was not materially affected by any of the different fuels, except peat, with which fuel the capacity was considerably reduced.

The gas made from all the fuels was reasonably free from impurities, the solid matter averaging not more than .025 grains per cu. ft. of gas. As all the tests were made with the same engine, it afforded an excellent opportunity to note the effect of the gas on the engine. After a run of over 1000 hours, the inlet and mixing valves were removed from the engine for inspection. A slight coating of lampblack was found, but from all appearances it should have operated even a longer period without requiring cleaning.

in the manufacture of steel and glass, the producers being of the updraft type and the gas is used hot and uncleaned. There are a great many applications, however, where the gas must be distributed through small pipes, throughout a plant, and consumed in small burners, and a cool, clean gas is required. This gas may be applied to small tempering furnaces, annealing ovens, enameling kilns, hardening, blueing and baking ovens, small melting furnaces, etc. It makes a particularly inviting proposition where power is required and a portion of the gas

Test No.	Kind of fuel.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur	B. T. U. per lb. as fired.	Average producer efficiency.	Combined producer and engine efficiency.	Pounds coal per B. H. P.-hr. Gas metered and purged out of stack.
1.	South American lignite.....	20.32	34.44	30.58	14.66	1.05	8035	76.0%	....	....
2.	Northern Colorado lignite.....	16.63	33.78	42.22	7.37	....	9589	78.8	16.7%	1.59%
3.	Pittsburgh run of mine.....	2.03	34.98	56.22	6.77	1.29	13,305	76.5	16.8	1.14
4.	Pittsburgh slack.....	2.27	31.86	53.91	11.96	1.76	12,689	76.0*	14.5	1.37†
5.	New England raw peat.....	38.10	40.54	17.86	3.50	0.19	6417	....	....	....
6.	Texas brown lignite.....	23.83	38.32	29.22	8.63	0.57	8007	76.7	17.5	1.82
7.	Pocahontas run of mine.....	1.39	16.01	74.28	8.32	0.69	13,983	80.0	16.5	1.10

* Net efficiency.
† Producer operated 10 hours a day. Fuel consumption and combined efficiency includes stand-by coal 14 hours per day.

The gas made on all the tests was of a good quality and reasonably constant in heat value. The average thermal value on bituminous coals was about 115 B.T.U., being a little higher on lignite gas. The heat value of the gas is rather low, compared with gas from an updraft producer. The reduction, however, is due to the lower H_2 and CH_4, content, a lower H_2 content being desirable for gas engine operation allowing higher compression and advanced ignition setting.

The labor required to operate the producer is comparatively small, it requiring about twenty minutes per hour of one man's time to perform all the labor on a 175-hp unit, using a Pittsburgh coking coal.

The standby loss of the producer when operating 55 hours may be used as fuel, as the labor on the producer in most cases would be the same.

Mr. Bulmahn gave in his paper diagrams showing the results of tests of a producer-gas-fired crucible furnace for melting copper billets. In one of these tests the gas and air were preheated, by utilizing the heat from the waste gases from the furnace. In the other test the gas and air were used at practically atmospheric temperature. A comparison of both tests shows the effect of preheating, in an appreciable increased economy in gas consumption, in higher furnace temperatures attained, and in the decreased amount of time required to take off a heat.

The paper elicited a very extended discussion, which is

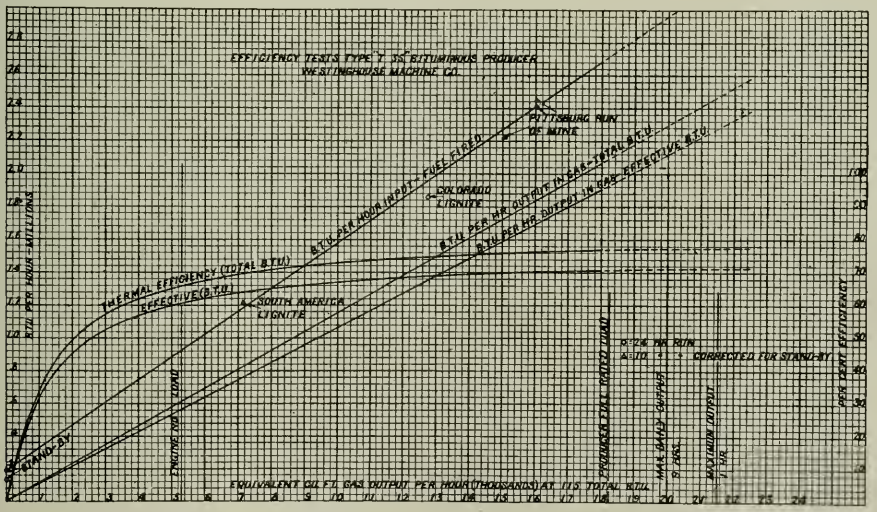

EFFICIENCY TESTS TYPE 'J' 35 BITUMINOUS PRODUCER
WESTINGHOUSE MACHINE CO.
PITTSBURG RUN OF MINE
COLORADO LIGNITE
S. AMERICA LIGNITE
THERMAL EFFICIENCY (TOTAL B.T.U.)
THERMAL EFFICIENCY (EFFECTIVE B.T.U.)
B.T.U. PER LB. COAL AS FIRED
B.T.U. PER LB. OUTPUT IN GAS - TOTAL B.T.U.
B.T.U. PER LB. OUTPUT IN GAS - EFFECTIVE B.T.U.
B.T.U. PER CU. FT. GAS
B.T.U. PER H.P. HOUR
EQUIVALENT CU. FT. GAS OUTPUT PER HOUR (THOUSANDS) AT 115 TOTAL B.T.U.
PRODUCER FULL RATED LOAD
PRODUCER NO LOAD
STAND-BY
CORRECTED FOR STAND-BY
PER CENT EFFICIENCY

FIG. 3.—EFFICIENCY CURVE OF 175-HP BITUMINOUS GAS PRODUCER.

per week represents about 0.9 per cent of the fuel when used when operating at full load.

Fig. 3 shows the producer efficiency curve, plotted from the above mentioned tests. It is particularly interesting in that it conforms to so wide a range of fuels and on account of the slight differences in the producer efficiency between one-half and full load.

The industrial applications of producer gas offer a wide field for gas producers. They are now quite extensively used

printed in full in the *Proceedings* of the Engineers' Society of Western Pennsylvania for January. In reply to several questions, Mr. Bulmahn explained that in the operation of the producer the idea is that in the top half of the producer all that is expected to be done is to coke the fuel and drive off the volatiles, the bottom working on carbon and ash only, so that the bottom fuel bed is operated as a simple updraft producer.

Finally, Mr. Bulmahn stated that with a double-zone pro-

ducer there is less trouble from clinker than in a single-flow producer, for the reason that having two combustion zones, with practically the same area, it gives a slower rate of combustion and a slower velocity through the fuel bed. This results in lower temperatures, preventing the formation of clinkers. Some clinkers are formed when starting the producer with a cold vaporizer. These usually form about the tuyere and are easily removed from the poke holes, of which there are about eight around the circumference of the vaporizer. On intermittent operation it usually takes about 15 minutes after starting the producer before the average amount of steam is generated.

In comparing an anthracite suction producer with this double-zone bituminous producer, Mr. Bulmahn said that in the latter there are some impurities in the form of lampblack and probably the carbon content in the ash is a little higher than with an anthracite producer, which causes a slight additional loss in the efficiency of the producer. The average efficiency of an anthracite suction producer tested by him was about 80 per cent, as against 77 per cent for the bituminous. As to the practicability of the two, while the bituminous producer requires a little more labor than the anthracite, he thinks that it is to-day more reliable than the anthracite for this reason. With a fuel containing volatile matter and where one does not depend entirely on the working reaction, that is the combination of carbon and oxygen, a little irregularity in the fuel bed will not be felt as much in the bituminous as in the anthracite producer.

As a basis of comparison between natural gas, fuel oil, and fuel gas (producer when) applied to smelting or heating furnaces, the following figures were given by Mr. J. K. Lyons.

Calorific values are assumed as follows:

	British Thermal Units.
Natural gas, per cu. ft	1,000
Fuel oil, per gallon	125,000
Fuel gas, per cu. ft (producer)	145

The following quantities are required for one million British thermal units:

Natural gas	1,000 cu. ft.
Fuel oil	8 gallons
Fuel gas (producer)	6,900 cu. ft.

The following table gives the equivalent costs per thousand cubic feet for natural gas and per gallon for crude oil, when compared with producer gas generated from bituminous coal with recovery of sulphate of ammonia as a by-product.

It is assumed that the plant will gasify not less than 48 tons of coal per 24 hours.

Cost of Coal Per ton.	Equivalent price for natural gas per 1000 cu. ft.	Equivalent price for fuel oil per gallon.
$1.25	2.208 cts.	0.276 cts.
1.50	3.588 "	0.4485 "
2.00	6.348 "	0.7935 "
2.50	9.108 "	1.1385 "
3.00	11.868 "	1.4835 "

The above prices cover the cost of the coal and operating costs, but interest charges and depreciation and the auxiliaries are not included. If the fuel gas is used for operating internal combustion engines, the heat in the exhaust gases from the engines would be utilized in waste heat boilers, which would afford ample prevision for all auxiliaries and without any addition to the operating costs.

An Efficiency Test of a Direct-Heat Dryer.

In connection with the various articles on dryer design and operations which have been published in recent numbers of this journal and which indicate that a high degree of perfection of design has now been reached on sound scientific principles, we are pleased to call attention to a short, but interesting article of Mr. W. B. Ruggles, president of the Ruggles-Coles Engineering Company, of New York, in the February issue of *Cassier's Magazine*.

Mr. Ruggles points out that while the determination of the total efficiency of the dryer using direct heat is a simple matter, it is not so easy to discover where the losses occur. There are four ways in which losses may occur—in unconsumed carbon, due to imperfect combustion; in heat carried out by the exhaust gases; in heat carried away by dried material, and in radiation from the furnace and shell.

Imperfect Combustion.—The loss due to imperfect combustion is generally ignored or rather included in the loss by radiation. However, recent tests have shown it to be a considerable factor, unless a large combustion chamber is supplied, in which space and time are given for the perfect combination of oxygen with the carbon monoxide. When the dryer is so arranged that the flames impinge directly upon a cascade of cold and wet material they are quenched suddenly and a considerable amount of carbon passes off unconsumed. This, in certain types of machines, may amount to 7 per cent or 8 per cent of the value of the fuel.

Heat in Exhaust Gases.—The heat carried away by the exhaust gases is generally the largest item of the losses and sometimes amounts to 30 per cent or more, but, as Mr. Ruggles points out, this is chiefly the case in dryers in which natural draught is used, because in this case the temperature of the stack gases is seldom below 400° Fahr., and often 600° or more. He, therefore, strongly recommends to use an exhaust fan and draw the gases against the flow of drying material, so that just before leaving the dryer it has been in contact with the cold and wet material. Undoubtedly some condensation occurs of the steam being carried out, but there is no ultimate loss, as the heat given up in condensation is absorbed by the material and aids in drying later in the process. When the current of the gases is reversed and flows in the same direction as the material, a temperature of at least 212° Fahr. must be had in the exhaust, so that no condensation may occur and moisture deposited upon the finished material.

Heat in Dried Material.—It is, of course, a matter of definition how much of the heat which is carried off by the dried material should be considered as a loss. In general, a temperature of 212° of the dried material leaving the dryer is necessary to the rapid drying of the material. But, as Mr. Ruggles points out, all heat of the material above 212° is unnecessary in most operations, and, therefore, should be charged as a loss. This is generally not great, and is, in most cases, due to imperfect operation and not to the design of the machine.

Radiation.—The radiation loss depends greatly on the design of the dryer. "In some, the shell is cool enough to place the hand on without discomfort; in others, it is not uncommon to see the shell red hot for 10 ft. or 12 ft. from the furnace. This, of course, is in a single-shell type, in which the flames impinge directly upon the shell. The loss by radiation in such a dryer is very great. In a bricked-in machine the temperature is not so high; but, as more surface is exposed, the total loss is almost as great." In general, the radiation loss is determined by difference of the total loss and of the other losses.

The following figures of a recent dryer test given by Mr Ruggles are very instructive as the test has evidently been carried out with great care. The coal used for fuel was dried and tested calorifically. The flue gases were measured by an aerometer and analyzed for CO, and the material tested for moisture and specific heat. The figures are given in round numbers.

It will be seen that the very high efficiency of 85.8 per cent of the dryer under test was found.

Amount of coal burned per hour	400 lb.
Amount of moisture in coal	10 per cent
Calorific value of dry coal	13,000 B.t.u.
Material Dried—Bank Sand.	
Amount of wet material per hour	45,000 lb.
Amount of dried material per hour	42,750 lb.
Specific heat of sand	.21
Moisture in sand	5 per cent
Temperature of air, sand and coal	60° F.
Temperature of dry sand	224° F.
Total moisture in coal	40 lb.
Total heat per pound of water 60° to steam	1,122 B.t.u.
Heat required to evaporate water in coal $40 \times 1122 =$	44,880 B.t.u.
Total heat in dry coal $360 \times 13000 =$	4,680,000 B.t.u.
Total heat for drying $4680000 - 44880 =$	4,635,120 B.t.u.
Specific heat required to heat sand 60° to 212°, $42750 \times .21 \times 152 =$	1,364,580 B.t.u.
Total water in sand	2,250 lb.
Total heat required to raise temperature of moisture and evaporate same $2250 \times 1122 =$	2,524,500 B.t.u.
Theoretical heat required	3,889,080 B.t.u.
Efficiency of dryer $\frac{3889080}{4635120} =$	85.8 per cent
Total losses in dryer	14.2 per cent
Excess temperature of sand	12°
Loss by same $42750 \times .21 \times 12 =$	117,730 B.t.u.
Percentage $\frac{117730}{4635120} =$	2.5 per cent
Quantity of air exhausted @ 100° per hour	450,000 cu. ft.
Weight of air exhausted @ 100° per hour	31,905 lb.
Loss by excess temperature $100° - 60° =$	40°
Specific heat of air	0.2374
Loss in exhaust air $31905 \times 40 \times .2374 =$	302,960 B.t.u.
Percentage $\frac{302960}{4635120} =$	6.5 per cent
Amount of CO in exhaust gases per hour	3.4 lb.
Value of CO as fuel, per pound	10,100 B.t.u.
Total loss $10100 \times 3.4 =$	34,340 B.t.u.
Percentage $\frac{34340}{4635120} =$	.7 per cent
Recapitulation.	
Loss by excess temperature of sand	2.5 per cent
Loss by heat of exhaust gases	6.5 per cent
Loss by imperfect combustion	.7 per cent
Loss by radiation and other causes	4.5 per cent
Dryer efficiency	85.8 per cent
	100.0 per cent

This test thus shows the surprisingly high efficiency of 85.8 per cent, which is explained by the fact that the dryer under test had almost perfect combustion, very low exhaust temperature, and that the average temperature of furnace and shell was not over 150° Fahr.

Moler Bricks

So much interest has been attracted by the article on moler bricks, published in our November issue, 1909, that we add here some further details for which we are obliged to the manufacturers, Frederiksholms Tegl- & Kalkvaerker, Copenhagen, Denmark. They are making the bricks for partitions, ceilings, etc., in sizes up to 240 mm by 240 mm by 150 mm size (9½ in. by 9½ in. by 6 in.). Particulars of the composition of the "molar" earth are given in the following chemical analysis, viz.:

Silica	67.8 per cent
Alumina	6.7 " "
Iron oxide	5.5 " "
Lime	1.2 " "
Magnesia	0.9 " "
Carbonic acid	Trace
Sulphuric acid	0.6 per cent
Loss by heating	7.4 " "
Hygroscopic water	8.3 " "
Alkalies (by difference)	1.6 " "
	100.0 per cent

The weight of a moler brick of Danish normal size, 230 mm x 110 mm x 55 mm, is about 1.30 kg, while an ordinary brick weighs about 2.50 kg.

The specific gravity of a moler brick is about unity. Pressure tests of a moler brick made on June 11, 1909, showed an average strength of 156 kg per square centimeter.

Moler brick can be supplied in all European sizes, both solid and perforated. Special moler bricks are made for ceilings, arches, partitions, etc. On account of their extreme lightness, combined with unusual strength, the moler brick is recommended for partitions, ceilings, arches, etc., and, further, owing to their insulating properties, it is specially suitable for bricking steam boilers (but not to take the place of firebrick), for inner walls of hot rooms and for cooling stores, etc. They cannot be used as outside bricks on account of their porosity. Moler bricks possess special deafening qualities and are greatly to be preferred to plaster for partitions.

The following special tests of moler bricks were made by the Royal Danish Bureau for Testing Materials:

In a fire test made at one of the Copenhagen municipal gas works, it was proved that moler bricks could withstand a temperature of 1100° C. without the stability of the wall being affected.

An insulation test was made in the following manner: Three testing chambers were built close to each other above the furnaces. One of moler bricks with lime mortar, one of common bricks with lime mortar, and one of Glenboig firebrick in fireproof mortar, silica brick (fireproof) in fireproof mortar, and Kulmit brick (fireproof) in fireproof mortar. Each of the testing chambers was 18 in. square.

The test was continued for three weeks. The temperature at the surface of the furnace was 164° C. to 170° C.; in the upmost layer of moler bricks it was 63° C.; in the upmost layers of the other insulating refractories it ranged from 73° C. to 74° C., showing a difference in favor of moler bricks during the whole period of the test of at least 10° C.

Precious Metals in Colorado.

The precious metal production in Colorado for the year of 1909 was valued at $42,575,878. It is distributed as follows:

Metal and amount.	Value.
Gold, 1,303,713 oz. at $20.67	$29,949,803
Silver, 11,614,500 oz. at 51.5 cents	5,981,461
Lead, 34,023 tons at $85.60	2,912,289
Copper, 12,829,561 lb. at 13 cents	1,667,843
Spelter, 24,262 tons at $107.20	2,600,920
Chemicals from Colorado ores	1,701,562
Tungsten, 1100 tons at $360	396,000
Vanadium, 120,000 lb. at $2.50	300,000
Uranium, Gilpin County	35,000
Manganiferous iron-silver ores	15,000
Pyritic iron-silver ores	16,000
Total	$42,575,878

The total increase over the 1908 products is $9,292,867, distributed as follows: Gold, $4,254,227; silver, $1,006,033; lead, $426,618; spelter, $806,317, and copper, $284,110.

The Vinson Walsh research fund has stimulated the search for pitchblende ores throughout the State of Colorado. The output of radioactive ores has doubled. Ores of vanadium and uranium, including carnotite, are found in the Gilpin, San Miguel and Ouray districts. The hunt is being directed by the Colorado School of Mines at Golden.

Boulder County leads the United States in the production of tungsten with a total of $396,000 against a total of $135,000 in 1908. The 1907 production was $550,000. The 1908 market was against production. Since the first of last July, however, an improvement in quotations started output at a rate that may make the 1910 product $1,000,000 in view of the $7 per unit price. The average price last year was $6 a unit for the 60 per cent fine material, or $360 a ton. The Primos mill of the Primos Mining and Milling Company received $200,000 of the total paid for tungsten. This mill will have a competitor in 1910 buying and shipping concentrates.

Colorado dividends in 1909 show totals as follows: Gold mines and placers, $6,403,853; gold-silver mines, $1,946,500; silver-lead mines, $740,000; gold-lead mines, $75,000; silver mines, $100,000; gold-tungsten mines, $125,000; total, $9,390,353.

The total record of seven smelting companies treating Colo-

rado ores in 1909 was $24,124,953. Some of these treated zinc only, but most of them treated three, and some all five metals —gold, silver, lead, copper and zinc.

The chlorination mills in the State are credited with a production valued at $12,589,874. The only amalgamation mill, the Tom Boy, at Telluride, produced $487,486. The output of the cyanide mills was worth $1,614,072.

The figures given are from the report of the State geologist of Colorado.

Oxy-Acetylene Cutting.

An interesting application of the oxy-oxy-acetylene torch for cutting metal is shown in the adjoining pictures illustrating the use of this process by the American Oxygen Company of Philadelphia for cutting holes thirteen feet in circumference in the steel piling of the new permanent intake well for

FIG. 1.—CUTTING WITH OXY-ACETYLENE TORCH.

the new Chester Water Supply system, for the city of Chester, Pa.

The holes were cut in the bottom of the well after the piling was driven into place, and were for the admission of the water conduits.

The web of the piling is ⅜ in. thick, while the male ends were 2 in. thick by 2 in. wide.

Those familiar with the use of pneumatic tools will readily see that cutting such holes as these by that method would have been a slow, difficult, and expensive operation, particularly as it would have been necessary to cut the male ends with a hack saw. The cutting in this instance was easily accomplished with the standard portable plant of the American Oxygen Company, the operation requiring only minutes, where hours would have been consumed in accomplishing the same results with compressed air tools and saws.

This is only one of the many instances where the oxy-acetylene process has proved itself a valuable factor in construction work. It is stated that the great expense of cutting such passages with air tools and by hand was formerly a very serious argument against installations of this nature. As this difficulty has now been overcome by the oxy-acetylene process, it will be readily seen that the field for steel piling installations will be greatly enlarged.

Swedish Titaniferous Iron Ores.

Concerning the deposits of titaniferous iron ores in Sweden, Consul-General E. D. Winslow, of Stockholm, in a recent report, furnishes the following information: The largest deposit of titaniferous iron ore in the middle of Sweden is Taberg, in the Province of Smaland, seven miles south of the town of Jonkoping. This small mountain is 450 ft. high and consists entirely of this iron ore. The area is figured to be 2,800,000 sq. ft.

The ore is called magnetite olivinite, being at the same time a species of hyperite, holding a comparatively small percentage of iron, which is generally not higher than 32 per cent in the purer sections, and sometimes only reaches 21 per cent. There are besides some smaller veins of rich ore, holding up to 60 per cent of iron, but these veins have only a thickness of about 3 in. The proportion of titanic acid is about 4 to 8 per cent. The presence of phosphorus is 0.055 to 0.10 per cent.

The largest deposits of titaniferous iron ore on the northern parts of Sweden are Ruotivare and Vallatj, situated 10 miles northwest of Kvikkjokk Church, in the Province of Norrbotten. The distance to the nearest harbor on the Norwegian coast is 68 miles over rocky mountains, and the distance to Jokkmokk, in Sweden, is about the same, mostly over navigable lakes and streams. A new addition to the State railway of Sweden, now under construction, will connect Jokkmokk.

The ore at Ruotivare consists of a mixture of titaniferous magnetite, ilmenite, spinelle and olivine. The area has been figured at 3,240,000 sq. ft.

Gold and Silver in 1908.

According to the joint report of the United States Geological Survey and the Bureau of the Mint the production of gold in this country during the calendar year 1908 amounted to 4,574,340 fine ounces, which represents a value of $94,560,000.

FIG. 2.—STEEL PILING.

The figures show an increase of 109,513 fine ounces, or $64,124,300 over the preceding year. The production of silver was 52,440,800 fine ounces and was valued at $28,050,600, this being a shrinkage from the 1907 figures of 4,073,900 fine ounces, or $0,240,900.

In addition the production of home smelters and refineries handling foreign ore, matte, and unrefined bullion, was as follows: Gold, 892,138 fine ounces, or $18,442,100; silver, 65,107,220 fine ounces, or $34,506,800. This supply was de-

rived chiefly from Mexico, Canada, South America, Central America, Korea, and Japan.

In the native production of gold, among the States and Territories, Colorado led with a total value of $22,871,000, while Alaska and California were close behind, credited with over $19,000,000 each.

The year's increase in gold production was thought to be due, in one degree, to the closing of many mines in the base-metal camps. A tendency toward increased production among small operators was noted. Large numbers of miners left the districts in which the output of lead, copper and zinc was curtailed and turned to placer work or took up leases in the gold camps. This depression in the base-metal industries did not otherwise affect the gold production, for the total contribution from the source amounts to but 7 per cent of the normal gold output.

Placer gold made 28 per cent of the whole production, and gold from siliceous ores amounted to 65 per cent. The base-metal ores, in spite of an exceptionally heavy tonnage, gave but 7 per cent. This shows that it is futile to look for a heavily increased production of gold from a stimulated activity in these ores.

Notes.

American Electrochemical Society.—The date of the annual meeting to be held in Pittsburgh has been changed at the request of the local committee. The meeting will not be held in the last week of April, but on May 5, 6, 7. The local committee has already outlined a very attractive program.

New York Section American Electrochemical Society.—A meeting will be held on March 16 in the Chemists Club. Dr. Baekeland will address the section on the Eighth International Congress of Applied Chemistry to be held in this country in 1912. Dr. F. Zimmermann, of Newark, N. J., will present a paper on the Decomposition of Carbon Monoxide at Certain Temperatures in the Presence of Various Catalytic Agents. Dr. R. Stevenson of the College of the City of New York will read a paper on Electrode Reactions.

The South African Institute of Electrical Engineers.—The only society connected with the electrical profession in South Africa—has recently been formed and with the promises of support already extended to it, a membership of over 500 is shortly anticipated. Mr. Fred. Rowland, 9 Transvaal University College Building, Johannesburg, Transvaal, is the secretary.

The Susquehanna Smelting Company has decided to close its electric furnace plant for the manufacture of ferrosilicon in Lockport, N. Y., on account of the continued unsatisfactory condition of the ferrosilicon market and difficulties in regard to obtaining an extension of lease for its present plant, which is owned by the Holly Manufacturing Company, of Buffalo. Ferrosilicon is now manufactured on a large scale in this country by the Electro Metallurgical Company (with works in Kanawha Falls, W. Va., and Niagara Falls, N. Y.). A Canadian company is operating at Welland. Metallic silicon is made by the Carborundum Company.

The Mexican Institute of Mining and Metallurgy.—The regular monthly meeting of this organization was held at the School of Engineers in the City of Mexico. The members of the Institute came by reason of a special invitation from the director of the school, and they made a complete inspection of the principal departments.

Nickel in Canada.—The Mines and Minerals Committee of the Canadian House of Commons has taken up an investigation regarding the nickel resources of the country and the condition of the trade in that metal. It has been thought that the high price at which nickel is held has unduly limited the development of the immense deposits which are available.

Preliminary Mint Report for 1909.—The production of both gold and silver by the mines of the United States increased in the calendar year 1909, according to the preliminary estimates made by A. Platt Andrew, Director of the Mint. These figures show the gold product to have been $99,232,200, against $94,560,000 for 1908; silver, 53,849,000 fine ounces, an increase of 1,408,200 fine ounces. At the average price of silver—$0.52016 a fine ounce for the year—the commercial value of the silver product for 1909 would be $28,010,000.

Large Gold Export.—A recent government report showed that the year 1909 witnessed the outflow of $132,880,000 gold from the United States. This is an unexampled record. In 1904 there was shipped $121,211,000, and in 1895 $104,967,000; no other year on the record approaches 1909. The actual balance of exports over imports last year was $88,793,000; in 1904 it was only $36,408,000, and in 1895 $70,571,000.

Montana Copper in 1909.—In spite of the low prices prevailing for copper the yield of the Montana mines for last year is placed by William Walsh, State mine inspector, at a figure slightly in excess of the preceding year. Mr. Walsh estimates the output of the State for 1909 as follows: Copper, $46,000,000; gold, $4,000,000; silver, $8,500,000; lead, $2,000,000; total, $60,500,000. From every district in the State come reports of renewed activity, and this is especially true of the Butte, Corbin, Helena, Saltese, Jardine, Basin, Zortman, Boulder, Dillon, Clinton and Libby districts.

Mexican Mineral Products.—Following is a résumé of the total values of the nine products for 1909, according to the report of the Minister of Finance to the Mexican Congress: Gold, $45,014,954; silver, $77,109,742; copper, $20,383,000; lead, $6,397,000; zinc, $1,043,000; iron, $1,200,000; other metals, $2,052,158; coal, $4,400,000; mineral oil, $2,800,000; total, $160,399,854.

Petroleum.—The world's production of petroleum in 1908, as given by the *Chemiker Zeitung,* amounted to 38,052,233 tons. In 1907 35,032,235 tons were produced. There was thus a material increase. The production in tons for the two years itemized by countries is given below. The United States is credited with an output about three times as great as that of any other country:

Country.	1908.	1907.
United States	23,942,997	22,149,862
Canada	70,400	105,200
Mexico	464,188	133,855
Peru	134,824	8,732
Russia	8,291,526	8,247,795
Dutch India	1,143,243	1,116,946
Galicia	1,754,022	1,175,974
Roumania	1,147,727	1,129,097
British India	672,938	579,346
Japan	276,124	268,129
Germany	141,900	106,379

India's Mineral Production.—A summary of the mineral production of India for 1908, which appeared in a recent issue of the official *Indian Trade Journal,* states that the total value of the minerals reported in that year was $38,074,254, as against $34,354,400 in 1907. The heavy output of coal contributed the principal share to the increase, but there was also an increase in gold, petroleum, salt, saltpeter, jadestone, graphite and magnesite. On the other hand, there was a reduced output of manganese ore, mica, ruby, tin, iron ore, chromite and diamonds.

European Zinc Combine.—Referring to a report regarding the European zinc trust, Consul H. Albert Johnson, of Liége (Belgium), says that it appears that the various zinc concerns making up the trust are arranged in two principal groups. The first group comprises 18 concerns, principally the zinc works of Upper Silesia and Westphalia, certain Austrian concerns, and several Belgian works. This group is organized as a German company. In Silesia the Heritier George Van Geische Company has remained outside the syndicate, although in many ways closely associated with it. The production of the first group has been limited for 1909 to 255,086 metric tons (280,595 short tons) and for 1910 to 264,232 metric tons (290,655 short tons). The second group comprises the zinc works mainly located in Belgium, notably the Vieille Montagne Company. The production of this group is limited to 174,519 metric tons (191,971 short tons) for 1909 and to 175,919 metric tons (193,511 short tons) for 1910.

Metallurgy in Norway.—By action of the general meeting of the Meraker-Gruben Company, the works of that concern were closed the middle of December, the reason assigned being the poor condition of the ore market. One of the principal owners was the Frankfurter-Metall-Gesellschaft. This company bought the mines 40 years ago and invested 2,500,000 kroner therein. The Erjes Nickelwerk have increased their capitalization to allow an enlargement of the works.

Geological Survey Changes.—Owing to the recent resignation of Professor H. K. Barrows, of the Massachusetts Institute of Technology, as district engineer of the Geological Survey for New England and New York, a rearrangement of district chiefs has been made. Mr. C. C. Covert will be in charge of New York, Mr. Cyrus C. Babb will direct the work in Maine, while Mr. T. N. Norcross will be district engineer for the remaining New England States.

Lin Sheshun has taken formal possession of the standards of weights and measures which the Chinese Government ordered last spring from the International Office of Weights and Measures, in Paris, for the purpose of introducing the metric system in China, in conformity with a law promulgated in 1908.

Aluminium Castings.—The Valley Foundry Company of Canal Dover, Ohio, has recently been organized for the manufacture of aluminium and probably brass castings.

Addition to Foundry.—The Commonwealth Steel Company, at St. Louis, Mo., is completing an addition of its foundry which will increase the capacity about 40 per cent.

Pressure and Vacuum Gauges form the subject of a booklet published by the Schaeffer & Budenberg Manufacturing Company, of New York City. Indicating instruments for measuring pressure, temperature and speed are dealt with in the first part, while the second part is given over to recording gauges of various types.

Centrifuges.—A new catalog just issued by the International Instrument Company describes the centrifuges manufactured by that concern. The machines are electrically operated, being built in combination with motors. Several of the chemical ends enumerated to which the machines can be put are to dry crystals for rapid production of chemically pure salts, to hasten settling of precipitates and thus avoid oxidation, to extract rapidly essences of compounds from drugs, and to refine sugar. The centrifuges vary in sizes, many being very moderate in price. Accessories are listed in the latter part of the booklet.

Slow-Speed Chilian Mills.—In the fourth edition of their catalog, which was recently issued, the Lane Slow-Speed Chilian Mill Company, of Los Angeles, Cal., print a general article on the use and operation of slow-speed mills. Following this come a detailed description of the company's product, a series of letters from actual operators, and a department of tables and other useful information.

Lava.—The material known as "Lava" and familiar to most people on account of its very universal use in gas tips and burners, is the subject of an instructive booklet issued by the American Lava Company, of Chattanooga, Tenn. The substance is not, as might be inferred, of natural volcanic origin, but is the mineral talc $H_2Mg_3Si_4O_{12}$. It is machined while soft in its natural state and through subsequent baking at a temperature of about 2000° Fahr. it attains its familiar hardness.

Gas-Burning Appliances.—An interesting bound volume of bulletins relating to gas-burning appliances has been recently issued by the Improved Appliance Company, of Denver, Colo. Besides listing the burners and ovens manufactured by the concern, it contains as well much useful data, including tables of melting points of various substances, heating values of gases, boiling points, cubic feet of gases to the pound, approximate water capacity and specific heats. There are also notes on the properties and uses of gases.

Coal Briquetting Machinery.—From the Mashek Engineering Company, of 90 West Street, New York City, comes an extremely well-arranged catalog describing plants and machinery, together with a brief discussion of processes and cost of manufacture. In a section dealing with the possibility of briquetting without binders, it is stated that there is no coal in the United States that can be handled in this way and give a commercially successful article. It is also said that the expenses of plants operating without the use of coal-tar pitch are higher than those of plants briquetting the same fuel with the use of the pitch binder.

The Toledo Electric Welder Company moved its offices from Toledo to Cincinnati on Feb. 4 and is now located in a new factory at Knowlton and Langland streets, Northside, Cincinnati, Ohio. The building is fully equipped with the best and most modern tools necessary for the building of their line of welding machines, the entire plant being motor driven. A 75-kw, single-phase, alternating-current generator supplies the power for testing the welding machines up to their full rated capacity. The shops are lighted by a motor-driven direct-current generator and mercury lights are largely in evidence. A night force will be put on at once to take care of the deliveries promised.

Leadite Melting Furnace.—A folder describing a furnace just devised for melting leadite has been issued by the Leadite Company, Land Title Building, Philadelphia, Pa. The furnace has been designed with special regard to safety, and as it uses gasoline for fuel, it saves much time in pipe jointing work when compared with a charcoal-burning furnace. It will melt 50 lb. of leadite or 325 lb. of lead.

Bleachery.—The Cluett, Peabody Company of Troy, N. Y., are planning to erect, at Cohoes, N. Y., a bleachery to work in conjunction with the shirt and collar manufacture.

The Metallic Purification Company is planning to erect a plant at Chattanooga, Tenn., which will cost approximately $1,000,000 and will employ 1,500 men. It is stated that the principal outputs of the plant will be alum and aluminium. The by-products will depend on the chemical analysis of the clays used. The information so far obtainable is meagre.

The American Oxygen Company of Philadelphia, Pa., was wrongly stated in an article in our last issue to be the successors of the American Ferrofix Brazing Company. The American Ferrofix Brazing Company is still in existence and the American Oxygen Company has no connection with it whatever.

Mines Census.—The Geological Survey and the Census Bureau have entered into a joint arrangement by which the Survey is relieved of the collection of statistics of the annual production of minerals for the year 1909. That work is, by the agreement, merged in the Census Bureau's activities under the Thirteenth Census act, which requires a census of mines and quarries for 1909. Consequently, the Survey will not this year send its own schedules directly to the mine and quarry operators. The Census Bureau's general and supplementary schedules relative to this branch of its decennial work include the inquiries in which the Survey is interested and such other data as may be necessary for the latter's reports will be furnished the Survey by the Census Bureau. The schedules will be collected by the Census special agents and enumerators. The co-operative arrangement was in recognition of the necessity of uniformity in the compilation of the statistics, the elimination of duplicate work and inquiries, and the desirability of economy in the collection of the required data.

The Dorr Cyanide Machinery Company has been incorporated with J. V. N. Dorr as president and offices at 728 Equitable Building, Denver, Colo. This was made necessary by the increasing business in Dorr classifiers and Dorr continuous thickeners, and incidentally permits Mr. Dorr to devote more time to his professional work as metallurgical engineer. An interesting application of the Dorr classifiers and Dorr pulp thickeners is noticed in an article in this issue on a silver cyanide plant in Mexico.

Electrical Mining Equipment.—A publication of interest to electrical engineers engaged in mining has recently been

issued by Crompton & Company, Salisbury House, London, E. C. It contains about 50 illustrations of plants and apparatus typical of the methods used in the supply and utilization of electricity in mines, a subject to which, as demonstrated by the extensive list of mines in which Crompton machinery is used, the company has given considerable attention.

Iron and Steel Activity in France.—Great activity is noted in the iron and steel manufacturing districts in France. This is due in a large measure to the placing of orders for 1000 cars for the state railroads and 2300 cars for the Paris-Lyons Railroad.

National Association of Cement Users.—The annual convention of this association has been held at Chicago, Ill., February 21-25, 1910. In addition to the regular committee reports and topical discussions a list of papers of general interest will be presented.

Carnegie Relief Fund.—The relief fund created by Andrew Carnegie for employees of the Carnegie Steel Company disbursed $200,545 in accident and death benefits and pension allowances in 1909, which is about $10,000 less than the amount paid out the previous year. This makes a total of $1,540,086 disbursed since the relief fund was instituted in 1902.

Iron Production and Consumption.—"The most distinctive feature in the iron trade of the world in the past year," the London *Statist* declares, "has been the marked contrast between the activity in the United States and the lethargy in this country. In truth, the consuming capacity of the American furnaces beyond the American sources of supply of the mineral presents one of the problems of the trade—or presents it again in a vivid form, for it is not a new one—namely, the future supply of the world's iron."

Cement Companies Merged.—By a merger filed with the New Jersey Secretary of State, on Jan. 22, the Cement Manufacturing Company, with a capital stock of $2,000,000, was absorbed by the Alpha Portland Cement Company, which has works in Warren County. The Alpha Company had a capital stock of $6,000,000, of which $2,000,000 was outstanding. The capital stock of the consolidated corporation is $10,000,000, divided into $8,000,000 common and $2,000,000 preferred, the latter bearing 7 per cent cumulative dividends. The officers of the new concern are: President, Arnold F. Gerstell, Easton, Pa.; vice-president, John M. Lockhart, Pittsburgh, Pa., and secretary-treasurer, Gabriel S. Brown, Easton, Pa.

New Engine Ordered.—The Sapulpa (Oklahoma) Steel & Iron Mills have recently placed an order with the Allis-Chalmers company for a 28-in. x 48-in. non-condensing, heavy-duty rolling-mill engine.

New Mining Equipment.—The mining engineering laboratories at the Massachusetts Institute of Technology have recently installed an Allis-Chalmers feeder and a Richards pulsator jig for experiments on ore concentration. This latter apparatus was invented by Prof. Robert H. Richards last year and it has proved of great value. In the mining department the following changes in assistants were made for the current scholastic year: F. Jaeger succeeds L. A. Dickinson; T. G. Chapman succeeds L. D. W. Bender; H. R. Batcheller succeeds C. A. Gibbons, and E. T. Almy has been appointed.

Water Turbines.—The Dayton-Globe Iron Works Company, of Dayton, Ohio, have recently issued catalogs on their "New American Turbine" for the development of water power. The turbines are of the reaction type, vertical or horizontal, and according to requirements are furnished with either balanced flutter, wicket or cylinder gates. Two classes of wheels are made: those for low head from three to fifty feet, diameters from ten to sixty inches, and outputs up to 2363 hp; and those for a high head of fifty to two hundred feet. The largest turbine listed operates with a two hundred foot head and a seventy-two inch wheel and delivers 6784 hp at a speed of 179 to 238 r.p.m. and a discharge of 22,380 cu. ft. per minute. Head gate hoists and gearing work for the wheels, also beating and washing engines and wood pulp grinders are included in the output of this company. The increasing hydroelectric installations among the mining and metallurgical plants of the West present a large field for water turbine equipment.

Salt Industry in Italy.—In Italy the salt industry is entirely in the hands of the government, it being a monopoly utilized as a means of taxation for raising revenue. The importation or production of salt by private individuals is prohibited, and it is sold to the public only in special shops where postage stamps and tobacco are offered for sale. All the salt produced in Italy comes from sea water, evaporated at the works on the islands of Sardinia and Sicily, and at Volterra in the Province of Pisa. The government both imports and exports salt. In 1907 the imports were 51,604 and the exports 99,191 tons. In 1908 the imports were 42,610 and the exports 85,489 tons. Salt was exported to the United States to the amount of 15,590 tons in 1907 and 7,871 tons in 1908.

Glass Manufactory to Rebuild.—The Muncie (Ind.) Glass Manufacturing Company has decided to rebuild its works, recently destroyed by fire, at an estimated cost of $70,000.

Transite Asbestos Wood.—A booklet describing one of their products, "Transite Asbestos Wood," has been issued by the H. W. Johns-Manville Company, of New York City. This material is mineral in character, being made from asbestos fiber and binding cements. It has the same physical characteristics as ordinary wood, except that it is harder and takes a higher polish. Among the advantages urged for this product are its adaptability to manipulation by means of wood-working tools, its fiber strength under transverse loads of 3500 lb., and its absolutely fireproof qualities, it having withstood the most severe tests by fire without being warped, distorted, or weakened in any way. The manufacturers specially recommend its use for flooring, sheathing, partitions, fire doors, fireproof cabinets and gages. A special form of the product, called "Ebony Asbestos Wood," is made to combine high electrical insulating capacity with its fireproof properties. It is somewhat similar in structure to the regular product, but it is saturated with a special insulating compound. When finished it resembles ebony. For example, it may be used for switchboards, panel boards, switch and fuse bases, insulating barriers, duct linings, car-floor linings, electric heater insulation, and rheostat tops.

Personal.

Mr. **F. A. S. Andrew** has joined the firm of E. A. Herbert & Company, Ltd., of London, England, as a director. He will supervise particularly the departments dealing with spiegeleisen, ferro-manganese, ferrosilicon, silico-speigel and other alloys, together with manganese, manganiferous iron and other ores. For the past 20 years Mr. Andrew has been associated with the Pyle & Blaina Works, Ltd.

Mr. **W. G. Bringham** has been appointed manager of the new tin plate plant of the Phillips Sheet & Tin Plate Company, at Weirton, W. Va. He was formerly manager of the American Sheet & Tin Plate Company.

Mr. **Fred H. Daniels** has been decorated by King Gustav of Sweden with the Cross of Knighthood of the Northern Star in recognition for courtesies extended to Swedish engineers in this country. Mr. Daniels is chief engineer and a director of the American Steel & Wire Company.

Mr. **A. E. Drucker** has been appointed consulting metallurgist of the Oriental Consolidated Mining Company in Korea.

Mr. **William E. Ellis** has left for Mexico and has again taken charge of the Guebeshe Mill at Ocotlan, Oaxaca, Mexico.

Mr. **W. Spencer Hutchinson** is arranging for the installation by the Vulture Mines Company of a cyanide plant designed by Mr. S. V. N. Dorr.

Mr. **J. P. Jennings**, district superintendent of the Pennsylvania Coal Company, has been chosen president of the Scranton Mining Institute for the ensuing year.

Mr. **Robert Kann**, consulting chemist to the Raritan Copper Works, Perth Amboy, N. J., has opened at 24 Cliff Street, New York City, a commercial laboratory, making a specialty of metallurgical analysis and research.

Mr. **Julian Kennedy** has been appointed engineer to supervise the erection at Buffalo of a new blast-furnace plant for the Rogers-Brown Iron Company.

Dr. **C. Offerhaus**, known to many of our readers through his connection during the past few years with several metallurgical companies of this country and with Columbia University, is now connected as metallurgical engineer with Messrs. Paul Schmidt & Desgraz, Prinzenstrasse 1a, Hannover, Germany.

Mr. **Leopold Salazer** has been appointed secretary of the Mexican Institute of Mining and Metallurgy to fill the vacancy caused by the resignation of Mr. Kirby Thomas.

Mr. **T. I. Stephenson** has been elected president of the Knoxville (Tenn.) Iron Company, succeeding Mr. W. B. Chamberlain, resigned.

Mr. **Thomas R. Stevens** has accepted a position with the Caucausus Copper Company, of Batum, Russia.

Mr. **R. K. Stockwell**, who has recently been with the Steptoe Valley Smelting & Refining Company, in the position of smelter engineer, has resigned that position and is now construction engineer for the United States Smelting, Refining & Mining Company. At the present time he is stationed at Eureka, Utah.

Mr. **Albert L. Waters** has completed successfully the installation of the new pyritic smelter at Mollen, Colo., six miles north of Denver.

M. **André Weill**, of Paris, has been inspecting a number of water-power sites in the vicinity of Ottawa, Canada, in the interests of the Creuzot Steel Company. This is thought to have to do with the contemplated erection of an electric iron-ore reduction and steel-refining plant near Ottawa.

Obituary.

Mr. **James W. Fuller**, president of the Lehigh Car, Wheel & Axle Works, died in Catasauqua, Pa., on Jan. 15.

Mr. **Fayette Brown**, president of the Brown Hoisting Machinery Company, died on January 20, at his home in Cleveland, Ohio. Mr. Brown was born at Bloomfield, Ohio, in 1823. In 1880, with his two sons, Harvey H. and Alexander E. Brown, he organized the Brown Hoisting and Conveying Company, which has since been reorganized, becoming the Brown Hoisting Machinery Company. At the time of his death Mr. Brown was chairman of the Stewart Iron Company and president of the Union Steel Scale Company and the International Chemical Company.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matters and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

Electric Furnaces (*Continued*).

No. 652,640, June 26, 1900, Henry N. Potter, Goettingen, Germany.

Resistance type, for forming tubular Nernst-lamp glowers, etc. The chamber and resistor is a tube composed of a mixture of dry electrolytes, e.g., magnesia and yttria, or zirconia and yttria, ground and mixed with paste, tragacanth or starch, and water, to form a dough. Around this tube is a jacket consisting of one of the constituents of the tube, e.g., magnesia or yttria, or some other material inert or non-injurious to the tube, e.g., thoria. The jacket is baked on the tube in a porcelain kiln. Then a heating-conductor is wound about the furnace, or about a tube of insulating material, talcite, surrounding the furnace. Platinum terminal wires are attached to the ends of the inner resistor tube. The composite tube is heated by gas to such temperature that the inner tube carries current. Upon the first electrical heating both tube and jacket crack. The cracks are filled with a paste of similar material. Several such patchings may be necessary. The jacket serves as a non-fluxing support for the cracked inner tube, having small electrical conductivity. The wire winding serves in use to bring the tube resistor to the temperature requisite to cause it to act as a solid electrolyte and heating resistor.

654,463, July 24, 1900, Henri Leleux, of Paris, France.

"Resistance" type. A furnace especially for the production of calcium carbide, comprising a pot mounted on a carriage or suspended from traveling pulleys, constituting the lower electrode, and an upper electrode adjustably depending from a fixed support. The pot has a flat hearth of carbon blocks and metal sides and casing. The hearth is partially subdivided by a horizontal insulating layer of non-conducting concrete, to concentrate the resistance-heating action of the electric current, passing upward from the metal bed-plate beneath, to the front portion of the hearth, near the tap hole. The metal casing automatically makes contact with an electric main when the pot is moved into place. The pot is surrounded on three sides by walls having openings and passages leading to a chimney, causing a down draught of cool air around the pot. The depending upper electrode consists of four square carbon bars or cores, spaced apart and imbedded in a coked mass of crushed carbon and tar, protecting the cores from air. The resistance of the coked mass is comparatively high, largely restricting the heating effect of the current to the cores. The electrode is cooled by the current of air passing downward around the pot. The upper electrode is adjustably supported by a chain and is connected to two sets of cable-mains. Contact with the cores is made by thin copper plates, which may be silvered, compressed against the inner faces of the upper ends of the carbon cores by wedges and screws.

654,467, July 24, 1900, James Mactear, of London, England.

Resistance furnace, for heating a mass of wood charcoal or other catalytic substance. The heated mass is employed to convert a mixture of ammonia and carbon monoxide into ammonium cyanide. The furnace chamber is oblong and has sides of porcelain, closed at the top and bottom by removable horizontal plates. Inlet and outlet gas mains lead through these plates. The chamber is heated by a series of platinum resistance-coils, wound on vertical refractory rods and mounted in refractory tubes depending from the cover of the chamber, giving a uniform temperature throughout the charcoal.

655,779, Aug. 14, 1900, William Smith Horry, of New York, N. Y.

A uniform current at constant voltage is maintained by moving the furnace electrodes relatively to each other or to the furnace chamber, which may be fixed or movable. The furnace shown is of the Horry wheel type, two electrodes of opposite polarity depending into the wheel receptacle at one side. The wheel is rotated by a gear on its shaft, driven by a worm and electric motor. A two-phase motor is shown, which may be driven in either direction by two switches, employing mercury cups. Pins which enter these cups are carried by a tilting lever. Gravity normally tilts the lever to make contact with one set of mercury cups. As the current in the furnace mains increases, it acts through a transformer surrounding one main and a solenoid energized thereby to tilt the lever and close the other mercury cups, reversing the motor and moving the wheel receptacle in such direction as to decrease the current. The current from the transformer may actuate the motor switches through an electro magnetic relay, instead of directly. The motor may be direct current, or a steam, air or hydraulic engine may be used.

655,780, Aug. 14, 1900, William Smith Horry, of New York, N. Y.

A modification of the preceding furnace-control mechanism. A slow, even movement of the current-regulating element of the furnace is undesirable, causing the mixture in the furnace to form an arch which delays response to the regulation. The improved mechanism is intended to give the regulating element a quick, jerky movement, causing the mixture to fall at once into the zone of reduction. The mechanism shown is a long bar, extending past a number of furnaces of the Horry wheel type, and continuously reciprocated. Connected to the bar at each furnace is a vertical pivoted lever, the upper end of which carries two pawls, faced in opposite directions and respectively engaging two ratchet-wheels on a shaft geared to the furnace. The pawls are weighted to normally remain out of engagement with the ratchet-wheel, but a solenoid is arranged to magnetize each pawl and cause it to engage. The solenoids are respectively energized by switches electro-magnetically controlled by the furnace current, as in the preceding patent.

656,600, Aug. 21, 1900, R. Doolittle, of Chicago, Ill.

Arc type. The furnace used for manufacturing calcium carbide in accordance with the process of patent 656,599. The furnace is a tall, narrow stack, having a hopper and feed-screw at its upper end. Burners for gas or oil open into the top of the chamber. Superposed pairs of opposed electrodes pass through the side walls of the stack, near its middle. A gas-exhaust pipe leads from the lower part of the stack. A removable receptacle closes the lower end of the stack and receives the carbide; or the carbide may be tapped out. In a modification, the lower end of the stack is surrounded by a cooling jacket or coil, and air is injected into the bottom of the stack to cool and granulate the carbide. Grates are shown to subdivide the carbide into small streams as it flows downward to the blast.

656,930, Aug. 28, 1900, Wilhelm Borchers, of Aix-la-Chapelle, Germany.

Arc or resistance type, especially for the manufacture of calcium carbide. The furnace is a vertical or inclined stack having a hopper at its upper end and opposed electrodes passing through its sides. The lower portion of the stack is surrounded by a water-jacket, cooling the product below 400° C. The jacket may be used as a steam boiler for running an engine. The contents of the furnace are gradually removed at the lower end, allowing the pig or carbide to descend. Portions are then broken off by chisels, passing through openings in the walls. In the inclined type of furnace, a toothed roller extends through the lower end, engaging and pulling the carbide downward. Surrounding the feed-hopper is a gas outlet chamber containing peat carried on a perforated bottom, which acts to filter the escaping gases and prevent their carrying away combustible dust.

(*To be continued.*)

BOOK REVIEWS.

A Text Book of Ore Dressing. By Robert H. Richards, S.B., LL.D., assisted by E. S. Bardwell and E. G. Goodwin. 702 pages, 374 illustrations. Price, $5 net. New York: McGraw-Hill Book Company.

Our German masters in technology have an expressive term whose exact equivalent does not exist in English. They distinguish the "Lehrbuch" from the "Handbuch"; the former is the *teaching* book, the latter the *reference* book. Professor Richards has made this separation in the realm which he has made almost his private domain. His Ore Dressing is inimitable, unmatched in its thoroughness, but for that very reason unsuitable for teaching the subject. Messrs. Bardwell and Goodwin have condensed the matter from the larger four-volume work on Ore Dressing, and did the actual writing of this book to which Professor Richards contributed only the general outline, and the storehouse from which all the material was extracted.

Ore Dressing is mostly a concentrating operation, to render the valuable material more available and to facilitate its extraction and utilization. We may therefore say that one of the most useful contributions which Professor Richards has made to this subject, comparable perhaps to anything else he has accomplished in actual value to the profession, is the operation by which, with a concentration ratio of 4 to 1, the present volume has been extracted from its immediate predecessors and progenitors. The result is a "concentrate" whose value to students, teachers, learners, laborers, and masters in this field is almost inestimable. Its completeness and accuracy disarm criticism. Its parts are splendidly balanced, the treatment is equable and judicial, and altogether it is a "Lehrbuch" in the best sense and quality of that word.

* * *

Welding and Cutting Metals by the aid of Gases or Electricity. By Dr. **L. A. Groth**, Consulting Engineer. Cloth, 281 pages, and many illustrations. Price $3.00. D. Van Nostrand Company, New York, 1909.

This work contains a full account of the various processes mentioned in the title, particularly as practiced abroad. There are introductory notes on the gases employed and descriptions of the apparatus for using them are very detailed. A comparison is drawn between the oxy-hydrogen and oxy-acetylene methods, and the burners or blow-pipes suitable for each are illustrated and the construction shown. The welding of aluminium is given a special chapter. In this no mention is made of Mary W. Emme, who secured a U. S. patent for welding this metal, antedating Heræus.

The Goldschmidt aluminothermic welding process is accorded a considerable space. Particular attention is paid to electrical welding devices, those of Thomson being prominent, and numerous illustrations are given of the apparatus used. The subject of welding marine boilers forms a chapter by itself, as also the cutting of metals by the blow-pipe. There is a chapter on reports of the efficacy of acetylene welding, and a final chapter on legislation.

The work is attractive and well printed. The style, while interfered with by many extracts from reports and trade descriptions, has a slightly foreign tinge.

* * *

Solid Bitumens. Their Physical and Chemical Properties and Chemical Analysis, Together with a Treatise on the Chemical Technology of Bituminous Pavements. By **S. F. Peckham,** A. M., Chemist to the Department of Finance, City of New York. 8vo, 324 pages, 29 illustrations. Price, $5. New York and Chicago: Myron C. Clark Publishing Company.

A praiseworthy attempt to bring order out of confusion in a very dark corner of chemical technology.

Part I (89 pages) deals with the natural history of the bitumens, their history, distribution, derivation and classification.

Part II (172 pages) is the principal portion of this book, dealing with the chemistry of the bitumens, in eight chapters. The principal topics are the methods of ultimate analysis, proximate analysis, technical analysis, special tests, and the testing of auxiliary substances, such as street mixtures, cements, concretes. A person who knows little of the subject will find here a great store of information, carefully and systematically collated; the expert who knows almost all about the subject will doubtless find the presentation valuable for reference and discover some original ideas contributed by the author.

Part IV (19 pages) deals rather too briefly with the physical properties of the bitumens. In our comparative ignorance we had thought the physical properties would have been given nearly as extensive a consideration as the chemical properties;

certainly a larger part of the usefulness of the bitumens finds its basis in their characteristic physical properties.

Part V (38 pages) is a brief presentation of the chemical technology of bituminous streets, and is considerably below the level of the rest of the book. This subject is handled so well in Dr. Clifford Richardson's "The Modern Asphalt Pavement" that Mr. Peckham would have saved himself trouble and ended his book more fittingly by simply giving this reference, and not have run the risk of spoiling the good impression made by the rest of his book.

While the author's style is here and there stiff and his treatment at times controversial, yet his writing is almost always interesting and his arguments instructive, even when they fail of being convincing.

* * *

Beet Sugar Making and its Chemical Control. By **Y. Nikaido**, B.Sc., M.A. Cloth. Price, $3. Easton, Pa.: Chemical Publishing Company; and London, England: Williams and Northgate.

In the preface of this book it is stated that "the aim of this work is to aid those who are starting their career in the beet sugar manufacture, but lack systematic training in the technique thereof."

The preface likewise offers a list of references used by the author in the preparation of the book. These are, many of them, of elementary character, such as should be available either in a private or public library to any one, who would venture to enter the beet sugar industry either as employer or employee. The industry requires trained men, and has little tendency toward the training of men from the start. Our chief criticism of the book is that it contains so much that is elementary and inadequate to the accomplishment of the purpose intended.

The first four chapters might well be eliminated and this is also largely true of Chapter V.

For young employees in the laboratory, inexperienced in beet-sugar analysis, which is possibly the class referred to by the author in his preface, the descriptions of the apparatus and analytical methods in Chapters VI to IX inclusive, will be of some use, and they will be useful also to the laboratory director because it will obviate the necessity for much elementary instruction. Otherwise the work seems of doubtful value.

The book is well made, excellently printed and bound, contains a good table of contents and list of illustrations, and a well prepared index, the latter so often wanting in technical works.

The book also contains six tables which may prove useful in some laboratories, but which in presence of the valuable books of Kent, Spencer, Biedermann, Stammer, Stohmann, and Classen, which should form part of the equipment of the laboratory of every beet-sugar works, are entirely superfluous and need not have been copied.

* * *

A Course of Qualitative Chemical Analysis of Inorganic Substances, with explanatory notes, by **Olin Freeman Tower**, Ph.D., 83 pages. Price, $1. Philadelphia: P. Blakiston's Sons and Company.

In the opening paragraph of the preface of this book the author properly remarks "Some apology is necessary for the appearance of another text-book of Qualitative Analysis." And he further properly remarks "most of the briefer treatises on this subject are entirely inadequate in the treatment of the subject from the modern standpoint and in lack of precautionary directions which are necessary to secure delicacy of practice."

What the author says is intensely true. Most of the "briefer courses" of qualitative analysis are mere "short cuts" to success, and in this class of work as in every other the short cut to success generally ends in failure.

This should not be true, by any means, of the work before us. It is up to date in every respect. It is practically an application of the modern theories and laws of physical chemistry to chemical analysis, and it is entirely new.

In the introduction are discussed briefly dissociation, ionization, reversible reactions, reactions between ions, equilibrium and the solubility product, hydrolysis, amphoteric electrolytes, the electro-motive series and oxidation and reduction; briefly, the chief principles of physical chemistry.

Then follows "Directions for Laboratory Work" in application of the principles set forth and described in the introduction. These begin with "basic" analyses or, in less modern parlance, properties of and tests for bases and the separation and detection of these substances, followed by sections or chapters on "acid" analysis and dry substances.

In the basic analysis the grouping of the elements is but slightly different from that adopted in the older text-books on the subject.

The seven groups of the author are as follows: Silver Group, Copper-Tin Group, Iron Group, Zinc Group, Calcium Group, Magnesium Group and Potassium Group.

In the discussion of the several groups in basic analysis the author treats each one in the following order, preliminary tests, precipitation, notes on precipitation, analysis and notes on analysis.

In the Tin Group the author adds a brief chapter on interfering substances and notes thereon.

In acid analysis he establishes three classes: the sulphuric acid group, the halogen group, and the nitric acid group; and in this division he follows a method of treatment somewhat different from that used in the basic division. He makes two brief chapters, the first on preliminary tests, and the second on speed tests.

The preliminary tests seem to show the presence or absence of ammonium and sodium, and the condition of reduction of the compounds of iron and of the tin salts which may be present.

A page of notes following the preliminary tests calls attention to precautions to be observed and oversights which may occur, as well as reactions which may be produced in the operations and which may affect the ultimate result.

Following the matter on preliminary tests are chapters devoted to the separation of each of the groups respectively and the analysis of the group separated, and each separation and analysis is accompanied by a page of notes relating to the procedure. This arrangement of the systematic analysis is novel, but it seems effective and should serve well in laboratory works.

A brief chapter is likewise devoted to analysis in the dry way, corresponding with the usual course of blowpipe analysis.

The book proper closes with a chapter on the preparation of solutions, and here the author offers instruction in the preparation of solutions and of solid substances for examination; first for basic analysis, and second for the acid analysis.

An appendix is added giving a table of reagents in solution, showing specific gravity of the solution, concentration per cent, and approximate normal concentration. A few paragraphs on special methods for preparing certain solutions, which will prove useful in every laboratory, a table of degree of dissociation of electrolytes, and a series of optional methods for treatment of the several groups respectively, are also added.

On the front cover lining is a table of atomic weights of the principal elements and on the back cover lining is a solubility table. The book is also of convenient form and size for use on the laboratory table.

The book is well worthy of the considerate attention of instructors in qualitative analysis and will doubtless find a place in many of the laboratories of instruction throughout the country.

Metallurgical and Chemical Engineering

A Consolidation of Electrochemical and Metallurgical Industry and Iron and Steel Magazine

Vol. VIII. New York, April, 1910 No. 4

PUBLISHED MONTHLY BY THE

Electrochemical Publishing Company

239 West 39th Street, New York

Pittsburgh Office, 413 Lewis Block.
European Office, Hastings House, Norfolk St., Strand, London, Eng.

J. W. Richards, Ph.D.....*President*
E. F. Roeber, Ph.D.....*Editor and Secretary*
C. E. Whittlesey.....*Treasurer*

Yearly subscription price for United States, Mexico and United States dependencies, $2.00; for all other countries, $2.50 (European exchange, 10 shillings, 10 marks, 12.50 francs).

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Entered as Second-Class Matter at the Post Office at New York, N. Y., under the Act of Congress, March 3, 1879

New York, April, 1910.

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Pittsburgh Meeting of the Americal Electrochemical Society.

Pittsburgh, the center of the iron and steel industry, a center of chemical engineering activity, and the home of one of the largest electrical manufacturing companies of the world, will be the place of the annual convention of the American Electrochemical Society, to be held in the beginning of May. The selection of Pittsburgh was a logical one after the great success of the symposium of papers on the electrometallurgy of iron and steel, at last year's annual meeting in Niagara Falls. The electric steel furnace is no longer considered with suspicion or contempt by the practical iron and steel man; it is now being recognized as a very useful adjunct to Bessemer-converter and open-hearth steel plants and it is evident that it will be in extended use before many years in steel works all over the country. Among the visits which have been arranged for the convention, there are two to steel works where electric refining furnaces are in operation and in every other respect the very full program of excursions and social functions, printed elsewhere in this issue, is likewise extremely attractive. The program of papers, which has not yet been announced, will, without doubt, be no less interesting. The Pittsburgh meeting should become another milestone in the career of the American Electrochemical Society.

The Decomposition of Carbon Monoxide in the Presence of Catalytic Agents.

A positive definition of a catalytic agent seems difficult. When a substance simply by its presence (without undergoing any change or without participating in the reaction at all) starts a reaction that would otherwise not go on by itself, we call it a catalytic agent. Ostwald defines in general a catalytic agent as one that changes by its presence the velocity of a reaction without entering into the reaction. Perhaps we should add that we speak of a catalytic agent only, if we do not understand the rationale of its action. If we place a well amalgamated pure zinc rod into a sulphuric acid solution it will not dissolve, or only slowly; if we wind a small piece of platinum wire around the zinc rod, the zinc at once dissolves with strong evolution of hydrogen. Of course, we know that we have here a short-circuited galvanic cell, and because we understand the reason, we would hardly speak of the platinum as a catalytic agent in this case. It is the subjective element of mystery that should find expression in a complete positive definition of a catalytic agent. While contact processes are mysterious, they are extremely important and have come very much to the front. A very small amount of catalyzer often exerts a very great influence on the reaction.

* * *

Dr. Zimmermann's paper, printed in this issue, on the decomposition of carbon monoxide in the presence of catalytic

agents at certain temperatures, will be found very interesting from a purely scientific standpoint and with respect to the reactions in the blast furnace. He shows that carbon monoxide will decompose into elementary carbon and carbon dioxide at temperature near 300° C., with cobalt, for instance, as catalyzer. The reaction is still more interesting and curious if iron is used as catalyzer at 445° C. Iron first acts as catalyzer, decomposing CO into CO_2 and C. It is then oxidized according to the equation $Fe + CO_2 = FeO + CO$. The CO is again decomposed by catalytic action and so on. Iron is here not a pure catalizer, while carbon monoxide appears in the peculiar role of an oxidizing agent for iron.

Zinc and Lead Pigments.

In most processes of ore reduction every effort is made to avoid the production of dust and to prevent any oxidation of the valuable constituents of the ore. It is true that volatilization processes have been devised from time to time in which the metallic constituents are driven off as a volatile salt such as the chloride, but they have never proved successful. Lead and zinc ores, however, have been found amenable to volatilization as mixed oxides and sulphates, and their treatment in this manner may become an important adjunct to the pigment and paint industry. The extended use of zinc as a pigment is interesting in view of a tendency toward the proscription of lead in the customary form of white lead, or basic carbonate. This has been especially true in France and Belgium, where laws adverse to the lead industry have been passed. This matter is more fully referred to in an abstract in the Synopsis in this issue. The manufacture of zinc oxide in the United States dates from 1855. Simply stated, the process consists in burning in a blast furnace of suitable construction a mixture of zinc ore and coal or coke, recovering the volatilized zinc oxide in bag houses. Application of the same principle has been made to lead ores and to mixed zinc and lead ores, giving a basic lead sulphate and a mixture of zinc oxide and basic lead sulphate, respectively.

* * *

It will be apparent that the direct production of clean pigments in a process no more complicated than this would be a decided improvement over first reducing the metal from the ore and then reconverting it into pigment. Furthermore, the treatment of a mixed sulphide of zinc and lead is of great value, as it obviates not only the necessity of reduction, but of previous separation of the minerals and this is frequently difficult and never accomplished without loss. Since most of our ores of zinc and lead are sulphides, and therefore amenable to this process, it would seem to be a matter of less moment whether the manufacture and use of white lead are ultimately prohibited or not. Industry is quick to change to a process which promises both technical and financial betterment and never awaits an act of prohibition to drive it to something new. Perhaps the manufacture of pigment by volatilization of the metal from the ore will prove so much more desirable commercially than the corrosion of the reduced metal, that white lead will be retired without resort to legislation, be it wise or foolish. It should not be beyond the range of possibility that some pigment be found to supersede the basic carbonate of lead, and, indeed, if the advocates of basic sulphate of lead, with or without admixture of zinc oxide, are to be believed, the new pigments may be superior to the old.

* * *

If the process should be feasible and profitable, we have an almost unlimited and suitable ore supply, so that the production of zinc and lead pigments by the new method should be limited only by the demand which can be created for them. It is of no commercial importance whether the pigment is a basic carbonate or a basic sulphate, and a manufacturer would just as soon make one as the other. In any event, the lead industry can only profit by these advances and improvements. As a matter of fact, the production of white lead in the United States has been on the wane. This industry reached its zenith in 1904 when the production was 126,336 short tons. In 1907, which was an off year, it dropped to 111,409 tons and was only 116,628 tons in 1908. Imports also have shown a slight tendency to decrease in that period. The production of zinc oxide, on the other hand, has shown a steady increase since 1897, and when the output of zinc-lead pigment is added to this, a decided increase is noticeable. Imports of zinc oxide have also shown a general upward tendency since 1900.

The Chemical Industries of the United States.

The admirable address delivered by Prof. Charles E. Munroe before the American Institute of Chemical Engineers and of which the first part is published in the present issue, gives an excellent sketch, based on the best statistical data available, of the chemical industries of this country, their present very large extent and the much greater possibilities of the future. Professor Munroe is right in pointing out the difficulty of stating exactly what industries should be considered as chemical. It is, after all, a matter of definition and the popular idea of the term "chemical industries" will adapt itself to the economical development of the future; it will depend on the energy and success with which the chemist and chemical engineer make their influence felt in various industries in an advisory and managing capacity. To mention one instance, the metallurgical works proper were formerly considered as an adjunct to the mine and were managed by mining and mechanical engineers. But with progressing specialization, rendered necessary by the more difficult metallurgical problems to be solved for poorer or more complex ores, it has become practice that a metallurgical engineer is made the responsible head of the metallurgical operations proper. But the sphere of interest and activity of the metallurgical engineer is much nearer to that of the chemical engineer than of the mining engineer. In fact, metallurgical engineers may be considered as a special (and very important and influential) class of chemical engineers in general, just as metals are a special class of chemicals.

* * *

The statistical figures of Professor Munroe (of which a great deal more will be found in the concluding part of his paper to be published in our May issue) are interesting in various respects, but special attention should be called to the figures of the percentage changes of wages, cost of raw materials, and price of products in periods of 5 or 10 years. In estimates of the economical working of a chemical or metallurgical plant these changes should be taken carefully into account; to neglect of analysis of costs may be ascribed many financial diffi-

culties in the electric lighting and railway field. While Professor Munroe's figures show a steady and healthy growth of the chemical industries of this country, they also emphasize the shortcomings. The fact that the value of "chemicals, drugs and dyes," imported into this country in 1908, amounted to $73,000,000, shows vast possibilities of future developments. Further, almost 50 per cent of our total exports in 1908 consisted of material which had not undergone any degree of manufacture whatsoever. Professor Munroe is right in advising to put in this country into manufactured form the raw materials of this country; this should really be part of the policy of the conservation of natural resources. To one special point Professor Munroe draws special attention: We sit back glorying in our country, its wide extent, its rich resources; and we continue to sacrifice the fertility of our soils to the support of older civilizations and remain content, while ranging ourselves with those nations that live solely on their primary resources, since the balance of trade is in our favor. But chemists know that this condition cannot last; the average fertility of the soil has been growing steadily less. The utilization of the soil as a chemical factory is but one of the broad problems with which the chemist has to deal.

* * *

To one point, not mentioned in Professor Munroe's paper, attention may be called here. It is often said and it is perfectly true within certain limits that the chemical industry of this country is an imported industry and that in many respects we are still far behind Europe. This is true, for instance, for the coal-tar industry, which has undergone such an enormous and splendid development in Germany that it will be a long time before this country can hope to reach the same state of development. But in other fields this country has gone steadily ahead and in one special field this country has been the pioneer of a new chemical industry—the electrochemical industry, which in various ways is specifically American. It is characteristic that it was founded by engineers rather than chemists. Whether Dr. Acheson's prophecy will turn out true that this is the dawn of a great electrochemical era, the future will show. But there can be no doubt that electrochemical methods and processes will steadily increase in commercial importance in many chemical and metallurgical fields; and the brilliant success achieved by the electrochemical industries of this country will be a source of inspiration to all American chemists.

Compromise in Smelter Fume Suits.

Judicious compromise is the order of the day in disputes arising from smelter fumes. With the decision of Judge Hunt still fresh in our memories, wherein he dissolved the injunction obtained by the farmers against the Anaconda company in the famous Montana suit, it is gratifying to note that a compromise has been effected between the litigants in a similar case in Shasta County, California. Judge Hunt's decision in Montana was broadly based on the principle of the greatest good for the greatest number. He regarded smelting as of primarily greater importance than agriculture to the Deer Lodge Valley and dissolved the farmers' injunction on the further ground that the smelting company had exhausted all means known to modern technical skill to control the enormous volume of fume which the plant was producing.

While conditions in California and Montana are not exactly comparable on account of the greater magnitude of metallurgical operations in the latter State, a distinctly progressive step has been made in the compromise of the California suit in that the smelting companies recognize both the possibility and advisability of suppressing fume. So successfully have the chemists and engineers of the smelting companies attacked the fume problem during the past two years, fighting for their very existence, as it seemed, that they have improved greatly on conditions as they existed during the Anaconda suits. Hence any plea of inability on the part of the smelters could not avail now. By the terms of the California compromise, the Mammoth Copper Company agrees to equip its plant with neutralizing and bag-house devices for wholly suppressing the sulphuric and arsenious acids now discharged into the air. The scheme to be used will probably be that adopted in Salt Lake Valley on account of similar suits, in which the sulphuric acid is neutralized by zinc oxide and all solid matter filtered from the gases through the bag house. An additional point in the California compromise is that the gases finally discharged into the air shall not contain more than seventy-five hundredths of 1 per cent of sulphur dioxide by volume. This will require the dilution of the gases by admission of large volumes of air. Both parties to the suit are granted the right and privilege to make inspections to see that the provisions of the compromise are carried out. The smelting company was allowed 90 days to carry out its work.

* * *

Referring again to the conditions in Montana, it may be said that the fume question is by no means permanently settled there. An appeal from Judge Hunt's decision is now in the Circuit Court. Whether the smelting company can be compelled to eliminate the fume nuisance or close down is a question. The magnitude of operations at Anaconda gives rise to conditions that can scarcely be appreciated on first thought. The company has taken cognizance of every known method of handling fume and has had to abandon all on account of the prohibitive expense or inefficiency. Wherefore it is not fair to assume that what is an efficient system for a smaller smelter might just as well be adopted at Anaconda. Many months ago the Anaconda company presented to the court an exhaustive report on the technology and cost of various systems for eliminating fume and showed their impracticability as far as that smelter was concerned. Mr. Mathewson, in charge of smelting operations, showed that a spraying system would give an acid mud difficult to dispose of, and the plant would cost $3,000,000. A sufficiently capacious bag house would cost $2,750,000, and entail a daily expense of $1,800. For the freezing system the plant would cost $4,000,000 and the cost of operation would be $10,000 per day. A zinc oxide neutralizing plant, such as is used in Utah and will be used in California, would cost $3,000,000 and require 500 tons of zinc ore daily. Thus, it will be seen that the problem is one of no mean proportions, although it is not wholly beyond the range of expectation that it will be solved, at least, in a measure. In the meantime, other plants are working on various schemes in a commendable effort to avoid conflicts with other local interests, and, perhaps, the smelter fume suit in the West will soon become a thing of the past.

Pittsburgh Meeting of the American Electrochemical Society.

The preliminary program of the seventeenth general meeting of the American Electrochemical Society, to be held in Pittsburgh on May 5, 6 and 7, has just been issued. The hotel headquarters will be at the Fort Pitt Hotel, quite near the Union Station.

On **Wednesday,** May 4, the secretary will be at the Fort Pitt Hotel and members and guests can register and receive badges after 12 m. At 2 p. m. it is proposed, if a sufficient number of members will be present, to make a visit of inspection to the Technological Testing Plant of the U. S. Geological Survey. Special tests of fuels, explosives and other materials of the highest technical interest will be arranged by the engineers in charge. Whether this visit is to be made will depend on the attendance of the members; all who can be present on this afternoon for this purpose should inform, in advance, the secretary, Dr. J. W. Richards, Lehigh University, South Bethlehem, Pa. Final notice as to whether the visit is assured will be given in our next issue.

On **Thursday,** May 5, a meeting of the board of directors will be held at 9 a. m. at the Fort Pitt Hotel to be followed at 10 a. m. by the annual business meeting of the Society, at which the report of the annual election and the annual report of the board of directors will be presented. This will be followed by a professional session in which a number of papers will be read and discussed.

For the afternoon of Thursday, a number of interesting excursions and visits have been arranged as follows: Alternatives: Park Company's crucible steel mills, Carnegie Steel Company's dried-blast plant at Isabella Furnace, Jones & Laughlin's Steel Works (Talbot continuous steel process), Pennsylvania Lead Smelting Company., Nernst Lamp Factory, Oxy-Acetylene Welding Company.

In the evening at 8 p. m. Dr. L. H. Backeland will deliver his presidential address at the Fort Pitt Hotel, to be followed by the reading and discussion of papers of general interest.

Friday, May 6, will be devoted to an all-day excursion, leaving Union Station at 8:30 a. m. and visiting the Allegheny Plate Glass Works at Glassmere, the Westinghouse Electric Works at East Pittsburgh, the Firth-Stirling Works at Demmler (where the Héroult electric steel furnace is in operation), the Carnegie Steel Works at Homestead (where a new type of combined open-hearth and electric furnace is in operation). Lunch will be served on the train; return to Pittsburgh at about 5:30 p. m. Cost of the excursion, $1.50 each.

In the evening at 7 p. m. an informal subscription dinner will be held at the Fort Pitt Hotel, followed by "light entertainment by Section Q."

On **Saturday,** May 7, a professional session for the reading and discussion of papers will be held at the University of Pittsburgh, Grant Boulevard. At 1 p. m. lunch will be taken at the Hotel Schenley.

For Saturday afternoon at 2.30 a general public meeting has been planned, the program being of general interest. It includes addresses on the present status of electrochemical industries (illustrated), on the advantages of Pittsburgh as an electrochemical center, and, by an eminent speaker, on the conservation of natural resources of power. Full details of this meeting, which it is hoped to attract a large audience and to arouse general public interest, will be given later on.

This meeting will be followed by visits to the Carnegie Institute and Museums, Carnegie Technical Schools, University of Pittsburgh, and other points of interest in the vicinity of the Carnegie Music Hall

In the evening members staying in the city will be invited to an "at home" at one of the local clubs.

The above program is subject to modification in any of its details. The final program, including the titles of papers to be presented, will be printed in our next issue.

Ladies are cordially welcomed to the meeting; their comfort and entertainment will be in the hands of a Ladies' Committee, and they are invited to attend the sessions, the dinner on May 6, the luncheon on May 7 and the excursions on May 4, 5 and 7.

A fine program and a highly interesting meeting is assured and the Local Pittsburgh Committee hopes to see a record-breaking attendance at the first gathering of the Society in Pittsburgh.

American Institute of Chemical Engineers.

The summer meeting of the American Institute of Chemical Engineers will be held at Niagara Falls, N. Y., June 22 to 24, 1910.

A prominent feature of the meeting will be visits to the important and interesting chemical industries in this locality.

American Foundrymen's Association.

The convention of the Allied Foundry Associations will be held in Detroit, Mich., June 6-10, 1910. Committees of the American Foundrymen's Association and American Brass Founders' Association are engaged in making the convention the best that has yet been held. An exhibit of mechanical equipment will be an interesting feature and extensive arrangements are being made for visitation of plants. The headquarters of the convention will be at the Pontchartrain Hotel. Information regarding the convention can be secured from Mr. Frank T. F. Stephenson, chairman of the general convention committee, 12-14 West Atwater Street, Detroit, Mich.

Chandler Testimonial.

In our February issue (page 65) we recorded the fact that after a very long period, almost a lifetime, of the most useful activity as head of the department of chemistry and dean of the School of Mines of Columbia University, Dr. Charles F. Chandler will retire from his duties at the end of the coming term, to be appointed professor emeritus of chemistry.

To commemorate this important epoch of his life the following organizations have taken the initiative in planning for a testimonial to be tendered to Dr. Chandler by the chemists of the United States: The Chemists' Club, the Society of Chemical Industry, the American Chemical Society, the American Electrochemical Society, the American Institute of Chemical Engineers and the Verein Deutscher Chemiker.

The form of this testimonial has been arranged, firstly, in form of a banquet at the Waldorf-Astoria, on Saturday, April 30, at 7 p. m. The cost of each dinner ticket will be $5, a sum, which, while not sufficient to defray the expenses, will insure a large attendance. Ladies will be admitted to the galleries and light refreshment served them at a nominal charge of $1 per person.

At this banquet a bronze bust of Dr. Chandler, in heroic size, to be executed by the famous sculptor, Mr. J. Scott Hartley, will be presented to him. This bust will finally be presented to the Chandler Museum of Columbia University. A replica of this bust is to be presented to Mrs. Chandler.

Finally, a Chandler Testimonial Fund will be created for the purpose of purchasing books for the library of the Chemists' Club.

Subscriptions for the banquet and the general expenses of the testimonial are to be sent to the treasurer of the committee of arrangements, Dr. Morris Loeb, 273 Madison Avenue, New York.

Mr. I. F. Stone is the chairman and Dr. Charles F. McKenna, 50 Church Street, New York, the secretary of the committee.

International Congress of Inventors.

The "International Congress of Inventors" was established in 1906, with headquarters in Rochester, N. Y., and incorporated in 1907. One of the objects of the congress is "to endeavor to establish a standing for a United States patent independent of any court action." Mr. George F. Gallagher is the president, Mr. Ralph T. Olcott the secretary. The headquarters are at 123-125 Ellwanger & Barry Building, Rochester, N. Y.

Under the auspices of the congress a "national convention of inventors" and exhibition of patents and models will be held at Rochester, N. Y., from June 13 to 18, 1910.

The Iron and Steel Market.

Each year in the American iron trade has shown a distinct color, and the current one is proving no exception, presenting a color so novel as to perplex the majority. We have had the spectacular boom of 1899, the quick reaction of 1900, the steady growth in prices and demand in 1901, the feverish activity of 1902, the reaction of 1903, the melancholy dullness of 1904, the sharp revival of 1905, the sustained improvements of 1906, the storm of 1907, the much talk and little performance of 1908, and the stupendous recovery of 1909. Through all these vicissitudes there has been created the machinery for doing an enormous amount of business with a minimum of noise and confusion, but unprepared by recent example for such performance, the iron and steel trade stands confused and is alternately looking for improvement and fearing a reaction, at a time when production of both pig iron and steel is at the highest rate ever recorded, passing by 20 per cent the average of the best previous calendar year, while close to 100 per cent of the tonnage produced is passing quietly into consumption and the small tonnage which may possibly be accumulating in the hands of mills and jobbers is bound to be distributed promptly on the appearance of really spring weather conditions.

This line of treatment may seem fanciful, but the solid ground for it can be shown by analysis of history. One must, in fact, go back to 1890 to find similar conditions. Then, despite the fact that the industry was much smaller than now, identical conditions were presented. There had been a reaction in 1887, a dull period in 1888, and a somewhat feverish recovery in 1889. In the early months of 1890 conditions were precisely similar, at all comparable points, with the present. Tonnage was enormous according to the then scale, yet the market moved doubtfully and prices showed a slight receding tendency. Throughout that year the market doubts continued, and yet it ended by disclosing a pig iron output 21 per cent in excess of the best previous record. If like conditions produce like results, we shall find, for this year, a maintenance of the present production rate, without realizing where all the demand comes from, and a substantial maintenance of prices, though with occasional slight recessions. Hitherto in recent years when the demand for iron and steel reached new tonnage proportions it strained the existing capacity and resulted in delayed deliveries and advancing prices; at this time there is no strain upon productive capacity, for it has been very largely increased since the last test, in the first 10 months of 1907.

Late in February a slight improvement occurred in demand for finished steel products, which was continued until nearly the middle of March. The tone of the market improved concurrently, and confident predictions were made of a continuance of the improvement. About the middle of the month the market experienced a slight chill, from which in due course it will no doubt recover.

Pig Iron.

The theory that because Lake Superior ores were advanced 50 cents for the coming season pig iron for second-half delivery would necessarily be higher than for first-half delivery was abandoned when, early in the month, the Westinghouse Electric & Manufacturing Company bought foundry iron for second-half delivery, 5000 tons for its Allegheny works, in Pittsburgh, at $16.50 delivered, and 10,000 tons for its Cleveland works at $16.75 delivered. The Pittsburgh iron was at a lower price than had been done for early delivery, while the Cleveland iron was at no more than a trifle above the ruling market for early delivery. In other districts pig-iron business was done at slight recessions from former prices, and Southern iron was done at $12.50, Birmingham, for May and June delivery. While the pig-iron market may not show a condition of total depravity, it is at least not in good form that when Northern iron failed to show a higher price for second half, on account of the advance in Lake ores, the Southern pig-iron industry, which is self-contained, should show lower prices for prompt than for second-half delivery, yet such is the case. Prices at valley furnaces (90 cents higher delivered Pittsburgh) are substantially as follows: Basic, No. 2 foundry and malleable, $16 to $16.25; gray forge, $15.25 to $15.50; Bessemer, $17.75. The price on Bessemer represents a decline of 25 cents, early in the month, brought about by sales of 10,000 tons to the Cambria Steel Company and 7000 tons to the Lackawanna at a nominal price of $18, valley furnace, but with an allowance of three months' interest.

Steel.

A large tonnage of billets and sheet bars has been moving on old contracts, while new business has been very light. There is occasional demand for open-hearth billets, particularly small sizes, which are scarce. Quotations remain at $27 to $27.50 for Bessemer billets, $27.50 to $28 for open-hearth billets, $28.50 to $29 for sheet bars and $33 for rods, all f.o.b. maker's mill, Pittsburgh or Youngstown districts. The Carnegie Steel Company has put in operation the last of the four 55-ton open-hearth furnaces at the Homestead Steel Works on which construction was commenced last summer. This gives the Homestead works 64 open-hearth furnaces and one electric furnace.

Finished Products.

Prices in the main show no change, but one or two products have trended slightly to weakness. Iron bars are a trifle easier in the Pittsburgh and valley districts, but are still quotable at $1.65 to $1.70, Pittsburgh, while the Chicago market has declined $1 a ton to $1.50, Chicago, and the St. Louis market has experienced a rather sharp drop of $2 a ton, sales being made at $1.40 at mill, St. Louis district. Shading of a dollar a ton has appeared in corrugated sheets, flat sheets being well maintained. Wire products show less shading than a month ago. Line pipe is being cut rather freely, while merchant pipe is nominally unchanged. While the price of $1.55, Pittsburgh, on plates and shapes remains for the Pittsburgh and Western markets, the shading of a dollar a ton from this basis continues in the Eastern market.

While there is no general buying movement in finished steel product, a good tonnage is being booked from week to week, the aggregate falling only a little short of current production, and with the large volume of business on books in most lines it would take a long time for orders to run out, while progress into the spring is expected to take up the slack very soon.

Regular prices, f.o.b. Pittsburgh, are as follows:

Standard steel rails, $28 for Bessemer; $30 for open-hearth, f.o.b. mill, except Colorado.

Plates, $1.55 for tank quality, ¼ in. and heavier.

Shapes, $1.55 for I-beams and channels, 15 in. and under, zees and angles, 2 x 3 and larger.

Steel bars, $1.45 to $1.50, base.

Iron bars, $1.65 to $1.70, Pittsburgh; $1.55 to $1.60, Philadelphia; $1.50 to $1.55, Chicago; $1.40 to $1.45, St. Louis.

Wire nails, $1.85, base; plain wire, $1.65; galvanized barb wire, $2.15.

Black sheets, 28-gage, $2.40; galvanized, $3.50; blue annealed, 10-gage, $1.75 to $1.00; painted corrugated roofing, $1.70 per square; galvanized, $3.

Tin plates, $3.60 for 100-lb. cokes.

American Institute of Electrical Engineers.

At Charlotte, N. C., March 30-31 and April 1, will be held a meeting of the American Institute of Electrical Engineers.

Seven papers will be presented at this meeting as follows: "Electric Drive in Textile Mills," by Mr. Albert Milmow; "Gas Engines in City Railway and Light Service," by Mr. E. D. Latta, Jr., Charlotte Electric Railway, Light & Power Company; "The proportioning of Electrodes for Electric Furnaces," by Mr. Carl Hering, of Philadelphia; "Modifications of Hering's Law of Furnace Electrodes," by Dr. A. E. Kennelly, of Harvard University; "Some Demonstrations of Lightning Phenomena," by Prof. E. E. F. Creighton, of Schenectady; "Economics of Hydroelectric Plants," by Mr. W. S. Lee, chief engineer, Southern Power Company; "Method of Protecting Insulators from Lightning and Power Effects," by Mr. L. C. Nicholson, electrical engineer, Niagara, Lockport & Ontario Power Company, Buffalo, N. Y.

A special train de luxe for visitors from the North will be provided by the Pennsylvania Railroad should at least 100 passengers be assured. Applications for reservations should be made in advance to the A. I. E. E. headquarters, 29 West Thirty-ninth Street, New York, in order that the railroad may know if the condition as to number of passengers will be complied with. The train will be made up of parlor, smoking car, observation car, and the most modern sleepers and compartment cars available, and will be electrically lighted throughout.

As the climate of Charlotte is particularly delightful during the season when the meeting will be held, a large attendance is anticipated, which may reach 500 members and guests. Such a meeting is a novelty in the South, and the prominent men of Charlotte and surrounding towns are particularly interested in making it a great success. The committee on entertainment are much pleased with the lively interest taken by the ladies of Charlotte in the social program, and it is anticipated that the reception and dance in the Auditorium on Thursday evening, March 31, will be a particularly agreeable affair. Another feature that doubtless will be much appreciated by visitors is the trip arranged for the delegates, their wives and friends on Friday, April 1, by special train, to the two great generating plants at Great Falls and Rocky Creek, a detail of which is a Southern barbecue near the site of the former. On Wednesday afternoon there will be a visit to some of the electrically operated cotton mills near Charlotte, and on Thursday afternoon a reception to the visiting ladies will be given by Mrs. Stuart W. Cramer at her residence.

CORRESPONDENCE

Efficiency of Drying Machines.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—The exact use of words, always a desideratum, is especially necessary in scientific and technical literature and writers on technical matters cannot be too careful in choice of words, precisely to express what is meant.

As far as I am aware, the word "efficiency" as applied to machines in the accepted technical use, always means the ratio of actual useful effect to the theoretical or ideally perfect effect.

The "efficiency" of a machine is expressed by a fraction, the numerator being the attained result and the denominator the theoretical possibility, or

$$\text{Efficiency} = \frac{\text{Actual result}}{\text{Theoretical possibility}}$$

Of course, if expressed as a percentage, it will be 100 times this fraction.

Now the denominator of the above fraction, the theoretically possible result, is never practically attainable with any machine.

If we consider, for instance, the efficiency of an engine, 100 per cent efficiency would mean the ideal theoretical result of an engine working between two given temperatures, T_1 and T_2, and any losses whatsoever in a practical machine would reduce the efficiency, notwithstanding that some, at any rate, of such losses are quite inseparable from the operation of any practical machine.

Similarly with a boiler, "100 per cent efficiency" would mean the complete utilization of the heating power of the fuel in evaporation.

The fact that the waste gases must necessarily be discharged at a temperature higher than that of the steam generated makes it impossible that any practical boiler can reach 100 per cent efficiency. The loss is an entirely necessary one, but nevertheless it equally detracts from the "efficiency" of the boiler.

In an article on the "Efficiency Test of a Direct-Heat Dryer," in your March issue (page 162), efficiency is used in an entirely different sense and one, I believe, quite unwarranted by ordinary scientific usage.

To evaporate a given weight of water from any given temperature requires a definite amount of heat, let us say *a* b.t.u. A certain quantity of fuel will furnish a definite number of heat units, let us say *b* b.t.u.

A drying machine receiving *b* b.t.u. from the fuel burned and utilizing *a* units in *actual useful evaporation* has an "efficiency" equal to the ratio of *a* to *b*.

Now there are certain necessary losses in any drying machine which are well enumerated in the article in question. All of these losses reduce the numerator of the above fraction, and, in any consideration of "efficiency," all of them must be charged against the machine.

Among these losses is that due to heating up the material to the temperature at which it is discharged from the machine. This particular source of loss does not differ from any of the other sources of loss. It is, indeed, a variable, dependent upon the type of machine and upon the material to be dried. Its magnitude will vary with different materials. Nevertheless, in considering the efficiency of a dryer in any given case, the loss due to this cause must be charged against the machine, precisely as must loss from any other cause.

The writer of the article under discussion, however, *credits* the dryer with the heat diverted from useful work in this manner. In other words, he adds the heat required to heat the material to the discharge temperature to the heat required for actual evaporation, and calls the sum of the two "the theoretical heat required."

The ratio between this so-called "theoretical heat required" and that actually expended in useful evaporation he calls the "efficiency."

This is an entirely incorrect use of terms. The true efficiency of the dryer in the case cited is the ratio of heat received to heat utilized in useful work, or (taking the figures in b.t.u. from the article referred to) [illegible], or in round numbers, 54.5 per cent "efficiency."

There is no reason why this particular form of loss should be credited to the dryer any more than any other form of loss. It is indeed an unavoidable loss, but its magnitude is largely dependent upon the type of dryer. In the case discussed, the material was discharged at a very high temperature (224° Fahr.) and consequently the loss was high, notwithstanding the low specific heat of the material.

Had the machine been of a type that discharged its product at a lower temperature, the loss from this source would have been less and the "efficiency" greater.

It is perhaps proper to add that the "efficiency" of a drying machine is a relative matter and depends on the material to be dried.

The efficiency of a drying machine can only be stated for any particular case and may vary widely with different materials.

In the case in point, the material handled (sand) contained only 5 per cent moisture to begin with. Therefore, the weight of dried material discharged was very great relatively to the evaporation, and this condition is not favorable to the realiza-

tion of a high efficiency, particularly in a counter-current machine delivering its product at a high temperature.

The foregoing is not a discussion of any type of dryer, but of a proper use of terms. The performance of the dryer in the test was fair, but in no way remarkable. It evaporated, according to the figures given, 5.62 lb. of water per pound of (very bad) coal fired, or 6.25 lb. water per pound of dry coal. This is probably well up to the average performance of dryers and a good deal nearer the truth than the absurd evaporation figures frequently claimed. Only last month we were favored with an account in your columns of a dryer evaporating 16 lb. of water per pound of combustibles (page 108 of February issue). Such figures are immune from criticism by their very absurdity.

While the figures given in regard to efficiency test under discussion are fairly consistent, they lead to some remarkable conclusions.

The discharge temperature of waste gases is given as 100° Fahr. and the weight of air discharged as 31,905 lb. per hour. This air has to carry off 2250 lb. of water, or each pound of air must carry 0.0705 lb. of water. To do this it would have to be supersaturated to the extent of about 57 per cent; that is, the discharged air would not only require to be completely saturated with vapor, but must carry an additional burden of water in the form of mist (not vapor) equal to 57 per cent of its saturation water. To put it otherwise, something less than two-thirds of the water discharged was actually "evaporated," the rest being discharged, not as vapor, but as liquid water (mist). Of course, it does not matter in what form the water is got rid of as far as the efficiency of the dryer is concerned. Indeed, if it is possible to discharge one-third of the water as liquid water, it opens up a way to extraordinary possibilities in drying economies. Ordinarily, however, it is sufficiently difficult to attain complete saturation, let alone supersaturation.

Bayonne, N. J. W. E. Wadman.

* * *

To the Editor of Metallurgical and Chemical Engineering:

Sir:—The above criticism by Mr. Wadman of the dryer test, published in your March issue, was forestalled by paragraph three under heading "Heat in Dried Material." There it was stated: "It is, of course, a matter of definition how much of the heat which is carried off by the dried material should be considered as a loss," and as I stated my position in the matter there could be no possible deception and the discussion resolves itself into an academic discussion of terms which is unprofitable.

I still contend that my method of computation is a fair one. There is a decided difference between the loss in heating the drying material and the other losses enumerated. To attempt to dry sand, rock or most other inorganic materials at a low temperature by absorption would be absurd on account of the small capacity such a process could give. I therefore assume it is *necessary* to heat such material to 212°, just as necessary, in fact, as to heat and evaporate the water contained, so the dryer should be credited in doing this work. It is not necessary to heat the material above 212° (except in special cases); it is not necessary to carry off inconsumed carbon nor to have an excessive temperature of the exhaust air, not of the shell; therefore, such losses should be charged to the dryer.

Regarding the condition of the water and air of the exhaust, the air is probably fully saturated and is accompanied by a considerable quantity of watery vapor which can be as plainly seen as the exhaust from a steam engine. With the type of dryer from which the test in your March issue was made, it is often the case in drying clay or other very wet materials to evaporate 4500 lb. of water per hour with not over 6000 cu. ft. of air per minute passing through the machine. This means there is 88 grains of water carried off for each cubic foot of air, while the air if completely saturated would not hold over 21 grains per cubic foot.

New York City. W. B. Ruggles.

Energy Losses in Electrodes.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—In the article by the writer on "Current Densities and Energy Losses in Electrodes," published on page 26 of your January issue, there occur some misprints, which should be corrected as follows:

Page 29, second column, second paragragh, line six, stands: "in the case," read: is the case.

Page 30, first column, first paragragh, line three, stands: "in the case," read: is the case.

Page 30, second column, fourth paragraph, line two, stands: (14), read: (15).

Page 31, first column, third paragraph, line eight, stands: $C_1\sqrt{1/a}$, read: $C_1\sqrt{1/a}$.

Page 31, second column, fourth paragraph, line nine, stands: agreement, read: square root.

Page 32, first column, sixth paragraph, line nine, stands: 1.07 volt, read: 0.07 volt.

Page 32, second column, sixth paragraph, line five, stands: extreme, read: electric.

Page 33, first column, sixth paragraph, line six, stands: correction factor, read: correction factor $\sqrt{1/2\,(1-c)}$.

Page 33, first column, eighth paragraph, line five, stands: $\sqrt{\frac{1}{2}\,(1-c)}$, read: $\sqrt{1/2\,(1-c)}$.

With reference to the editorial remarks in the same issue regarding this article the writer wants to point out, in order to avoid misconceptions, that his introduction of a coefficient c, appearing as a factor to the Joulean heat loss, ought not to be interpreted as a lumping together of the heat leakage into the surrounding medium with half the Joulean heat loss, but as an expression for the fact that this heat leakage will reduce the efficiency of the "check-valve action" of the Joulean heat upon the heat conduction from the charge. Using the language of the editorial, where H is the heat which would flow if there was no current, the electrode loss is expressed in the following way, when there is heat leakage into the wall: $[H-(1-c)\,i^2r] + i^2r$ as against the theoretical case when no such leakage occurs: $[H-\frac{1}{2}\,i^2r] + i^2r$.

Höganäs, Sweden. Jakob Forssell.

* * *

Silundum.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—From the letter of Dr. E. G. Acheson in your issue of February I learn that he did not express himself, as reported, at the meeting of the American Electrochemical Society. I see, however, in your number of January that you refer, page 64, to silundum as having the same chemical compositions as siloxicon.

Dr. Acheson states that "in analyzing the possible reactions and the products that may be obtained from these three elements (carbon, silicon, and oxygen) we find that the furnace content may be so proportioned and the furnace so operated as to produce either graphite, metallical silicon, carborundum or amorphous carbide of silicon, or we may have all the three elements so associated as to produce siloxicon. As to the possibility of any other product being made, I am not advised, but the foregoing were old before the world was informed of Mr. Bölling's experiments."

I stated in my original article: "A process was worked out and patented by which it is possible to convert any piece of carbon, in form of tubes, round or square, rods, bricks, crucibles, pieces of coke, charcoal into *silicon carbide*. Silicon carbide was known so far only in the amorphous and crystalline state, and it was supposed that it was formed by the sublimation of silicon and carbon at high temperatures." Silundum, as the new material has been named, or silicified carbon, is, therefore, a product that is obtained when carbon is heated in a vapor of silicon "and, farther on, silundum is another form of silicon carbide."

I am of the opinion that my original article on silundum states clearly that it is a silicon carbide, and I do not see any reason why a discussion should take place on the chemical composition of silundum. If Dr. Acheson reads my original paper he will find that silundum falls under the products named in his letter. I do not claim to have discovered a *new chemical compound*. What I claim is the discovery of a new process to convert any piece of carbon into silicon carbide or covering it with silicon carbide preserving its original shape.

Silundum contains silicon in various percentages. Pieces are found that look like carbon, but are incombustible, others in a farther state of conversion have the appearance of siloxicon and amorphous silicon carbide. The next state is a material that has a metallic appearance; it contains the largest amount of silicon. If pieces of any of these modifications are heated to a higher degree, silundum crystallizes and is converted into carborundum.

Silundum may be converted into graphite in modifying the charge. There seems to be some probability that under certain conditions the black diamond may be obtained from silundum by depriving it of its silicon.

I may say that the hardest modification of silundum is used in some places for boring purposes, replacing the diamond.

Oberwesel, Germany. F. Bölling.

* * *

The Trust from the Standpoint of an Engineer.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—In the March issue of this magazine there appeared an interesting discussion of "The Trust from the Standpoint of an Engineer," by Mr. W. McA. Johnson, evidently inspired by reading the brief for the defense in the action against the American Tobacco Company, brought under the so-called "Sherman Anti-Trust Law." In view of certain statements which appeared in Mr. Johnson's letter, a further short discussion may be of general interest, especially as regards the Supreme Court of the United States.

The fact that the Federal Anti-Trust Act passed in 1890 has remained on the statute books unamended for 20 years tends to show that it was not the product of a discontented minority, but was and is the crystallization of public opinion into law, a law that is worthy of a conclusive test both as to legality and practicability. When the Supreme Court decides a case under this statute, the duty of the justices is simply to construe the law and apply it to the concrete case presented, leaving to the law-making body the responsibility as to the statute's expediency. Now the members of the court will bring to the decision of the Tobacco Company case both ripened expert knowledge of law and an impartial habit of mind, and, furthermore, they will examine *all* the facts in the light of arguments presented by the prosecution, as well as by the defense.

If the Court decides that the facts proven make the Tobacco Company guilty of a misdemeanor as defined in the law made by Congress, Mr. Johnson implies that it will forfeit a large share of his respect. A man who is not an expert in law and who has read simply the argument for the defense (made by a lawyer hired solely because of his ability to present one side of the question) is hardly in a position to criticize a decision reached by the highest of our courts after an impartial and complete examination of all the evidence.

The statement that the "trusts" are necessarily the survival of the fittest and are therefore assured of longevity depends on the meaning attached to the word fit. The simple plan that "they should take who have the power and they should keep who can" has undergone extensive revision in the ascent from barbarism to civilization, so that society to-day compels the takers and keepers to confine their operations within the bounds of necessary laws, which are as real a product of an evolutionary process as anything else.

Mr. Johnson considers one great corporation "fit" because it is approved by public opinion and evidence of this general approval he finds in the fact that the people have not demolished it. The conclusiveness of this reasoning is open to serious question. It is a safe opinion that the majority of American citizens would not resort to physical violence until wrongs far greater than any ever charged against the Standard Oil Company have been proven to them, and then not until the Government, which is established for preserving order and justice, is shown to be unable to cope with the situation. Rather will the average man wait to see the status of the giant monopoly established by the judicial machinery created for that purpose, namely, the Supreme Court of the United States. The legal status being determined (one factor in the judgment of its "fitness") its survival as an economic power will be decided by its future life and prosperity.

If manipulation of prices or other facts show that the trust is guilty of a misdemeanor as defined in the "Sherman Act," and Mr. Johnson thinks those facts should not constitute a misdemeanor, his objections should be aimed not at the Court, but at the law-making body and at the dormant public opinion which permits the law to remain unamended.

Mr. Johnson's dismissal of rebates as special favors to friends and as similar to a discount given to a grocer who promises to handle a single brand of goods, seems inaccurate, for the analogy is superficial. A grocer's business is private, while the railroads are "common-carriers," a class of beings who, even before the steam locomotive was dreamed of, were considered to be under very special obligations to the public. The railroads are the highways of the nation, deriving their rights, the land on which they are built, and their corporate existence from the State. The prevailing idea is that in giving up these highways of commerce to be developed by private enterprise, the State should insist that all its citizens be treated on an equal basis. Instead of being a matter of friendship between carrier and shipper the transaction was often in the nature of a hold-up. The evils of rebating are matters of business history; and public opinion, forcibly expressing itself in the Interstate Commerce Act, has rightly banished the practice.

The "trusts" which control the selling prices of commodities seem in certain industries to be the most efficient form of organization now known and are (as Mr. Johnson states) the result of a process of evolution following definite economic laws. But there exist no economic laws compelling corporation managers to be scrupulous in controlling prices. A few of them at least have not been so. There is nothing in any evolutionary theory that precludes adequate protection of the public welfare; indeed, the general welfare demands that these leaders be subject to the reasonable restraints of the law. It is the opinion of the majority of competent judges that the English common law, which was gray-headed before the beginning of our industrial era and was chiefly developed in a state of society widely different from our own, is inadequate unless revised and amplified. The necessity for such modification is at once a reason and justification for legislation by Congress. This will tend toward an equitable division of the profits resulting from the corporate form of organization among the three parties of interest, namely, the corporation itself, its employees, and the consuming public.

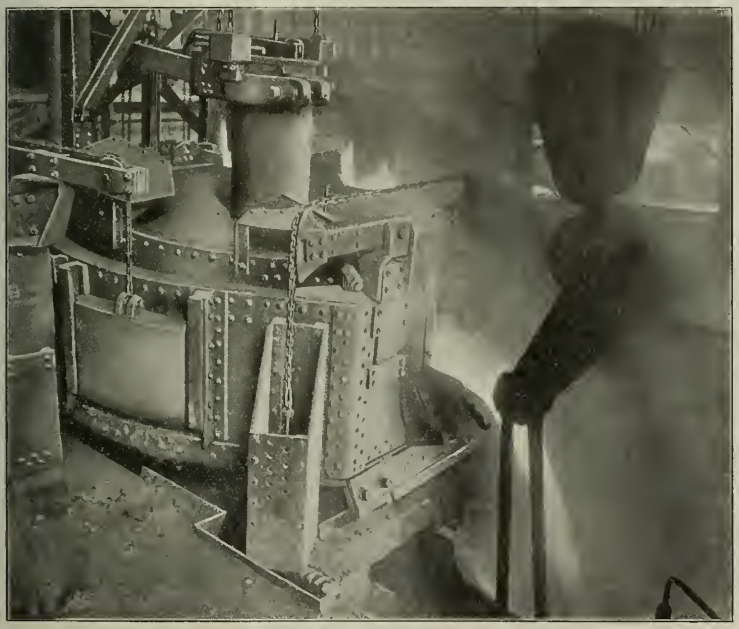
Beginning a little over a century ago with the perfecting of the steam engine and the resultant displacement of hand processes by power-operated machinery, there has come a wonderful creation and development of industries, accompanied by a corresponding revolutionary change in the employment and living conditions of civilized peoples. Also accompanying this evolution there has been a progressive development in the organization of industry, culminating in the "trusts" created within our own time. The institutions and practices which this century of change has produced have existed but a brief interval, historically speaking; they are now on trial, and will be altered or discarded as experience teaches.

Washington, D. C. Ray L. Stinchfield.

The South Chicago Electric Furnace Plant of the United States Steel Corporation.

When in February of last year the United States Steel Corporation adopted the Héroult electric furnace for steel refining, it was decided to erect, as a beginning, two Héroult furnaces, each of 15 tons capacity, one at the South Chicago works of the Illinois Steel Company and the other at the Washburn & Moen plant of the American Steel & Wire Company, at Worcester, Mass. At South Chicago the electric furnace is used for refining molten metal from the Bessemer converter, at Worcester for refining molten metal from the basic open hearth. The object of the combination of the ordinary acid-lined Bessemer converter and the electric furnace (South Chicago) is to produce steel of open-hearth steel quality or better at less cost. The object of the combination of basic open-hearth and electric furnace (Worcester) is to produce steel better than the best acid open-hearth steel in quality, at a slightly advanced cost.

The undertaking of the United States Steel Corporation involved some interesting and important novel points. The largest



HEROULT ELECTRIC STEEL FURNACE AT SOUTH CHICAGO.

Héroult furnaces installed before that time had a capacity of five tons. In the Héroult furnaces of the Steel Corporation this capacity was trebled. Further, the Héroult electric furnace had been used before on a commercial scale with cold charges, and with molten metal from the open-hearth furnace (as in the pioneer American Héroult plant of the Halcomb Steel Company in Syracuse); but in South Chicago it was to be adopted for the first time in America on a large scale in combination with the Bessemer converter. Finally, while in the past the Héroult process had been used essentially as a substitute for the crucible steel process for the manufacture of high-grade tool steels, etc., it entered into a much wider field of usefulness at South Chicago and Worcester, where it is used for large-tonnage products (tonnage steel), like rails, axles and wire.

The South Chicago plant has been in continuous operation since May of last year, while the Worcester plant was started in the beginning of this year. The South Chicago plant is perhaps the more interesting of the two, since it includes the more involved metallurgical process of both dephosphorization and desulphurization of the steel in the electric furnace, while the metal charged from the basic open-hearth into the electric furnace at Worcester is already largely dephosphorized.

Further, the combination of the electric furnace with the Bessemer converter seems a matter of momentous commercial importance. For a number of years the cry of the consumer has been for better steel for many purposes, like rails, etc. As a result, there has been a decided tendency away from the converter and toward the open hearth. Many have prophesied that it would be necessary to use open-hearth steel almost exclusively in future and that the Bessemer converter plants were doomed. If the electric furnace in combination with the Bessemer converter can make as good or better steel at the same or less cost than the open hearth alone, it means that the electric furnace gives a new lease of life to the converter. Far from being a competitor to the old methods of steel metallurgy, the electric furnace becomes the savior of the converter plants and prevents their being scrapped. From this viewpoint the results obtained with electric refining at South Chicago are of broad commercial significance and importance.

By a special invitation extended to us by Dr. Paul Héroult we were enabled to inspect, on March 12, the electric furnace plant of the Illinois Steel Company at South Chicago. The inspection extended almost over the whole day; we were given every possible facility of inspection, while information on all questions asked was given without reserve. Our thanks are due to Dr. Héroult, to Mr. W. A. Field, general superintendent of the Illinois Steel Company, and to the many officials and employees of the Illinois Steel Company who gave us information.

The most important question that will be asked is, of course, whether the Héroult furnace in combination with the Bessemer converter is now beyond the experimental stage. This question must be answered unreservedly in the affirmative. There is nothing experimental whatever about the operation of the Héroult furnace in South Chicago. There is no "fussing" about the furnace. No nervousness is displayed as to whether something might go wrong. The whole operation is carried on with such self-evident certainty and assurance as comes only from firmly established routine.

That the Héroult furnace at South Chicago is a commercial metallurgical apparatus and no longer in the experimental stage is also proved by the regularity with which the heats follow each other. The furnace is in continuous operation during the whole week, and operation is discontinued only on Sunday, the converter plant not working that day. When the converter plant closes down in the night from Saturday to Sunday, the metal from the last converter blow is transferred to the electric furnace and refined; then the electric furnace plant closes down till Monday, when the first metal blown in the converter plant is transferred to the electric furnace.

As an indication of the regularity of operation we give in

Table I the net weight of metal charged, the kw-hours consumed, and the durations of 11 successive heats made on March 10.

TABLE I.

Net weight charged. Lb.	Kw-hours used.	Time.
29,000	2000	1 hour 20 minutes.
28,200	2300	1 " 35 "
28,200	2500	2 " 15 "
28,000	2200	1 " 30 "
26,600	2100	1 " 15 "
26,000	2200	1 " 40 "
25,700	2900	2 " — "
23,000	2500	1 " 35 "
28,000	2400	1 " 35 "
27,000	2300	1 " 35 "
27,400	2700	1 " 45 "

On the average 12 heats are made a day. The figures given under time are the time in hours for which the current was on during each heat. It will be seen that during the 24 hours of this day the current was on for 18 hours and 5 minutes, so that the intervals between the successive heats (during which no electrical energy was consumed) amounted to 5 hours and 55 minutes, or 33 per cent of the whole time.

The average specific energy consumption of the 11 heats of Table I is 88.4 kw-hours per 1000 lb. net weight, or 194.5 kw-hours per net metric ton. In this case the electric furnace treatment, it should be remembered, involved both desulphurization and dephosphorization.

If the charge supplied to the electric furnace is already low in phosphorus, the specific energy consumption is, of course, less. As an example we give in Table II the figures for 9 successive heats made in 12 hours on Jan. 28.

TABLE II.

Net weight Lb.	Kw-hours.	Kw-hours per metric ton.
28,000	1400	110.0
27,600	1700	135.5
27,000	1400	114.1
28,000	1000	78.5
28,000	1300	102.1
28,400	1300	100.8
27,800	1000	79.2
27,600	1500	119.5
28,100	1200	93.9

The average specific energy consumption of the nine heats of Table II is 47.2 kw-hours per 1000 lb. net weight, or 103.8 kw-hours per net metric ton.

As to the electric power supply, it should be stated that the electrical energy is generated by means of gas engines operated with blast furnace gas and that the two plants of the Steel Corporation at South Chicago, Ill., and Gary, Ind., are electrically connected in parallel. The power generation is highly economical. The charge for energy supply, made to the electric furnace plant, is ½ cent per kw-hour actually consumed, as taken from the meters.

As to the reliability with which the desired composition of steel is produced, we give here the specification and analyses of eight successive heats made during part of March 11. In all these cases axle steel was made.

The first line, marked "Specification," gives the desired specified composition of the steel, while the next lines, marked "Analysis," give the analyses of the steel produced in these eight heats.

TABLE III.

		C	S	P	Mn
Specification	Axle steel	0.35 to 0.45	0.03	0.03	0.37
Analysis	[illegible] lbs	45	0.032	0.03[illegible]	0.49
	[illegible],000 "	.37	.028	.033	.42
	[illegible],000 "	38	.028	.026	.39
	[illegible],000 "	.35	.031	.037	.38
	[illegible] "	.41	.030	0.39	.47
	[illegible] "	.48	.025	.032	.41
	[illegible],000 "	.42	.030	.027	.44
	[illegible] "	35	030	.028	.31

It should be said that the above data were not selected by the writer for any particular purpose, but were taken at random from the regular works records.

The regularity of operation which makes the Héroult furnace at South Chicago a thoroughly commercial apparatus is due in no small degree to the simplicity and compactness of construction of the Héroult furnace and the total absence of any complications whatsoever, either electrical or otherwise, with the possible single exception of the electrodes at the top, which will be particularly discussed.

The furnace as used in South Chicago is shown in the adjoining illustrations, Figs. 1 and 2, for which we are indebted to Mr. R. H. Wolff, Dr. Héroult's representative in this country. The Héroult furnace has been so often described in this journal that there is no necessity for any further details at this place. To characterize the design briefly, it is essentially built like the body of an open-hearth furnace, the heat being supplied from the electrodes through the top. There are three electrodes in the South Chicago furnace, since it is operated by three-phase currents.

The simplicity of the construction results in great convenience of repairs of the furnace proper. During the whole period of operation at South Chicago, from May, 1909, until the present, it is stated that there has never been any necessity to stop the Héroult furnace for the sake of repairs.

The lining is kept in good condition by throwing some shovels of broken burnt dolomite into the furnace upon the corroded portions of the hearth. This is done between successive heats and requires not more than a few minutes; no extra help is required. The consumption of dolomite for repairs is given as 10 lb. of dolomite per ton of steel.

The silica roof has been generally repaired or replaced during the regular stop on Sundays. The last roof, however, lasted 129 heats. The replacement of the roof is a very simple matter, a new roof being always kept in reserve.

The banks (side walls) of the present furnace have stood since Nov. 16, 1909, and there is not yet any necessity for reconstructing them.

The cost of repairs is, therefore, very low. It is given as follows: A silica roof costs $60; if it lasts for 129 heats the cost of the roof per ton treated is about 3 cents. The 10 lb. of dolomite ($6 per ton) required for repairing of the lining per ton of steel cost also about 3 cents. Hence, the total cost of repairs of the furnace proper is approximately 6 cents per ton, which is a remarkably low figure.

As to the electrodes, their consumption is given as averaging 6 lb. per ton of steel, and this figure is stated to be true both for graphite and for amorphous carbon. It is the item of the cost of electrodes which, it is expected, can be very materially reduced in future.

The original intention was to use amorphous carbon electrodes. But it was found very difficult to make very large homogeneous carbon electrodes. While with smaller furnaces amorphous carbon electrodes are in satisfactory successful use, the manufacture of such electrodes for the large 15-ton furnace has met with difficulties in baking and it appears that the problems involved have not yet been solved.

In the beginning of operations, when baked amorphous carbon electrodes were used at South Chicago, they frequently cracked at a most inopportune moment, and caused great trouble. Since the quality or strength of the carbon electrodes obtainable did not materially improve, it was necessary to have then recourse to the more expensive graphite electrodes. These have been and are now doing the service very satisfactorily and no such electrode troubles are now experienced. The intention is to try to solve the problems involved in the baking of large durable homogeneous amorphous carbon electrodes, as their use would reduce the cost of the electric refining process considerably. In every respect except higher first cost the graphite electrodes are exceedingly satisfactory.

The electrodes as used at present are made up of Acheson graphite rods, 48 in. long and 8 in. in diameter. Three such rods are butt-connected to a total length of 144 in., and three

such 144-in. rods are arranged side by side to form a single electrode, consisting thus of a solid bundle of three rods, each 144 in. long, and with cross-section a true shamrock. We might say each electrode is made up of three rods in parallel, each rod consisting of three 48-in. rods in series. A water-cooled jacket is provided around each electrode, just above the openings in the roof.

We now give the record of the operation during one complete heat. The temperatures given in this record were measured with a Mesuré-Nouel optical pyrometer by Dr. Jos. W. Richards, who accompanied the editor of this journal in the inspection of the plant

11 a. m.—Finished steel poured from electric furnace into ladle (temperature 1495° C.) and then from the ladle into ingot. The two side doors of the electric furnace are open. Dolomite shoveled into electric furnace to repair lining.

11.10 a. m.—Mill scale charged for next heat.

11:16 a. m.—Charging launder put in place in mouth of electric furnace.

11:18 a. m.—The new charge of Bessemer metal (30 to 40 seconds overblown) arrives in ladle and slag is poured off. Temperature of empty electric furnace 1300° C.

11:21 a. m.—Pouring of Bessemer steel from ladle into the electric furnace. Temperature of Bessemer metal while being poured into electric furnace, 1500° C.

11:24 a. m.—All Bessemer metal poured into electric furnace. Temperature of empty Bessemer ladle, 1500° C. Electric circuit closed. The electrodes are first regulated by hand, later by automatic electromagnetic regulation (Cuenod).

11:28 a. m.—Charging launder removed. Yellow flames seen, probably from the oil and grease in the mill scale.

11:40 to 11:45 a. m.—One hundred and twenty kw-hours consumed in five minutes. Power-factor, 90 per cent.

11:50 a. m.—Twelve shovels of mill scale added.

12 m.—Temperature inside furnace, 1425° C. Temperature of hottest electrode just outside the furnace, 1050° C.

11:45 a. m. to 12 m.—Three hundred and eighty kw-hours consumed in 15 minutes. Power-factor between 82 per cent and 86 per cent.

12:28 p. m. to 12:33 p. m.—About half the slag skimmed from the bath. Temperature of slag, 1400° C.

12:35 p. m.—Added manganese ore.

12:37 p. m.—Added lime.

12:45 p. m.—Half of the "phosphide slag mixture" charged (see patent of Humbert on page 48 of our January issue). This slag mixture had been previously made up of coke dust and fluorspar.

12:54 p. m.—Rest of "phosphide slag mixture" added.

12:58 p. m.—Ferromanganese and ferrosilicon added.

1:01 p. m.—First test sample taken.

1:08 p. m.—Test made shows that heat is ready for tapping.

1:11 to 1:13 p. m.—Furnace emptied. Temperature of empty furnace, 1500° C.; this temperature dropped 200° C. in 15 minutes, while doors were open to patch up the furnace and make ready for the next heat.

There are altogether four attendants at the furnace: one head melter, one assistant and two workmen. A helping hand is lent by other workmen around the works while tapping and pouring.

At the time of our visit, and during the preceding week, axle steel was made. Rail steel has also been made, also high-manganese steel, ordinary steel castings and manganese-steel castings.

Simply to try what can be done with the electric furnace, a very severe test was made some time ago in which, it might be said, the electric furnace was used as a scavenger. As Mr. Field, the general superintendent of the Illinois Steel Company, explained, they swept up the works to find the worst kind of scrap (plate scrap, rusty nails and everything that was found) and tried to make a fine heat out of it. The electric furnace proved equal to the task.

The specification was to produce steel

C, 0.35 to 0.45; S, 0.04; P, 0.04; Mn. 0.35 to 0.50.

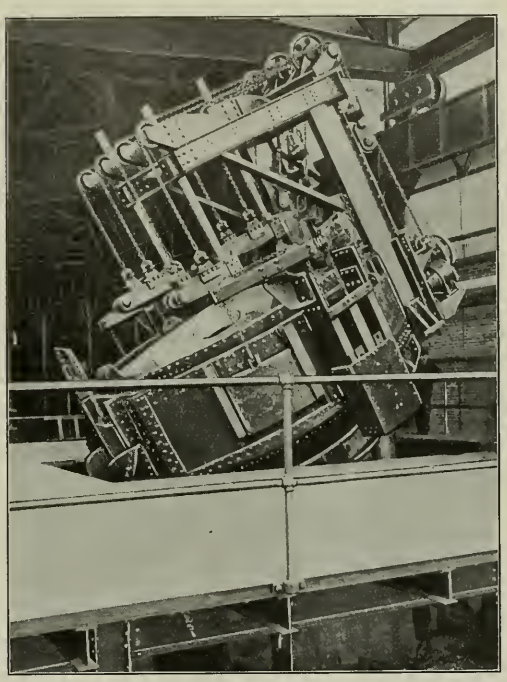
The steel produced analyzed:

C, 0.46; S, 0.037; P, 0.024; Mn, 0.34.

There were charged in this test 9000 lb. hot pig iron, 12,000 rail ends and 9000 stove-plate scrap, "the worst stuff to be found." The product was 31,800 lb. of steel.

The energy consumption during this remarkable test was 14,450 kw-hours. This test was simply made to see what could be achieved with the electric furnace, and proved that it has apparently no ordinary limitations to its refining capabilities.

As to the mechanical tests of the steel now being made by the



HÉROULT FURNACE DIPPED.

Héroult Bessemer-electric process at South Chicago, the following data were given at the mechanical testing plant of the Illinois Steel Company: Elastic limit, 35,000 to 47,000; tensile strength, 60,000 to 70,000; elongation, 25 per cent to 30 per cent; reduction of area, 43 per cent to 60 per cent. As to the difference between this electric steel and acid open-hearth steel it was stated that the same tensile strength is obtained with the electric steel with lower carbon than with open-hearth steel. For example, an 0.10 C electric steel averages the same tensile strength as an 0.15 C to 0.17 C acid open-hearth steel.

As to the general arrangement of the plant it may be said that there was not space available to erect the electric furnace in the immediate neighborhood of the Bessemer plant. The molten Bessemer metal has to be brought from the Bessemer plant to the electric furnace over a distance of a fourth of a mile. Of course, in a new works, built for the proper combination of converter and electric furnace, this would be arranged differently.

In conclusion, we congratulate Dr. Héroult and the United States Steel Corporation on the signal success achieved in their pioneer Bessemer converter-electric furnace combination plant.

which, we have no doubt, is the forerunner of many similar installations, and marks the beginning of a new commercial epoch in practical steel making in America.

The Decomposition of Carbon Monoxide in Presence of Catalytic Agents and in the Blast Furnace.*

By Fritz Zimmermann, Ph.D.

At the last meeting of the American Electrochemical Society in New York, Mr. Edward R. Taylor presented a very interesting paper on the electric smelting of iron ore. In the discussion following this paper, it was pointed out that an attempt should be made to construct electric furnaces in such a way that the escaping gases should consist of practically nothing but N and CO_2 and that, if this would be possible, the energy efficiency, of course, would be very much greater than if they contained a large percentage of CO.

Mr. Lidbury pointed out that there is a ratio between the CO and the CO_2 which is governed by the conditions prevailing—that is, the temperature and the nature of the substances present, which fixes a limit beyond which no further reduction will take place at the expense of the CO.

It might be of interest to give a short compilation of some studies and experiments that have been undertaken by others and myself to study some of the phenomena which are taking place in the blast furnace.

The production of iron in the blast furnace was, until almost the middle of the nineteenth century, purely empirical. Bunsen undertook the first steps to put it on a scientific basis in order to obtain the best possible results. His work on gas analysis enabled him to investigate the composition of the blast furnace gases in different parts of the blast furnace. He found that the carbon dioxide escaping from the furnace always contained more than the double amount of carbon monoxide, naturally causing a big loss of fuel. He was the first one to say that "we cannot really speak of utilizing our fuel properly until someone is able to construct a furnace from which only carbon dioxide will escape besides the nitrogen of the blown-in air."

Assuming that the large amount of carbon monoxide, which usually escapes from the ordinary low furnace, could be forced to reduce more iron ore, Lowthian Bell, the celebrated English metallurgist, began about 1860 to construct furnaces of a very great height. His reason for constructing these furnaces was that the carbon monoxide, instead of passing out into the air, as it always does in low furnaces, should meet more iron oxide, which it could reduce, so that all the carbon monoxide would be utilized. His experiments were a failure, and he was unable to explain why the proportion of CO to CO_2 remained the same as in the old low furnaces.

Le Chatelier, in his "Recherches Experimentales et Theoriques sur les Equilibres Chimiques" (1888), found an explanation. He came to the conclusion that an equilibrium is established between iron, iron oxide, CO and CO_2 which does not depend much upon the temperature. Gas, containing less CO than necessary for this equilibrium, is not able to act reducing on iron oxide; if more iron be present than corresponds to the equilibrium the reversibility of this process will show itself by partly oxidizing the iron until there is again equilibrium.

But the process of reduction is not the only one taking place in the blast furnace. Sometimes we observe a strong precipitation of finely divided carbon in a certain zone of the furnace. This may be so considerable as to clog up the furnace and stop the running of it. For a long time this precipitation of carbon could not be explained until something similar was found in the manufacture of steel.

In continuing certain work done by Berzelius and Saunderson, Caron found that, in passing CO over iron, finely divided carbon was precipitated. He thought it was silicon that caused this precipitation. In his work "Sur la Dissociation de l'Oxyde de Carbone," Saint Claire Deville used a very ingenious apparatus—his cold and warm tube. This consists of a porcelain tube, closed with two double perforated stoppers; through one set of holes a brass tube is placed, which serves for passing cold water through it; through the others, gas may pass. Heating this tube to a strong red heat, he passed pure CO through it, while cold water was flowing through the brass tube; under these conditions, CO was decomposed into CO_2 and carbon. He explains it as a partial decomposition of CO while hot into oxygen and carbon. He assumes that, at the moment the products of dissociation touch the cold tube, they are taken

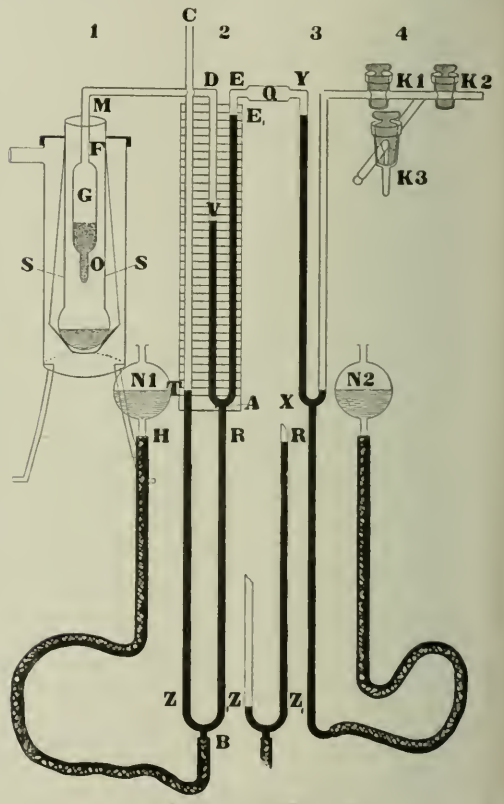


FIG. 1.—ARRANGEMENT OF TEST.

away from the influence of the heat, which otherwise would unite them again. Carbon is precipitated on the cold brass tube, and the free oxygen combines with free CO, forming CO_2.

Stammer got similar results by passing CO over iron oxide. He found that, while the oxide was reduced to metal, iron carbide was formed at the same time, and a precipitate of carbon took place upon the iron.

Gruner, and later Berthelot, worked on similar subjects—the first saying that only the presence of oxide caused a decomposition of CO and that pure iron could not decompose CO, while the latter assumed that a complicated polymerization took place.

Boudouard undertook a series of experiments to study the decomposition of CO in the presence of metallic oxides which, as he assumed, would act as catalytic agents. He used the oxides of nickel, cobalt and iron in the presence of CO in a sealed tube, and analyzed the products of reaction. He found that the speed of reaction and the equilibrium depended upon

*A paper read before the New York section of the American Electrochemical Society on March 16, 1910.

the temperature. At a temperature of about 445 deg. C., in the presence of the metal oxides as catalysers, decomposition should be complete according to the equation of $2\,CO = CO_2 + C$, while, by increasing the temperature, the reverse reaction $C + CO_2 = 2\,CO$ takes place more and more.

To get a better knowledge of these cross reactions, Professor Schenck, then in Marburg and now in Aachen, induced me to make a special study of the character of these reactions. The reaction $2\,CO = CO_2 + C$ is effected with a contraction in volume of 50 per cent, for we start with two molecules CO and come to only one molecule of CO_2, C being precipitated.

By constructing a proper apparatus, this contraction will allow us to follow the speed of reaction.

After building a great many apparatus, I finally came to the following construction, which seemed to answer my purpose in every respect. The apparatus consists of four main parts.

I.—*G* is a vessel of Jena glass of about 100 c.c. capacity, intended to be filled with a gas and whatever catalyser shall be used; from *F* on the right side of the apparatus it consists of capillary tubing, so that its capacity may be neglected compared with the 100 c.c. of the heating vessel.

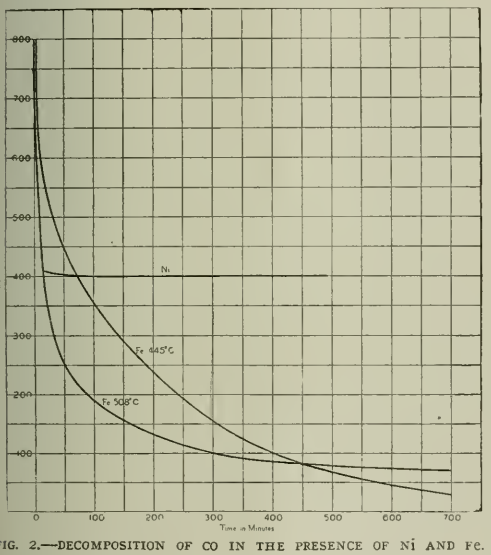


FIG. 2.—DECOMPOSITION OF CO IN THE PRESENCE OF Ni AND Fe.

Before being used *G* is cleaned well with sodium hydrate and afterwards with chromic and sulphuric acid. Then it is filled with a catalyser through the wide tube *O*, which afterwards is closed by melting it together in a gas flame; then the capillary tube is molten on the apparatus at the point *M*. In order to heat *G* uniformly, it is put in a little furnace consisting of a wide tube *S* of Jena glass standing on an iron bath, protected by asbestos and sheet iron. In *S*, different substances, like diphenylamin, mercury, sulphur or sulphide of phosphorus may be boiled.

II.—Consists of two barometers, one put on top of the other; the tube *AB* is 800 millimeters long, *AE* and *AD* are of the same length; *BC* is a little above 1600 mm; BH_1 is heavy rubber hose, around which heavy linen is sewed in order to prevent it from expanding by the weight of the mercury in the vessel N_1. If N_1 is lowered to Z, the mercury will stand equally high in both tubes *BA* and *BC*. By evacuating the apparatus, the mercury in *BA* will rise to *R* and the height of mercury Z_1R or the distance between *Z* and *R* being the atmospheric pressure. Of course, the mercury cannot rise higher than *R*. If gas is allowed to flow into the apparatus, it will fill *G* and the mercury in *AB* will fall according to the pressure of the gas. If it is less than an atmosphere, N_1 does not need to be changed from its position; if it is more than an atmosphere, N_1 must be lifted up. The distance between *R* and *Z* is plus or minus according to the pressure of the gas. After gas has passed into *G*, N_1 is lifted to *T*, sealing off the gas entirely. If we evacuate now, the mercury in the tube on the right *AE* will rise to *E;* the distance between E_1 and *T* being the atmospheric pressure. The distance between E_1 and *V* indicates the absolute pressure of the gas in *G;* so that the pressure in *G* can be measured either by the scale distance between E_1 and *V* or, what is the same, the distance between *T* plus atmospheric pressure and *V*.

Q is a vessel to collect drops of mercury that might fly out of *AE* or *XY*.

III is another double barometer, the only difference between it and II being that it has no connection with the outside air as *BC*.

IV.—K_1 and K_2 are ordinary gas cocks; K_3 is a three-way cock leading to vacuum pump, gas developer or gasometer or hood respectively.

In order to get the finely divided catalyser, pumice stone cut to the size of a pea was used. As acid alone did not remove all the iron oxide in the pumice stone, it was reduced first with hydrogen; after this, it was easy to remove all the iron and get an absolutely white product.

In the following experiment, cleaned pumice stone was impregnated with a solution of pure nitrate of nickel—2 grams of nickel to 20 grams of pumice stone being used; the solution was evaporated together with the pumice stone, and afterward the impregnated pumice stone was highly heated on a blast burner until the nitrate was decomposed and only nickel oxide remained. After the apparatus had been thoroughly cleaned and dried, the pumice stone impregnated with nickel oxide was placed in vessel *G*, which was then evacuated and heated by immersion in boiling sulphur. CO was then passed into the apparatus and the vessel N_1 raised to *T*. No contraction could be seen.

It might have been that the reaction and contraction had been completed in the first 30 or 40 seconds which were required to flow in the gas and raise the mercury vessel, so that this contraction could not be seen. In order to be sure whether this was the case or not, the apparatus was filled, while cold, with CO, heated to 445 deg. C. and let cool again. The volume was equal to its original. The same result was obtained in heating to 320 deg. C., 360 deg. C. and 508 deg. C. The analysis of the gas contents showed CO_2 only. This proves that the reaction taking place is only a reducing reaction $CO + NiO = CO_2 + Ni$, which, of course, does not show any contraction.

I also observed no contraction after using cobalt oxide or iron oxide. The only reaction taking place in each case was a partial reduction of the metallic oxide. This proves that the oxides of nickel, cobalt and iron are no catalytic agents able to bring about the decomposition of CO to $CO_2 + C$, as had been claimed.

In the next experiments, pure metals were used instead of the metallic oxides; the metals being obtained in a finely divided state by reducing the metallic oxides precipitated on the pumice stone. Nickel was used first; after using every possible precaution, CO was allowed to pass into the vessel at 445 deg. C. Reaction took place immediately, causing strong contraction, which came to a standstill after heating for about an hour. A contraction of 80 per cent of the pressure, observed in the beginning, had taken place.

The experiment was repeated several times, practically with the same result.

At 360 deg C. contraction, which could be measured, was slower than at 445 deg. C. and amounted to 91 per cent.

At 310 deg. C. the temperature of boiling diphenylamin, the reaction was slower and came to standstill after seven hours; the decomposition being 96%. Cobalt gave similar results.

In order to see whether the equation $2CO = CO_2 + C$ and with it a contraction of 50 per cent takes place, cobalt as catalyser was used. The vessel was filled, while cold, with CO, then heated and afterward allowed to cool in cold water to the same temperature as before. The contraction was exactly 50 per cent, showing that CO really had been decomposed into $CO_2 + C$. The gas analysis showed practically nothing but CO_2.

With nickel, the same experiment could not be made, on account of nickel tetracarbonyl being formed. In order to see whether the presence of CO_2 would change the reaction, a mixture of CO and CO_2 was passed into the apparatus. The contraction was 50 per cent of the CO contents, so that the CO_2 did not seem to have any influence on the reaction.

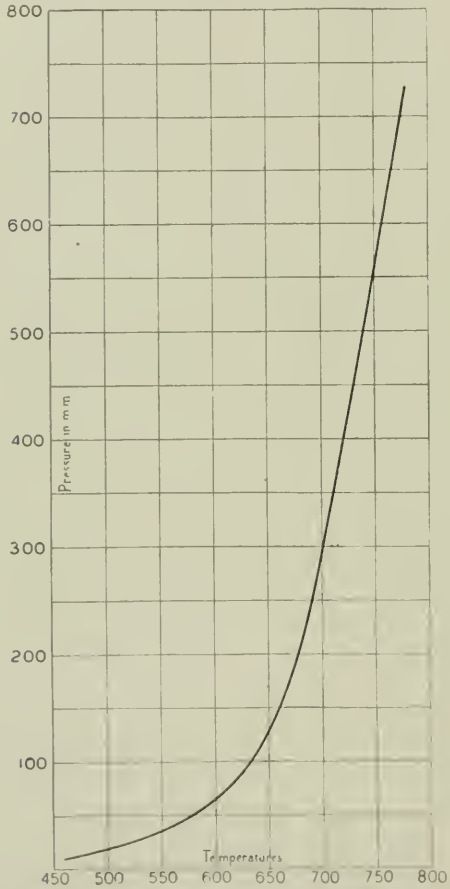


FIG. 3. PRESSURE CURVE CO + CO_2 AT DIFFERENT TEMP. IN THE PRESENCE OF Fe ACCORDING TO SCHENCK.

In the following experiments, iron as a catalyser was used. At 360 deg. C. the reaction was very similar to that of nickel and cobalt, but at 445 deg. C. a very curious phenomenon took place. The decomposition did not only go to 50 per cent, but the contraction continued and came almost to a vacuum. For instance, starting with 787 mm pressure, after heating nine hours, we came to 32 mm. In another series of experiments at 508 deg. C., very rapid contraction took place, but the final pressure was a little higher than at 445 deg.

The explanation of this phenomenon with iron is as follows: While the decomposition, in the presence of nickel, cobalt, and iron at low temperatures, occurs according to the equation $2CO = CO_2 + C$, the iron, at temperatures of 445 deg. C. and 508 deg. C. enters itself into the reaction. It is oxidized and reduces the CO_2 according to the equation $Fe + CO_2 = FeO + CO$. The CO, formed in this way in the presence of fresh metallic iron, is decomposed again. These reactions continue until nearly all the gas has disappeared. The CO which was present at the beginning is then almost all decomposed, its carbon precipitated, and its oxygen bound to iron. This proves that carbon monoxide under these conditions being converted into CO_2 is really an oxidizing agent for iron.

The dual influence of the iron, viz.: that it first acts as a catalyser to bring about the reaction $2CO = CO_2 + C$, and then secondarily enters itself into the reaction as soon as the CO_2 concentration is high enough, may have a great influence on the efficiency of an iron-reducing furnace. As soon as the CO_2 concentration exceeds a certain limit, the gas will not only be unable to reduce FeO to metal, but will even reoxidize iron which may have already been reduced, and, in this way, bring about the contrary of what is intended, viz.: oxidizing iron to oxide instead of reducing oxide to metal.

I might add that, in order to see to which point CO can act oxidizing, Professor Schenck and Dr. Heller undertook a great number of determinations. They used two methods:

First.—They allowed CO, at constant temperatures, to act on finely divided iron and determined the pressure at which the reaction came to a finish.

Their second method was to mix ferrous oxide with carbon and to heat the same to constant temperatures. The reduction took place with development of CO and CO_2. The development of gas stops after the pressure of the total equation is reached. For both methods, they used a quartz tube, which was connected with a glass manometer

The results of both methods were the same, and they were able to prove that CO is only able to act oxidizing up to a temperature of 780 deg. C. Above this temperature, CO can only act reducing.

By these experiments, they proved that metallic iron only enters into the equation of CO, CO_2 and C below 780 deg. C., while above this temperature it does not react any more with CO.

This reaction, $2CO = CO_2 + C$, in the presence of metallic iron at low temperatures, also explains the clogging up of the furnaces in which CO is decomposed in the presence of metallic iron. This will, for instance, take place if a furnace, in which iron has been reduced at higher temperatures, suddenly cools off and CO comes in contact with this metallic iron.

This investigation, of course, is by no means complete as to all the reactions that are taking place in the blast furnace. These experiments were simply undertaken to study the decomposition of the carbon monoxide in the presence of metals. The other reactions that are taking place in the blast furnace, the equation between carbon, carbon monoxide and carbon dioxide at different temperatures and the action of these with different iron oxides have been investigated very completely, both theoretically and practically by Boudouard, Schenck, Baur, and Glaessner.

Indigo White.—Electrolysis is used in a French process of Henri Chaumat, of Paris, to reduce indigotine to indigo white. The process is carried out in a bath of alkali carbonate protected from the air. The anode is carbon and the cathode indigotine. The latter is in form of a powder, which may be agglomerated with powdered graphite or other good conductor of electricity. Under the influence of the electric current hydrogen is formed at the cathode reducing the indigotine and producing indigo white which dissolves in the alkali simultaneously produced, forming a dyeing solution.

Production of Aluminium as a Laboratory Experiment.

By B. Neumann and H. Olsen.

It is peculiar that the methods of operation used in an industry which produces almost 2000 tons of metal per year are so little known that the production of this metal on a laboratory scale meets with difficulties. The two articles published in this journal during the past year on the production of aluminium in the laboratory confirm this remark.

The first detailed laboratory experiments have been published by Haber and Geipert,[1] who used a carbon crucible and employed a current of about 400 amp. M. deKay Thompson[2] could not, however, obtain satisfactory results when employing the method described by the above authors. He used a cell of graphite plates, and a subdivided anode and employed currents up to 1000 amp. The arrangement of Tucker,[3] who used a conical graphite crucible, can hardly be considered an advance over former apparatus.

We have also tried to make aluminium on a laboratory scale and have endeavored to imitate as much as possible the conditions of large-scale practice. The results of our experiments are given in the following article.

Process Used in Practice.

A photograph of an aluminium cell has been published only once, as far as we know, by J. W. Richards[4]; the cell shown is evidently a cell of Charles M. Hall, as used some years ago. The illustration shows 20 round anodes in two parallel rows in a high iron box. Quite detailed notes illustrated by sectional diagrams were given later by Winteler.[5] The cells described by Winteler are also rectangular and are also provided with double rows of carbon electrodes. More recently circular cells have been introduced, the anodes being distributed over the whole surface.

The aluminium cells are simple reinforced wrought-iron boxes on the bottom of which there is a series of carbon plates. The bottom plate acts as cathode at the start of electrolysis; the aluminium which is set free collects on the carbon plate and then acts itself as cathode. The reason of covering the bottom with carbon is that at higher temperatures aluminium alloys with iron; further, in the beginning, when the mixture of cryolite and alumina is molten by means of a carbon resistor, the local heat effect may easily be excessive, so that the iron bottom would be burned through.

The cell is made so wide that the walls do not come into direct contact with the molten bath, but are completely covered with a layer of solidified charge. The size of the cell depends on the current. For a current of 8000 amp the cells have a length of 1500 mm (4 ft. 11 in.), a width of 1000 mm (3 ft. 3 in.) and a height of 400 mm to 500 mm (1 ft. 4 in. to 1 ft. 8 in).

In each cell there are 6 to 12 carbon anodes, which are suspended from insulated supports.

Formerly carbon blocks of about 100 sq. cm (15.5 sq. in.) cross-section and 1 m (3.28 ft.) length were used and current densities of 3 amp to 5 amp per square centimeter. The distance of the anodes from the walls was 15 cm (6 in.). The distance of the different anodes from each other was 10 cm (4 in.). If the distance is less the anodes are liable to rapid destruction and carbon particles fall into the molten bath and foul it.

This disadvantage is now avoided by using very short electrodes with a much larger cross-section. Such electrodes are 40 cm (16 in.) high and have a cross-section of 35 cm x 35 cm (14 in. x 14 in.). The current density used is also lower, namely, 0.8 amp to 1 amp per square centimeter. Overheating of the bath is thereby avoided, but the bath remains quite fluid.

With the new short electrodes a relatively smaller voltage is lost in the electrodes. A disadvantage is the greater amount of anode scrap, but this is practically of no importance, since it is worked over again.

The method of supporting the cells is of great importance. Formerly the cells were placed free on insulated supports, the intention being to cool the walls and the bottom by contact with the air. An adjustable current of air was passed below the bottom. This method, of course, involved a great loss of heat and this had to be made up by the use of higher current densities. In modern practice the cells are not free, but enclosed, although some free space is left to permit the passage of an adjustable air current for cooling the bottom and sides.

The most suitable composition of the charge is a mixture of 80 per cent to 90 per cent of cryolite and 10 per cent to 20 per cent of alumina. Formerly some per cent of sodium chloride were added to reduce the melting point. The purity of the raw materials used is of great importance; any iron or silicon present are especially deleterious since they pass into the final product and deteriorate the physical and electrical properties of the aluminium. It is well known that it is impractical to refine aluminium afterward.

The voltage of a cell is about 6 volts, but depends considerably on the distance between anode and cathode. A low voltage is an indication of a correct composition of the charge. If there is not sufficient alumina in the charge the voltage rises.

The mixture is fused by inserting between the carbon bottom and each anode a little rod of carbon and heating it to white incandescence. When the charge is molten the carbon resistors are removed and the anodes are lowered into the bath. The temperature of the bath should be such that the color is about dark cherry red. On the surfaces of the bath crusts are constantly formed, which must be broken from time to time.

Former Experiments.

Haber and *Geipert* use a hollow carbon block as cathode and furnace, a movable carbon anode being suspended from above. The inside diameter of the circular hole was 11.3 cm at the bottom and 13.8 cm at the top; the depth was 7 cm. The diameter of the electrodes was 6.6 cm, the cross-section, therefore, 34.2 sq. cm. The use of a turned carbon block is not very advantageous for this purpose, because cracking of the block cannot be avoided. In two experiments artificial cryolite was used; in the third experiment natural cryolite. The charge consisted of 1000 grams of cryolite and 200 to 300 grams of alumina.

Haber and Geipert fused the mass by means of an arc and then started the electrolysis with 7 volts and 400 amp. The use of the arc for melting the charge is unsuitable because quite a quantity of cryolite and aluminium fluoride evaporate and because the arc causes the formation of aluminium carbide.

The experimenters did not pay special attention to the anodic current density, although this is of greater importance than the cathodic current density. Since the average current was 310 amp to 336 amp, the anodic current density was about 10 amp per square centimeter. That is much too high, and this is the reason why the anode was always consumed so as to form a sharp point.

The statement that the color was bright red shows that the temperature was too high. This is easily understood if 300 amp and 8 volts were used for 1200 grams to 1300 grams of charge. The amp-hour efficiency was 54.3 per cent. 44.84 per cent and 43.56 per cent.

M. deKay Thompson tried to reproduce exactly the experiments of Haber and Geipert, but the amp-hour efficiency obtained by him was only 38 per cent; the temperature of the bath was found to be 1100° C.; the voltage 10.5 to 12.5. The charge in the crucible soon became darkened and contained graphite. Then the anode effect occurred.

Another experiment was made in a box of graphite plates

[1] *Zeit. f. Electrochemie*, 1902, pages 1 and 26.
[2] *Electrochemical and Metallurgical Industry*, 1909, page 19.
[3] *Electrochemical and Metallurgical Industry*, 1909, page 315.
[4] *Electrochemical Industry*, 1903, Vol. 1, page 160.
[5] *Aluminium Industrie*, 1903 (published by Vieweg & Sohn, Braunschweig).

1.9 cm thick; the area of the bottom was 17.8 cm x 17.8 cm; the anode consisted of four graphite plates in the form of a grid 12.5 cm in height, breadth and length. The current used was, first, 500 amp, and later 900 amp to 1000 amp. The temperature of the bath was approximately 1000° C. The amp-hour efficiency was 49 per cent. The voltage was between 9 and 10.

Thompson was the first who mentioned the phenomenon of the anode effect, which occurred always after working for one-quarter of an hour or for half an hour, even when he repeated Haber's experiments. The current density used by him with the divided anode was much too high, namely, about 10 amp per square centimeter.

Tucker used a conical crucible turned from a 12.5-cm diameter graphite rod. The diameter of the conical hole was 12 cm at the top and 3 cm at the bottom and a graphite electrode, with a diameter of 3.15 cm, was the anode. He also fused the charge by means of the arc. He employed a mixture of cryolite with 15 per cent alumina, and used a current of 325 amp at 10 volts to 12 volts. The anodic current density was 28

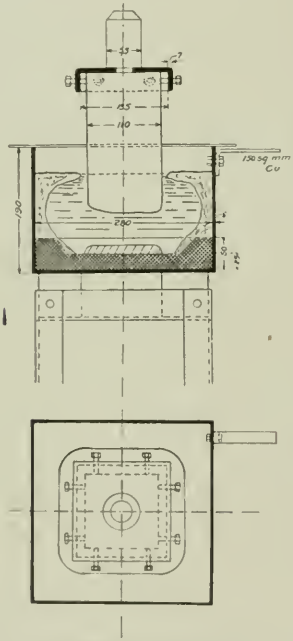


FIG. 1.—FURNACE CONSTRUCTION.

amp per square centimeter. The anode was quickly destroyed. This method seems unsuitable.

Experiments of the Authors.

1. **Furnace Construction, Electrodes.**—In order to imitate as much as possible large-scale practice, we used similar furnaces. All our furnaces were made of wrought-iron sheets and had a bottom plate of carbon. A copper bar was screwed to the iron box to make the connection to the circuit. The bottom carbon plate was firmly pressed against the bottom by means of screws. The various furnace designs employed differed only by form and size.

We used in our experiments a direct-current machine which gave 300 amp at 14 volts. But it could be easily so connected as to give 150 amp at 28 volts. This was quite an advantage for our experiments. The furnace dimensions were so chosen that the whole current of the machine could pass directly through the cell without use of series resistances. The most suitable design for this purpose was a simple square box of 5-mm wrought-iron sheets, which were welded at the edges, the bottom surface being 28 cm x 28 cm and the height 19 cm. An Acheson graphite plate was fixed to the bottom. Its thickness was increased later to 2.5 cm. This was found to be the best thickness.

The anodes were carbon blocks of smaller or greater size up to 14 cm x 14 cm cross-section. As shown in the illustrations, they are held by a wrought-iron head with the aid of copper sheets by means of eight screws.

The furnace box was placed on an iron frame and could be cooled at the bottom and at the sides by means of air or when necessary could be heated from the outside by means of burners. In the first experiments the fusion of the large quantity of powdered substance was quite difficult, so that the possibility of heating from the outside was at times quite an advantage. After we had gained some experience, it was no longer necessary to heat from the outside.

In order to melt the charge a hard carbon resistor of 6-cm length and 1.6-cm diameter was inserted between anode and bottom plate and heated by the current to white incandescence. In this way the mixed charge surrounding the carbon resistor was slowly fused. In the beginning the melting of the charge was sometimes not realized, because the anodes, which consisted of blocks of graphite, conducted too much heat away. This difficulty disappeared when we used hard-carbon blocks, which were furnished to us from the Plania Werke, in Ratibor. The graphite anodes are also easily destroyed during electrolysis, and the particles which dropped off spoiled the bath. It might probably be advantageous to make the bottom plate also of hard carbon instead of graphite.

We have also used smaller furnaces of 20 cm. diameter and less. The melting of the mass was quite successfully accomplished. But the small furnaces have the disadvantage that the charge is fused right to the walls of the iron box; during electrolysis as soon as the resistance near the bottom increases on account of undissolved alumina the current no longer passes to the bottom, but to the side walls.

2. **Observations During Operation.**—When melting a mixture of cryolite and alumina, care is to be taken to have the carbon always covered by the mixture, since otherwise the carbon would burn through at the place where it is uncovered. A charge which has already once been fused is more easily melted than a fresh powdered mixture. A charge of 12 kg requires about 3½ hours for melting. In this case a layer about 2 cm. thick around the walls remains unmolten.

In order to start electrolysis the anode is quickly raised, the carbon resistor is removed from the bath by means of an iron tongue and the anode is then suspended again into the bath. If the anode has the correct temperature, electrolysis starts at once. Graphite electrodes which are good conductors are cooled so easily that when the electrode is raised out of the bath it becomes covered with a thin layer of solidified charge and when it is again suspended in the bath only a small amount of current passes through the cell; the whole bath may then freeze. With hard carbons the danger is less.

When in our experiments the anode had accidentally cooled off too much (while taking it out of the bath in order to remove the carbon resistor) and when it was observed that the current decreased after the anode had again been suspended in the bath, we changed the connections of our machine, so as to run at the double voltage (28 volts); this overcame the trouble. The connections of the machine were then again changed and the electrolytic process took place absolutely quietly.

We have observed several times that in melting a fresh charge the carbon resistor was surrounded by a tube of a mixture rich in alumina which would not melt. The cause is probably to be found in the moisture in the charge. When thoroughly preheating the alumina the phenomenon did not occur.

The electrolysis was always very quiet in the large furnaces

The distance of the anode from the bottom plate was in the beginning 6 cm; later somewhat less on account of the reduction in volume of the charge. The anodes were regulated about every 30 minutes. During electrolysis (as is also the case in practice) a hard crust formed on the bath, which had to be broken for regulating the electrodes or for making additions to the charge.

With this relatively small cell the uniform solution of the alumina in the bath offered sometimes difficulties. Part of the charge added later often directly forms crusts and, therefore, does not get into the bath at all, so that the level of the bath falls. Sometimes the added charge also falls to the bottom of the bath without dissolving quickly enough, so that the alumina accumulates near the bottom and thickens the bath and increases the resistance. The amount of the addition of alumina was calculated on the basis of an average amp-hour efficiency of 50 per cent. One hundred amp-hours produce theoretically 33.7 grams of aluminium, so that with 50 per cent amp-hour efficiency only 16.85 grams are obtained, corresponding to the consumption of 31.8 grams of alumina. But as hardly all the alumina is dissolved, we used practically 40 grams of alumina per 100 amp-hours in mixture with the same amount of cryolite. When the electrolysis proceeded properly, the calculated amount of this mixture was added every thirty minutes. If there was trouble, either alumina alone or cryolite alone was added, according to the circumstances. An addition of aluminium fluoride was mostly unnecessary.

Whether the composition of the bath is correct, is well indicated by observing the temperature of the bath and the voltage. When the electrolysis has been going on for some time the aluminium deposited acts as cathode. In the beginning of our experiments we have melted several times a certain amount of aluminium on the supposition that this would facilitate the first deposition of aluminium, but we could not find any considerable advantage.

If some undissolved alumina deposits once on the bottom, all kinds of disturbances are experienced. The heavy layer presses the aluminium upwards, and the contact between the aluminium and the bottom plate gets poorer. Or the metal no longer covers the whole surface, but only a part, and therefore grows toward the anode, whereby an overheating effect is produced. By stirring the bath and ladling out the aluminium the difficulties can be overcome.

The occurrence of the anode effect mentioned by M. deKay Thompson has been observed by us also occasionally. The anode effect is due to the fact that the anode is no longer wetted by the bath. It seems that the anode is surrounded by a gas layer, small arcs passing over between anode and bath. On account of this transition resistance the voltage increases by several volts. Fluorine is then set free at the anode.

According to our observations the anode effect occurs if the anodic current density is too high, say, more than 4 amp per square centimeter. This phenomenon also occurs when large anodes are irregularly consumed. The composition of the charge, however, also plays a part. The less the content of fluorine in the bath the more often does the anode effect occur. By quickly opening and closing the circuit, this trouble can be overcome.

Occasionally an ill-smelling gray layer of molten metal is found on the bottom; in contact with water this metal evolves a gas which consists mainly of hydrogen. We could not prove the presence of aluminium carbide. The gray color is caused by a metallic mist of very finely divided aluminium.

In the following we give the details of two different experiments; in one case trouble was experienced, while the other experiment was absolutely quiet.

A.

12 kgs. of charge, consisting of 10 per cent alumina and 90 per cent cryolite; cross-section of anode 80 sq. cm.

Melting, 3 hours 50 minutes.				
8.45 } 12.35 }	300	amperes	7	volts
Electrolysis				
12.40	150	"	13.5	" 900° C.
12.45	200	"	10.4	"
1.15	200	"	10.0	"
1.45	185	"	10.5	"
1.55	200	"	10.0	"
2.00	215	"	9.6	"
2.05	225	"	9.2	" 950° C.
2.20	250	"	8.8	"
2.35	260	"	8.6	"
2.45	270	"	8.4	"
3.00	280	"	8.1	"
3.20	240	"	9.0	"
3.35	270	"	8.3	"
3.40	280	"	8.0	"
3.45 to 58	Anode effect remedied.			
4.00	230	amperes	9.0	"
4.10	200	"	9.8	"
4.20	180	"	10.6	"

Eighty grams charge were added at 1.45, 160 grams charge at 2.05, 200 grams charge at 3.00, 30 grams alumina were added at 3.20 and 100 grams cryolite at 3.40.

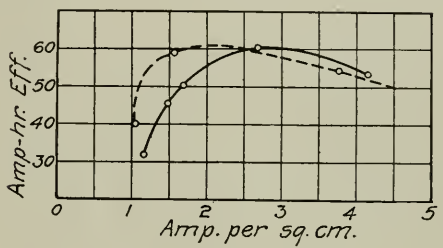


FIG. 2.—EFFICIENCY CURVES.

The experiment was then stopped on account of the anode effect. The anode was finally so much corroded that the anodic current density had increased to an amount which produced the anode effect.

The time of the electrolysis was three hours and forty minutes, the average current 214.5 amp, so that 778.5 amp-hours had been consumed, which should have yielded theoretically 265 grams of aluminium. The output was in reality 161 grams of aluminium, so that the amp-hour efficiency was 60.8 per cent. The current density was 2.68 amp per square centimeter.

B.

12 kg. of charge, consisting of 10 per cent of alumina, 10 per cent of sodium chloride, and 80 per cent of cryolite.

Anode surface 11 by 11 cm = 121 sq. cm.

Melting, 3 hours 40 minutes.				
11.00 } 2.40 }	300	amperes	7	volts
Electrolysis	60	grams aluminium added.		
2.40	200	amperes	14.4	"
3.00	200	"	9.0	"
3.30	200	"	9.0	"
4.00	200	"	9.0	"
4.30	200	"	9.0	"
4.50	200	"	9.0	"
5.10	160	"	9.2	"
5.30	160	"	9.2	"
5.45	160	"	9.2	"
6.10	150	"	9.4	"
6.30	150	"	9.4	"
7.00	150	"	9.4	"

During this experiment 100 grams of the mixture of alumina and cryolite were added every 30 minutes. The experiment was interrupted because the current from the central station was disconnected. Seven hundred and eighty-five amp-hours had been consumed. The theoretical output should have been 266 grams, while 156 grams of aluminium were obtained in fact. The amp-hour efficiency was therefore 58.8 per cent. In this experiment 14.6 per cent of sodium chloride evaporated.

In another experiment the loss of fluorine was determined. There were in the beginning 50.26 per cent of fluorine in the bath, while after one hour's electrolysis the bath contained only 47.47 per cent, so that the loss was 2.79 per cent.

3. **Results of Experiments.**—In the following we give the results of a number of tests made with different anodic current densities:

Charge.	Anode surface.	Ampere hours.	Time of electrolysis.	Current density in amperes per sq. cm.	Ampere hour efficiency.
10% alumina and 90% cryolite	169 cm.	750	3 hrs. 50 min.	1.17	31.8%
	144	690	3 " 15 "	1.38	45.5
	121	875	4 " 15 "	1.70	49.8
	81	545	3 " 10 "	2.15	49.6
	80	778	3 " 40 "	2.68	60.8
	50.3	1010	4 " 50 "	4.15	53.0
10% alumina, 80% cryolite, 10% NaCl	169	685	3 " 45 "	1.08	40.0
	121	785	4 " 10 "	1.57	58.8
	50.3	336	2 " 5 "	3.12	53.9
	50.3	373	2 " — "	3.72	54.2

If these results are plotted graphically it will be seen that the amp-hour efficiency is a maximum for an anodic current density of 2 amp per square centimeter. On the other hand, the results are very poor with a current density of 1 amp per square centimeter. But this does not agree with the results from operation on a large scale. It should not be forgotten, however, that in experiments with electric furnaces the conditions in the laboratory furnace are very different from the conditions in the large commercial furnace, and that in the latter the losses from conduction and radiation can be made very much smaller. An anode which is too large, such as is used in our experiments, with a current density of 1 amp per square centimeter, carries off so much heat that the bath cannot be maintained at the proper temperature. The above results are, therefore valid only for our design of furnace and are not valid in general. The best amp-hour efficiency which we obtained was 70.4 per cent with a current density of 2 amp per square centimeter.

From the diagram of Fig. 2, it is seen that an addition of 10 per cent of sodium chloride exerts a beneficial influence only with small current densities, while with higher current densities the bath which contains no sodium chloride gives better results.

Some experience is required to carry out successfully the experiments. In the beginning we experienced difficulties especially in melting the charge. In such cases it is to be recommended to cover the side walls of the iron box, and perhaps also the upper portion of the anode, with asbestos board to reduce the radiation loss and to heat the bottom of the box with gas burners. Much time is saved in this way. It is also to be recommended not to select at once too big an anode for the first experiments.

Elektrochemisches Institut, Technische Hochschule, Darmstadt, Germany.

Chilling or Heating Action of Furnace Electrodes Versus Least Electrode Loss.

By Carl Hering.

In determining the proper proportions of electrodes for furnaces, the choice may sometimes arise between allowing the loss in the electrode to be slightly greater than the least possible, or allowing the electrode to slightly chill or heat the product in the furnace; either may be adopted at a sacrifice of the other. The difference is due to the variations of the conductivities with the temperature.

The purpose of the present article is to discuss this choice from the standpoint of the designer and constructor of furnaces, and to analyze in a general way the underlying principles, believing that the results of such an analysis may be of some use as a guide in determining what the best proportions are for any given conditions, either by showing how to take account of these factors or by showing that they may be neglected.

In the writer's earlier papers* it was shown that when the variations of the conductivities with temperature can be neglected, the conditions under which there will be no chilling of the furnace product by the electrode are identical with those for the least possible loss of energy in the electrodes. And that the conditions for the latter also correspond with those for the avoidance of excessive temperatures (higher than those in the furnace) within the furnace walls.

While any further corrections in these useful general results, due to the variations of the conductivities with the temperature, are probably too small to require attention in practice, and that, therefore, the writer's simplified solutions will probably suffice in most cases, yet the question has been raised as to the effect of these variations and much stress has been laid on them by others, much more, in the writer's opinion, than is warranted by their small importance. But as it was not known what the effects were, or how large they were, it left an uncomfortable uncertainty.

The purpose of the present article is to endeavor to show the nature of these, more with a view of clarifying the situation than to lay any stress on their importance; on the contrary, it is believed that it will rather tend to show that we may be content to neglect the quantitative effects of these troublesome temperature variations in ordinary practice; or when greater accuracy is required, it will show in what direction the allowances for these corrections should be made.

Dr. A. E. Kennelly, in a very able and interesting paper* was the first, and so far the only one, to demonstrate that these variations give rise to only slight differences between the law for the minimum loss and that for no flow of heat to or from the furnace—that is, the two laws are then no longer quite identical. This will also be seen to be made evident by following out the new method of analysis used in the writer's article in the March issue of this journal, page 128, on "Properties and Behavior of Furnace Electrodes," to which the reader is referred for a further description of the method to be used below.

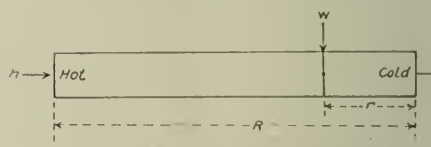


Fig. 1.—Center of Gravity of Electrode.

In the adjoining Fig. 1, which represents an electrode, let it be assumed that the material, carbon, for instance, has a falling temperature coefficient, in which case the colder half will have the greater resistance; hence, the greater half of the watts will be generated in this half. The center of gravity of this uneven distribution of the generated watts will, therefore, be to the right of the middle, as shown by the point of the arrow. It may, therefore, be assumed for the present purpose that all the watts w are generated in the section drawn through this center of gravity at the arrow marked w.

It was shown in the article in the March number, just referred to, that the number of watts necessary to keep any heat from flowing from the furnace—that is, to prevent all chilling action by the electrode, increases as this center of gravity is nearer to the cold end; if it were quite near, the number of watts would be extremely high, hence the loss very great.

It hardly requires an algebraic analysis to show that in such

* This journal, October, 1909, Vol. VII, p. 442, and revised and enlarged in *Trans. Am. Electrochem. Soc.*, Vol. XVI, p. 265.

* "On the Modifications in Hering's Laws of Furnace Electrodes Introduced by Including Variations in Electrical and Thermal Resistivity." Proceedings Am. Inst. Elec. Eng., March, p. 267.

a case if the section be increased somewhat it would reduce the number of watts generated by that same current, and in the same proportion, hence it would reduce that part of the total loss which is due to the resistance. This increase in section would, however, cause some heat to leave the furnace—that is, it would cause some chilling action. But as the latter would be small, while the reduction in the former would be relatively large, it follows that the total loss would be diminished.

Hence, without any intricate algebraic analyses, it is evident that with carbon or any other electrode material having a decreasing resistivity for an increase of temperature, the total loss could be diminished, if desired, by making the section somewhat larger than that required for no flow of furnace heat, and allowing it then to chill the furnace somewhat. The designer has his choice between the two evils. An increase of size is an evil in this case because carbon electrodes are in general already undesirably large in section before this increase is made.

In making this choice it must be remembered that a chilling action of the electrode signifies that the capacity of the furnace is reduced thereby, and it must be considered whether the gain in a smaller loss is worth this reduction in the capacity of the furnace; in general, it probably will not be. One of the chief advantages in proportioning electrodes as recommended in the writer's earlier papers, is that the capacity of the furnace may thereby be raised to its full extent.

The physical meaning of this action of carbon is that it is more economical in power to send some of the energy in the form of electricity through the whole length of the electrode into the furnace and then let it flow back again and out at the cold end, its original entrance, in the form of heat. In other words, such an electrode allows electrical energy to flow more freely in one direction than it allows the same energy in the form of heat to flow back in the other direction. It is an action somewhat analogous to a leaky valve.

With electrode materials like iron, on the other hand, which have a rising resistivity, a similar argument shows that the total loss could be reduced also, but in this case the center of gravity is nearer the hot end, and the section must therefore evidently be decreased. Hence, instead of chilling the furnace product by this change, the flow of heat would now be from the electrode into the furnace—that is, with iron electrodes it is possible not only to reduce the former minimum loss, but, in addition, to actually send more heat into the furnace than it did before when the loss was greater. It is no longer a choice between two evils, but a double gain. The attending objectionable feature, however, is that somewhere within the walls there will then be generated a higher temperature than in the furnace itself.

While it is evident that by making such changes in the section the total loss may be decreased, it is also evident, when this change of section is greatly exaggerated, that the total loss will again increase and become much greater than it was originally. This means that there is a certain section at which the loss is a minimum.

The amount of this change of section to reduce the loss to a minimum, of course, will depend upon how far the center of gravity is from the middle; the farther it is, the more will be the possible reduction in the loss. This means that the greater the change of the resistivity with the temperature, the greater will be the reduction of power loss which becomes possible by operating at a minimum loss instead of with no chilling action and with no high temperatures within the walls.

As stated above, this possible change of section therefore depends on the shifting of this center of gravity; if there is none, the conditions of minimum loss will correspond with those for no flow of heat, as in the simplest case discussed in the writer's earlier papers. If the position of this center of gravity is given by n so that $n = R/r$, in which R is the total thermal resistance and r that from the center of gravity to the cold end; and if a represents the change of cross-section necessary to change from the condition of no flow at the hot end when the section is S, to that of minimum loss when the section is s—that is, if $a = s/S$; then it can be shown that for a minimum loss $a = \sqrt{n-1}$. The proof of this law for the minimum is given below.

To illustrate this with examples, suppose $n = 4$, that is, $r = 0.25\,R$, which is believed to be a very exaggerated case, probably far beyond any that would occur in practice. It means that the center of gravity has been moved one-half the distance from the center to the cold end, if the thermal conductivity is uniform, or, if not, then to such a distance that the thermal conductivities are in that proportion. Then $a = 1.73$, that is, the new section would be almost twice as large.

Let W equal the loss for the original section when there was no flow of furnace heat. From the formula (11) given below, the minimum loss will now be $0.866\,W$, or only 13.4 per cent less than before. The generated watts w are now, from (2), equal to $0.577\,W$, or a little over half as much as before. The loss of heat h from the furnace now is, from (10), equal to just half of w, or $0.289\,W$—that is, over a quarter as much as the total loss was originally. As a check, the total loss now is the sum of w and h: namely, $(0.577 + 0.289)\,W = 0.866\,W$ as before.

Hence, even under greatly exaggerated conditions, the gain in power loss by making this change would be only 13.4 per cent, and this involves almost doubling the size of the electrode and a loss of furnace heat equal to over one-quarter of the original total loss, which would be quite likely to chill the furnace appreciably, thereby reducing its effective size. Under ordinary conditions it would therefore probably not be worth while to make this correction due to this change of resistivity.

Let it now be assumed that the resistivity rises and to such an extreme extent as to bring the center of gravity half way to the hot end. The results, calculated in the same way, now are $n = 4/3$, $a = 0.577$, $w = 1.732\,W$, $h = -0.866\,W$, and total loss $0.866\,W$. Hence, in this case the saving of power would again be 13.4 per cent, the size of the electrode would be reduced to nearly one-half, and the generated watts would be nearly doubled, but just half of them would now flow into the furnace, thereby contributing heat to it instead of chilling it as before. This would be a gain in all features, except for the fact that a higher temperature than that in the furnace would be generated within the walls. Whether this is a disadvantage depends chiefly on the temperature conditions. In a low-temperature furnace for tempering castings, for instance, it might perhaps be taken advantage of.

In making the above corrections due to the temperature coefficients, the change of section would bring about slight changes in the temperature gradients, and it will be found that these in turn react to again reduce the advantages of the change somewhat. Hence, the exact gains would be even less than the small ones given by these formulas.

This correction involves both conductivities; hence, a material like graphite, which has a slightly falling electrical resistivity and a rapidly falling thermal conductivity, would presumably act in this respect like a material with a rising resistivity—that is, like the metals and unlike carbon.

The general conclusions of the present investigation, therefore, are that, in general, these corrections are too small to be considered in practice, and that the engineer need consider them only in so far that when the material has a falling resistivity (like carbon) he had better make the section a little larger rather than smaller, and when the material has a rising resistivity he may feel safe to make the section a little smaller than the simple formulas require. This lack of importance of these temperature corrections is indicated in the writer's earlier papers by the flatness of the minimum curve, as was described at that time.

PROOF OF THE LAW.

Let W = watts generated in the electrode when there is no flow of heat through the hot end.

w = watts generated in the electrode when the total loss is least.

h = actual flow of heat through the hot end when the total loss is least.

X = total minimum loss.

a = the ratio of the section for minimum loss to the section for no flow through hot end; that is, the proportion in which the section for no flow at furnace must be increased or diminished to give the section for minimum loss. The length is assumed constant.

R = thermal resistance of whole electrode.

r = thermal resistance of the part from the center of gravity of the watts to the cold end.

n = the ratio of these two thermal resistances, that is R/r.

T = total temperature drop in the electrode.

Let all the values be known for the condition that there shall be no flow of heat through the hot end. Hence, W is known. These are determined by the laws given in the writer's previous papers, when the constants used are those determined by the method therein described,* in which they are based on the conditions of no flow at the furnace end. Now let it be required to find how much larger or smaller the section should be made in order to reduce the loss to a minimum. This increase or decrease of section is represented by the ratio a.

It is evident that when the section is now changed, the current and, of course, the temperature drop being the same, there will always be some actual flow of heat through the hot end; this is represented by h; when the section is increased the flow will be from the furnace, that is, h will be positive; when the section is decreased, it will be from the electrode into the furnace, that is, h will be negative. Attention is here called to the fact that in the present discussion h represents the actual flow of heat and not that which would flow if there were no electric current.

The total minimum loss is now the sum of that generated electrically in the electrode and that flowing from the furnace; hence

$$X = w + h \qquad (1)$$

in which w and h are to be determined.

The value of w will, of course, vary inversely with the section; hence,

$$w = W/a \qquad (2)$$

But the value of h is not so easily determined. The electrical analogy again indicates the solution. If an electric current h enters one end of a conductor, as in Fig. 1, and another, w, enters somewhere between the ends, the two joining and flowing out of the other end together; then if the condition is that in all cases the drop of potential T between the two ends is always the same, the relations between these two currents are readily found as follows:

The total drop of potential T will be the sum of that in the first part, where h is flowing alone, plus that in the second part where h and w flow together. Hence, if R is the total resistance and r is that of the second part, then

$$T = h(R - r) + (h + w)r = hR + wr \qquad (3)$$

This applies equally well to thermal currents and in this equation the letters for the corresponding thermal quantities have been used; it, therefore, represents the thermal relations

In the original electrode, in which there was no flow of furnace heat, h, this quantity h is zero; hence, if r' is the thermal resistance of that part of that original electrode between the center of gravity of the generated watts and the cold end, then

$$T = Wr' \qquad (4)$$

As the difference of temperature T between the two ends is the same for both, these two expressions may be equated, and if at the same time they are simplified by substituting

$$n = R/r \qquad (5)$$

and

$$r' = ra \qquad (6)$$

the final result reduces to

$$h = W(a - 1/a)/n \qquad (7)$$

Hence formulas (2) and (7) will give the values of the watts w generated electrically and the flow of heat h out of or into the furnace through the hot end, when the value of a is known for which the total loss will be least. It is evident that some such value of a must exist, because if the section be either increased or decreased greatly it is easily seen that the total loss must increase; hence, if it increases in either case there must be some value for which it is least.

To find this, the calculus must be resorted to, but the solution is extremely simple. The total loss X is always the sum of w and h (remembering that h here is the actual flow while the electric current is flowing, and not the flow when there is no current). Hence,

$$X = w + h = W/a + W(a - 1/a)\,n = W/n \quad \left(a + \frac{n-1}{a}\right) \qquad (8)$$

Differentiating this and placing the first differential coefficient dX/da equal to zero, gives, after reduction, the very simple relation

$$a = \sqrt{n - 1} \qquad (9)$$

which is that value of a which will make X a minimum.

Hence the change a in cross-section necessary to change over from the condition of no flow through the hot end to that of minimum total loss is very readily found from the known value of n.

Knowing this value of a, all the other quantities are easily calculated. The new generated watts, w, are found from the simple formula (2). The flow of furnace heat h may be found from formula (7), but by combining it with (9) and (2) it reduces to the simpler forms

$$h = \frac{W}{a}\left(\frac{n-2}{n}\right) = w\left(\frac{n-2}{n}\right) \qquad (10)$$

which latter also shows the relation of h to w. When this value becomes negative it means that heat is flowing from the electrode into the furnace—that is, there is a high temperature point in the electrode.

The total minimum loss is the sum of w and h, and is given in formula (8), but it may now be reduced to the more simple form.

$$X = \frac{2aW}{n} \qquad (11)$$

It might be of interest to state that when $n = 2$, that is, when the center of gravity is in the middle, the formulas all give the same results as in the writer's original paper, as, of course, they should.

Conclusion.—Incidentally, the present investigation is a good illustration of the recommendations of certain distinguished scientists, namely, that it is best to solve a new problem first under the simplest possible conditions, leaving the refinements to a second or third approximation, if indeed these refinements are necessary at all. This is the practice which the writer has endeavored to follow in his solution of the electrode problems and it has led to simple results which are thought to be amply sufficient for all practical purposes.

Philadelphia, March 9, 1910.

* More in detail in a paper on "A New Method of Measuring Mean Thermal and Electrical Conductivities of Furnace Electrodes." *Trans. Am. Electrochem. Soc.*, Vol. XVI, p. 317. Abstracted in this journal, Vol. VII, p. 514.

Near Alloys.

In the issue of Feb. 22 of *Metallurgie* Prof. K. Friedrich, of the Institute of Technology of Breslau, Germany, makes the following interesting and suggestive remarks:

Thermal research work of the last decades has proved that the melting and freezing process of an alloy does not occur at a certain fixed temperature, but takes place within a wider

range of temperature. This at least is the rule. Exceptions are the case of the end constituents of the alloys, viz., the pure metals, the chemical compounds, and finally certain solutions to which belong the eutectics and those compositions for which the curves of crystal mixtures show a minimum or maximum.

Within this freezing or melting interval, the extension of which depends on the ratio of the quantities and the nature of the constituent parts, the alloy consists of a molten and a solid portion. If the former is in excess we have a fused bath in which crystals are floating, while if the latter is in excess we have crystals impregnated with liquid drops. If we select a proper ratio of the constituents we can produce a substance which is plastic and can be kneaded. Now let us add materials of a higher melting point, work them into our dough-like substance, let us mix the mass more intimately, perhaps by means of a certain pressure, let us cool it, and we will get a product which is not yet known in practice, namely, an alloy closely interspersed with foreign substances, without having had before a homogeneous solution.

Professor Friedrich mentions the following two examples from his laboratory practice:

He makes an alloy of 90 parts by weight of tin and 10 parts by weight of copper, fuses the two substances together by heating to about 500° C., and cools to about 220° C. In this neighborhood is the eutectic point. By proper regulation of the temperature (which in this case is not difficult, since the melting interval extends to about 450° C.) he obtains a dough-like mass. He now adds the foreign substance, which may be particles of light-blue cobalt silicate, the size and quantity of which is left entirely to his judgment. He kneads it all at a constant temperature, finally using pressure, so as to get it into the desired form. He then cools it, grinds and polishes it, and thus obtains a product which shows blue interstices in a white metallic field in a fairly uniform distribution.

In his second example Professor Friedrich begins with a mixture consisting of 98 parts by weight of lead and two parts by weight of antimony. First, he forms the alloy and then cools it until the eutectic temperature—about 250° C.—is reached. The mass is dough-like. He now adds iron filings; 10 parts by weight can easily be introduced. This is followed by mixing and pressing. The result is a kind of hard lead.

Are the products thus obtained true alloys? "Alloys in the fused state are essentially solutions of two or more metals or metalloids in each other."[1] They are produced from their constituents by fusing them together, therefore, by formation of a liquid solution. That is the rule. Where other modes of procedure have reached practical importance they are based at least on the tendency of the materials concerned to enter into solution. We may call attention to the process of diffusion of carbon into solid iron low in carbon if this iron is in direct contact with carbon or carbonaceous substances.

With our products there is no necessity that the substances employed are soluble in each other. Our products are, therefore, not true alloys. They are alloy-like structures, metallic compositions, near-alloys or whatever we may call them. At all events they are but mechanical mixtures. As far as known, such mixtures have not yet found any application in practice. They have been produced, however, before, especially by Spring,[2] who pressed powdered solid metals together under high pressure. This does not result in genuine alloys, as was said of Spring's experiments, but simply in mixtures as proved recently by Masing.[3]

Professor Friedrich's method differs from the method of Spring in that he does not start from solid metals, but from a dough-like substance. This means an advantage over Spring's process. The use of the substance in the dough-like condition permits the production of intimately mixed mixtures without employing very high pressures. The added particles can be rather coarse grained.

Of course, the dough-like condition is limited to a certain, and often brief, range of temperature. It is, therefore, necessary not only to heat the mixing and pressing devices, but to regulate the temperature exactly. With small quantities the latter condition is easy to fulfill by employing electric heating. But even with larger quantities the temperature may be maintained constant in practice without any great difficulty. Professor Friedrich draws attention to the Pattison process, where in a proportionately large kettle a silver-lead bath can be kept for one or two hours at a temperature of about 300° C. within 10° or 15°.

As to the properties of these new near-alloys it is, on the one hand, a disadvantage that they are only mechanical mixtures, but, on the other hand, it is an advantage that we are almost perfectly at liberty to select the material to be added. The only limitation is that the added substance remains solid and unaffected at the temperature of the mixture.

To go into details Professor Friedrich admits that in regard to tenacity, toughness, resistance against chemical influences, and good thermal and electric conductivity his near-alloys can hardly compete with genuine alloys even if a strong pressure has been employed in their preparation. For it would be exceedingly difficult to produce a mechanical mixture with such an intimate intermingling of its constituents as in an alloy that has been produced from homogeneous liquid solution.

The greatest disadvantage is that it is impossible to cast the near-alloys. On this account they cannot be used for many purposes.

On the other hand, with our near-alloys, we have the possibility of making metal-like substances perhaps of an especially low thermal and electric conductivity, since we can add to the fundamental alloy oxides, silicates, and, in general, poor conductors.

Further, the hardness of the near-alloys can be adjusted to much greater nicety than has been possible heretofore. Quartz, corundum, carborundum can be embedded in alloys. In a lead-antimony alloy iron may be embedded and its hardness adjusted exactly to that of the material of the axle.

The greatest field of usefulness of these near-alloys would be, according to Professor Friedrich, the production of "terrazzo"-like mixtures with a metallic base, as well as articles for general usage and for decorative purposes with peculiar color effects. As substances to be added there are at our disposal precious metals, artificial and natural minerals, especially precious stones, further enamels and glasses. Besides the tin-copper alloy mentioned above, silver-copper alloys would be also suitable as a starting material. For a composition of 90 parts by weight of silver and 10 parts by weight of copper the range of freezing temperature extends from 778° C. to 890° C.

Finally, the process permits perhaps the replacement of more expensive alloy constituents by cheaper ones. Professor Friedrich mentions the substitution of iron for antimony in hard lead, although in this respect the impossibility of casting the near-alloy is especially troublesome. But there might be left, or could be found, a small market anyway for this new kind of hard lead.

Altogether it is evident that near-alloys cannot enter into serious competition with genuine alloys. They are no full-fledged substitutes for the same, and can only be considered as of an inferior grade. But where they extend the present applications of alloys they may find use.

[1] Ledebur, Die Legierungen. 1906, p. 1.
[2] Bull. Acad. Belg. since 1878; especially Vol. 49 (1880), p. 323.
[3] Zeit. f. anorgan. Chemie, 1909, Vol. 62, p. 265 to 309.

The Magnetic Properties of Electrolytic Iron and Some Commercial Steels.

By Charles F. Burgess and James Aston.

In connection with tests of the various properties of iron alloys made from electrolytic iron,* measurements were made of the magnetic quality. The results on certain of the binary

* Investigation of electrolytic iron and iron alloys made in the Chemical Engineering Laboratories of the University of Wisconsin under a grant from the Carnegie Institution of Washington.

series have been presented in several articles in previous issues of Metallurgical and Chemical Engineering. In these discussions it was pointed out that the primary material in the preparation of the alloys was electrolytic iron of high purity (99.9 + iron); and a common standard of comparison throughout all the tests was a bar forged from an ingot of electrolytic iron.

In this paper we are presenting data of tests made upon various samples of electrolytic iron, together with those upon the original anode material, indicating thus the effect of refining upon the magnetic quality. We have also embodied for comparison the results upon commercial samples of different degrees of purity.

Magnetic testing is not on such a basis at present that data of different investigators, at different periods of time and by different methods of testing, are strictly comparable. Even in the absolute ballistic method results are often markedly inconsistent; and comparisons of heterogeneous collections of data are likely to be misleading. All of our data, while open to the common objection to intercomparison, were obtained by the same operator upon the same apparatus, with every precaution taken, before any series of tests were made, to have consistent checking to a standard; the results, therefore, may be considered strictly comparable among themselves.

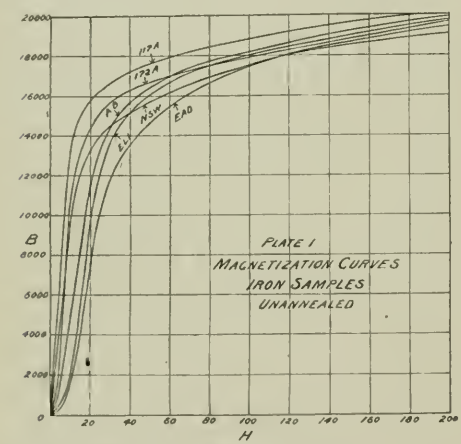


FIG. 1.—MAGNETIZATION CURVES. IRON SAMPLES. UNANNEALED.

Electrolytic Iron Tests.

Tests under four different heat treatments have been made upon the sample noted in the following table:

Bar	Character.	C.	P.	Si.	S.	Mn.	Fe.
NSW	—Swedish anode, forged	0.260	0.007	0.109	0.007	0.021	99.596
EAD	—Single refined as deposited, not forged	0.019	0.013	0.002	0.006	0	99.960
117A	—Electrolytic, melted and forged	0.047	0.016	0.062	0.005	0	99.870
AB	—Electrolytic, melted and forged	0.120	0.014	0.001	0.009	0	99.856
ELI	—Electrolytic, melted and forged	0.012	0.031	0.008	0.019	Tr.	99.920
172A	—Electrolytic, melted and forged						

NSW is forged from one of the anode bars as used in the single refining tanks, and was bought for high-grade Swedish iron. The analysis indicates a somewhat high carbon content, but the material has the desirable quality of being very low in phosphorus, which is a troublesome element in the refining operation, as will be seen by examination of the phosphorus content of the refined bars. The indications are that all of the phosphorus had been carried over to the cathode, and the percentage content in the product increased because of the loss of iron in the slimes. However, it must be noted here that analyses of the double refined iron showed a phosphorus content of only 0.003 per cent. The analyses above show also the difficulty of preventing the absorption of carbon by the charges, due to the reducing action of the iron upon the carbon monoxide furnace atmosphere.

Bar EAD was cut from a heavy single refined cathode sheet without melting or forging (in fact, without any heating whatever) and is thus representative of the electrodeposited material. Bars 117A, AB, ELI, and 172A are made of electrolytic iron melted into ingots and forged into rods. For 172A, double refined material was used.

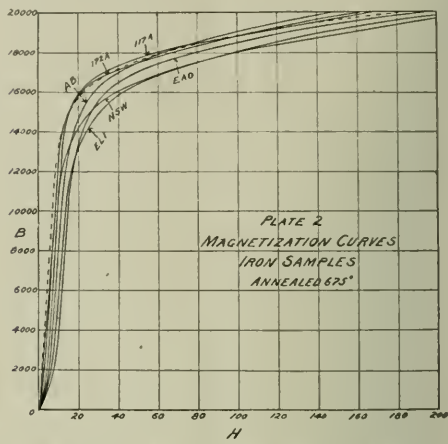


FIG. 2.—MAGNETIZATION CURVES. IRON SAMPLES. ANNEALED 675°.

The results of the tests are indicated in Tables I and II and in the curves plotted in Plates 1, 2, 3 and 4. In the unannealed condition, the forged electrolytic iron bars (117A, AB, ELI, and 172A) represent a considerable variation in relative quality. This cannot be ascribed to the difference in purity, as will be noted on comparison of the analyses. The differences are very likely due largely to the molecular condition of the samples, as

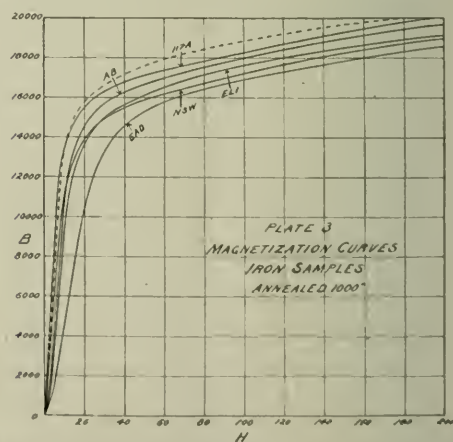


FIG. 3.—MAGNETIZATION CURVES. IRON SAMPLES ANNEALED 1000°.

a result of lack of uniformity in the forging temperature and to strains set up in working. Bars 117A and 172A are especially good, the former in particular being so uniformly good in the unannealed state and throughout the entire length of the curve that it was chosen as the standard comparison curve for all our tests.

The effect of successive heat treatments is seen by inspection

of the curves and tables. For comparison, the curve for 117A as unannealed is indicated by a dotted line. A marked improvement throughout the entire range of the curves is the result of heating to 675° and slowly cooling, due no doubt to the removal of all forging strains and the opportunity afforded for the grain size of the material to become normal. The improvement is most pronounced in the poorer samples of the unannealed tests (AB and ELI) and it is very slight in bar 117A, indicating, perhaps, that the forging treatment left it in practically its best molecular condition.

A second heating has been accompanied by an improvement of the quality in the lower ranges of the magnetizing force,

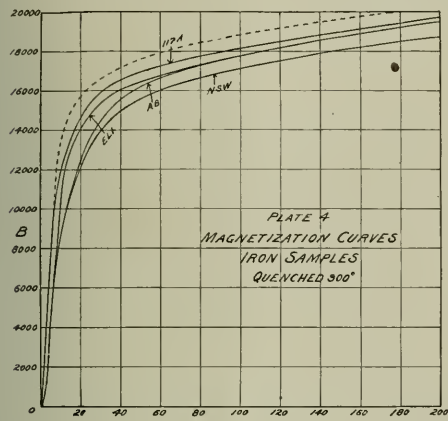


FIG. 4.—MAGNETIZATION CURVES. IRON SAMPLES. QUENCHED 900°.

but except for these ranges, the densities fall below those for the lower annealing temperature.

Quenching from a temperature of 900° C. has been accompanied by a deterioration of the quality for all magnetizing forces. This effect is most marked for the lower valves of H, but, all in all, it is very slight, indicating in a way the purity of the material, since, although the temperature was sufficiently high, the absence of carbon had prevented any hardening. This is more clearly brought out in Table II, where the coercive force and retentivity are indicated for the various treatments. The coercive force, indicative of the magnetic hardness, which in turn varies with the physical hardness, has become less under the successive heatings.

ment the superiority of this Swedish anode is less marked, even at low values of H; and in all cases, at higher values of the magnetizing forces (H above 20) the electrolytic curves reach higher maxima of density. Further annealing at 1000° C. has only served to increase the general effect just mentioned.

The influence of the carbon is most interestingly evident in

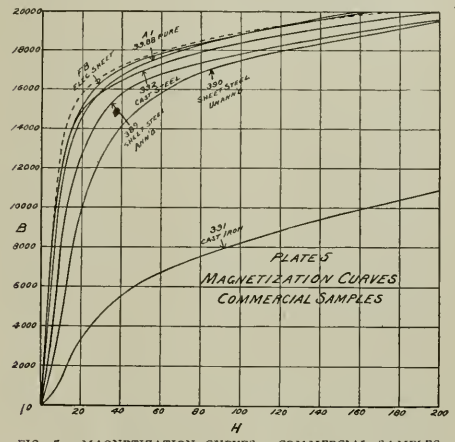


FIG. 5.—MAGNETIZATION CURVES. COMMERCIAL SAMPLES.

the following table, where the coercive forces are given after annealing at 1000° C. and again after quenching at 900°.

Bar.	Per cent, C.	Coercive Force Annealed, 1000°.	Coercive Force Quenched, 900°.
ELI	0.012	6.0	5.5
117A	0.047	3.1	2.5
AB	0.120	5.5	5.5
NSW	0.260	3.9	5.0

In this group of curves, one of the most interesting is EAD, made, as mentioned previously, from electrolytic iron as deposited, without melting or forging. In the original condition it may be classed as rather poor material, falling well below the other curves in the lower ranges of magnetizing force, and having the high coercive force of 18.5. After the first annealing, however, it has become very good material, taking a place in the midst of the others. The coercive force has fallen off to 9.3, or one-half of its former value, and the value of B for H = 20 has almost doubled, from 7300 to 13,250. The effect may be ascribed to the driving off of the hydrogen occluded in

TABLE I.—DATA OF MAGNETIC TESTS.

Bar. Composition.	H = 10 Unannealed.	H = 10 Annealed, 675°.	H = 10 Annealed, 1000°.	H = 10 Quenched, 900°.	H = 20 Unannealed.	H = 20 Annealed, 675°	H = 20 Annealed, 1000°	H = 20 Quenched, 900°.	H = 50 Unannealed.	H = 50 Annealed, 675°.	H = 50 Annealed, 1000°.	H = 50 Quenched, 900°.	H = 100 Unannealed.	H = 100 Annealed, 675°.	H = 100 Annealed, 1000°.	H = 100 Quenched, 900°.
NSW—Anode	9,500	10,000	10,850	8,400	13,200	14,200	13,900	12,050	15,700	16,500	15,850	15,550	17,450	18,000	17,200	17,100
EAD—As deposited	1,650	4,000	4,700	....	7,300	13,250	10,300	....	14,550	17,050	15,300	....	17,450	18,500	16,900	....
117A—Electro-forged	13,100	10,000	13,350	11,800	15,750	15,950	15,450	14,600	17,600	17,700	17,050	16,950	18,850	18,850	18,200	18,150
AB—Electro-forged	4,900	7,700	10,900	8,400	11,450	14,650	14,600	12,400	16,400	17,500	16,650	16,200	18,150	19,050	17,600	17,750
ELI—Electro-forged	2,000	6,000	9,300	10,000	9,550	13,050	13,600	14,050	16,000	16,400	16,100	16,450	18,000	18,050	17,600	17,750
172A—Electro-forged	10,400	12,500	8,000	....	14,300	15,900	12,700	....	16,700	17,450	16,100	....	17,900	18,850	17,600	....
AI—High purity	10,800	....	....	....	14,500	....	....	....	17,200	....	....	....	18,800	....	....	....
277—Commerc'y pure	11,100	....	....	....	14,500	....	....	....	17,100	....	....	....	18,600	....	....	....
258—Norway	11,700	....	....	....	14,800	....	....	....	16,500	....	....	....	18,100	....	....	....
131—Bessemer	7,900	....	....	....	13,100	....	....	....	16,400	....	....	....	18,200	....	....	....
FB—Sheet	....	11,900	....	....	....	15,200	....	....	....	17,500	....	....	....	18,800	....	....
390,389—Sheet	4,200	8,200	....	....	9,800	12,700	....	....	15,100	16,300	....	....	17,400	17,800	....	....
392—Cast steel	....	11,700	....	....	....	14,900	....	....	....	16,800	....	....	....	18,300	....	....
OH—Open hearth	7,800	....	....	....	12,200	....	....	....	15,700	....	....	....	17,700	....	....	....
CRU—Crucible	3,600	....	....	....	8,100	....	....	....	13,700	....	....	....	16,600	....	....	....
248—Tool	400	....	....	....	1,600	....	....	....	10,100	....	....	....	14,900	....	....	....
391—Cast iron	1,300	....	....	....	3,200	....	....	....	6,200	....	....	....	8,200	....	....	....

The effect of the refining is indicated by comparison of the curves for bar NSW, the anode material, with those discussed above. Unannealed, and at lower magnetizing forces, it is very good magnetically; in fact, it excels two (AB and ELI) of the four refined bars. 117A and 172A are better, however, throughout the whole range of tests. After the first heat treatment the electrodeposition, and perhaps the allied alteration in crystalline structure.

Commercial Samples.

Included in Tables I and II and plotted in Plates 5 and 6 are the results of tests on commercial samples of types representing

varying degrees of purity. The descriptions of the materials are as follows.

Bar. Character.

A. I. High purity commercial iron, made by basic open-hearth process. Purity, 99.884; C, 0.030.
Sample cut from rolled bar and tested without heat treatment.

277. Commercially pure iron—composition and heat treatment unknown.

258. Norway iron—composition and heat treatment unknown.

131. Bessemer steel—composition and heat treatment unknown.

F. B. Sheet steel—made and treated especially for electrical purposes.

390. Second quality sheet steel from large electrical manufacturer—not annealed.

389. Same as 390, but annealed.

392. Cast steel used by large electrical manufacturer.

O. H. Open-hearth cast steel for electrical work.

CRU. Crucible cast steel from same source as O. H.

248. Tool steel.

391. Cast iron.

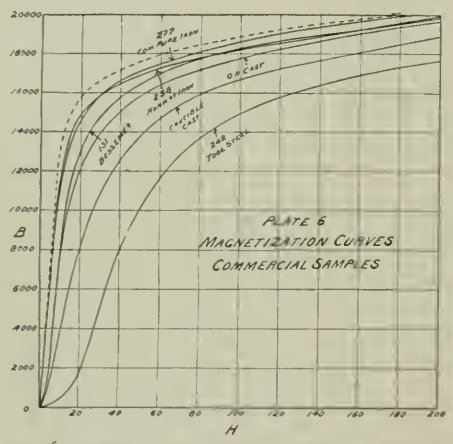


FIG. 6.—MAGNETIZATION CURVES. COMMERCIAL SAMPLES.

The direct comparison of these materials with the electrolytic samples can be made through the medium of standard 117A, which is shown on Plates 5 and 6 by a dotted curve.

TABLE II.—COERCIVE FORCE AND RETENTIVITY.

Bar.	Composition.	Coercive Force. H (MAX) = 200 Unannealed.	Annealed, 675°.	Annealed, 1000°.	Quenched, 900°.	Retentivity. H (MAX) = 200 Unannealed.	Annealed, 675°.	Annealed, 1000°.	Quenched, 900°.
NSW	Anode	5.5	4.8	3.9	5.0	11,400	12,700	9,000	8,500
EAD	As deposited	18.5	9.3	6.2	.	11,200	12,000	4,900	....
117A	Electro-forged.	5.5	6.2	3.1	2.5	12,300	13,800	10,000	8,000
AB	Electro-forged	10.3	8.1	5.5	5.5	12,000	13,800	10,700	8,500
EL1	Electro-forged.	7.9	9.5	6.0	5.5	13,000	12,000	10,900	13,400
117A	Electro-forged.	7.0	5.0	5.8	..	12,400	12,300	9,400	....
AI	High purity	4.0	..	..	..	9,600	....	....	....
277	Com. pure	3.8	..	..	..	10,100	....	....	....
258	Norway	4.2	.	..	..	11,350	....	....	....
131	Bessemer	7.1	..	..	..	12,250	....	....	....
F B.	Sheet	..	3.0	..	.	....	9,800	....	....
390, 389	Sheet	9.3	5.2	.	.	9,650	8,900	....	....
392	Cast steel	..	7.6	..	..	....	9,700	....	....
OH	Open hearth	7.1	..	..	..	11,400	....	....	....
CRU	Crucible	11.5	..	..	..	9,200	....	....	....
248	Tool	29.1	..	..	..	10,150	....	....	....
391	Cast iron	12.3	..	..	..	4,800	....	....	....

But little comment is necessary in this comparison. The effect of chemical purity of the material is indicated by the high quality of those materials of higher purity. The result of judicious annealing is shown in curves 389 and 390, a low-grade material markedly improved by the heat treatment.

Hysteresis tests on several of these materials are given below, all being carried to a uniform maximum flux density of B = 12,000 per square centimeter.

Bar.	Loop Area.	Relative Area.	H (Max.).
117A	8.65	1.	8.2
NSW	8.14	0.94	13.0
389	11.36	1.31	16.1
390	20.56	2.38	28.0
FB	7.25	0.84	10.3
AI	9.18	1.06	12.0

Bars 117A and NSW were tested after quenching, and this should be borne in mind in making comparisons.

A few general conclusions may be pointed out as a result of the tests.

The refining of commercial steels results in improvement of the magnetic qualities.

The amounts of impurities met with in commercial practice do not result in a serious deterioration of the quality.

Judicious heat and mechanical treatment has an importance equal to that of chemical composition, and proper annealing undoubtedly results in marked improvement in the magnetic quality.

Chemical Engineering Laboratories,
University of Wisconsin.

A Rotary Extractor for Precious Metals from Solutions.*

By W. D'Arcy Lloyd and Ernest T. Rand.

From time to time attempts have been made to precipitate gold and silver from their cyanide solutions while the precipitating agent was in motion, with more or less success. For example, zinc shavings have frequently been subjected to mechanical motion while in contact with the gold and silver cyanide solutions, and in most cases as far as actual precipitation of the precious metals was concerned, some success has been achieved, but mechanical difficulties have developed due to the method of applying motion by attrition, etc.

The main idea underlying all the attempts to perform the operation of precipitation with the precipitating agent in motion, is to avoid the fouling of the zinc, which is so painfully evident in the first compartments of all extractor boxes after a three days' run from time of dressing. It is well known that after thorough washing and redressing, the zinc will go on precipitating as briskly as ever until fouling starts again. The natural inference from this is that if the zinc could be kept continuously in motion, thus preventing fouling, the highest efficiency would be maintained. Having this in view, we have evolved the machine of which we endeavor to give some idea.

The precipitating agent used was zinc in the form of shavings and clippings, the latter being cut from sheet zinc and approximately ¼ in. to ⅝ in. square. The machine is a rotating one, with a water-tight casing of comparatively light steel plate, the casing being fitted with cross-sectional plates to divide it into compartments. In the machine with which the experiments were carried out, there were four of these compartments, but there is reason to think that the result required would be attained with three, the third being one-third larger than the first and second.

In the center of these division plates are securely fixed bearings, in which a hollow shaft rotates. Two circular disks, in each compartment distant from each other almost the full length of the compartment, are securely bolted to flanged collars, which in turn are fixed to the shaft. The shaft, with these discs and other attachments, form the rotor.

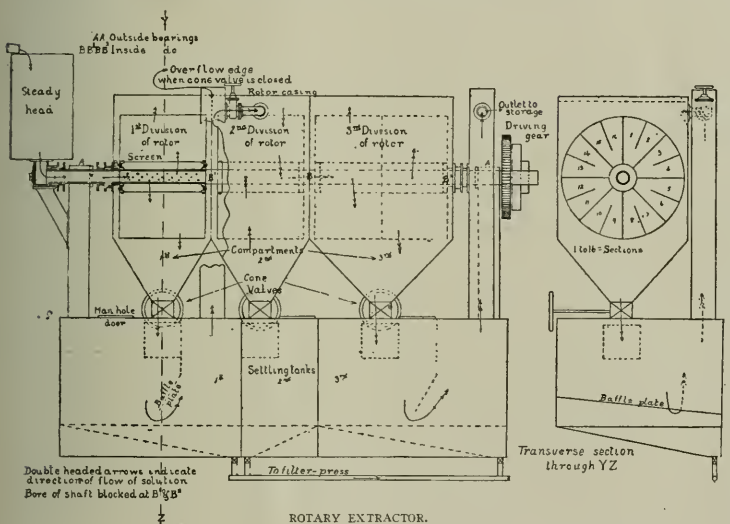
The rotor, besides being divided into compartments in length,

*A paper read before the Chemical, Metallurgical and Mining Society of South Africa. From the December, 1909, issue of the Journal of the Society. A short abstract of this paper appeared in our last issue, page 152.

is subdivided radially, in appearance much as a transverse section of an orange. There are sixteen radial divisions, which number was arrived at experimentally, the pressure of solution during rotation of a smaller number tending to press the zinc against the wall of the divisions, leaving a space between the zinc and the next wall. The use of the larger number of radial divisions obviated this tendency. These radial divisions are removable, and are lightly and strongly constructed of perforated (3/16-in. holes) steel plate.

Between and attached to the discs a suitable framework is fixed to contain the sections. There are many simple methods of retaining the sections when in position. Each section has a door of the same material as that of which the section is constructed, so that each is independent of its neighbor.

Around the shaft, and fixed securely to the flanged collars to which the rotor discs are fixed, is a strong screen of ⅛-in. mesh, leaving a space of about 2 in. between itself and the shaft, to prevent any fine material entering the shaft. The shaft is hollow, the bore being of ample size to allow of the free flow of the solution. The parts of the shaft enclosed by the flanged collars are perforated by inch holes to allow of the free egress and ingress of solution. At the required position the bore of the shaft is tightly closed to regulate the flow of the solution to wherever it is required.



ROTARY EXTRACTOR.

At the inlet end of the rotor for the solution is a small tank to act as a steady head, the top of this tank extending about 18 in. above the top of the rotor casing. At the outlet end of the casing, the pipe for carrying away the effluent is so placed that a constant level of the solution is maintained within the casing, which insures that the rotor will be completely submerged.

The bottom of each of the casing compartments is made in the form of a cone, and to this is fixed a valve which connects the cone with a settling tank, whose total capacity is equal to one and a half times that of the casing; this is divided into compartments to correspond with the number of those in the casing. All the solution passes from the casing into the settling tank from both the first and third compartments. The solution passes from the second compartment into the third compartment of the casing along the hollow shaft.

The compartments in the settling tanks have a sloping bottom to one point in each, at which point a valve and pipe connection is made communicating with one main pipe, which leads to a filter press. To the first compartment of the settling tank a stand-pipe of ample capacity is fixed, and stands outside and close to the rotor casing and projects a few inches above it. A pipe from this standpipe delivers into the second compartment of the rotor casing. A similar standpipe is supplied to the third compartment of the settling tank, and the outlet pipe from this delivers to the sump, the position of this pipe determining the level of the solution in the rotor casing, as before mentioned.

In the case of the first and third compartments of the settling tank a baffle plate is placed between the inlet of the rotor and casing and the exit to the standpipe. Each compartment is supplied with a manhole door. A screen basket is placed underneath each inlet to the tank to catch any fine zinc that may have passed through the rotor casing. A suitable gearing for rotating the machine is fixed to an extension of the shaft at the exit end of the machine, and about 1 hp is absorbed in a machine precipitating 250 tons a day.

The operation of the machine is as follows:

The solution enters the steady head, passes into the hollow shaft, and as the bore is blocked at the position corresponding to the end of the first division of the rotor, it passes out through the perforations, coming in contact with the zinc and leaving the rotor peripherically. The solution next passes into the settlement tank, and beneath the baffle plate, rising to the constant level in the standpipe to the second division. From here it again enters the perforated shaft, passing inwards through the zinc, along the shaft, and outwards through the zinc in the third section, again downwards, through the settlement tank (where the speed of the flow is greatly lessened by the size of this receptacle) beneath the baffle plate and up the standpipe, thence to the storage sumps.

By the arrangement of the settling tank it will be seen that there is but small chance of slimes being carried over to the other division. The rotor revolves from one to one and a half times per minute, and this is fast enough to allow of a slight motion of the zinc in their sections, which is as much as is desired.

All gold slime loosened from the zinc passes to the tank beneath; this can be removed at any time without stopping the machine, by closing the valve connecting the rotor casing and the settlement tank, and allowing the solution to overflow the division plate, forming the first and second compartments. The contents can then be pumped into a filter press, the solution from this being returned to any rotor desired. By means of the manhole before mentioned the inside of the tank can be washed out, this closed and valve slightly opened until settling tank is full of solution, then the valve opened full.

With reference to zinc consumption, it may be stated that the only zinc taken to acid was that which passed through the rotor in the form of shorts during its operation.

The following is a summary of the results obtained over a period of from November 4, 1908, to May 26, 1909.

Runs were made with strong, weak and slime solutions, respectively, the results obtained being as follows:

	Tons of solution per cubic foot of zinc per 24 hours	Value of solution entering, dwts.	Value of solution leaving, dwts.	Percentage of extraction Per cent.
Strong	1.01	2.221	0.053	97.6
Weak	1.84	0.383	0.029	92.3
Slimes	1.53	0.460	0.031	93.2

The above figures represent the mean of assay results over the individual runs.

At the clean-up the solution is shut off, the belt on step cone shifted to faster speed and rotor run on this for a few minutes, then stopped. (We might here mention that the sections on our experimental machine were fixtures, also there was no settlement tank below.) The sections on the top side were opened up, the zinc taken out and rinsed, though it was found that there was but a small quantity of loose slimes entangled. The zinc was returned and the section filled up with zinc from the second division. We found that finely cut filiform zinc broke up into shorts, when using strong solution, so that coarsely cut zinc would be preferable.

When coarse zinc or clippings are used, the adhering gold may be removed in an acid bath, as it was found in the case of clippings that 94 per cent of the gold was removed in a 5 per cent acid bath in 15 minutes, any deteriorating effect on the clippings being negligible. After the acid bath the zinc is thoroughly washed in clean water, the sections opened and examined, and fresh zinc added where necessary; it is found that the zinc in this form can be used in the rotor for a long time with a minimum consumption of both zinc and acid. To remove the gold from the zinc the sections are placed in a drum, which is revolved in the acid bath. We experimented for three weeks in our rotor with clippings and found the precipitation as good as under the conditions of the same space packed with filiform zinc (finely cut), the weight of clippings being 11 lb. to 1 lb. of shavings.

Our experimental machine was somewhat crude, and devoid of perhaps the greatest mechanical essential to its success, i. e., the settling tanks, the solution passing directly from one division to another without separation of precipitated slimes. The advantages of the rotor may be summed up as follows: (1) Ready access by authorized persons to the gold slimes, without stoppage of precipitation, enabling a daily clean-up to be made if required. (2) With clippings and coarsely cut filiform zinc, absence of breaking down and readiness of removal of adhering gold in an acid bath without consumption of all the zinc, with a consequent saving of zinc and acid. (3) Clean-up quick and cleanly with a consequent advantage of a minimum of loss. (4) Shortness of time rotor is out of action during dressing. (5) No fouling of zinc. (6) While in operation no danger of theft of gold slimes, as all valves would be locked.

In the discussion which followed the reading of the paper, the chairman, Dr. James Moir, remarked that on theoretical grounds, any rotary process will expose a larger surface of zinc, but, of course, it remains to be seen whether any particular device is successful commercially.

Mr. A. F. Crosse thought that there was "a great deal in this paper." The authors claim several advantages for their method, but they have left out one most important advantage, namely, the mechanical driving off of the hydrogen bubbles from the zinc; this removal of hydrogen helps precipitation.

Mr. E. T. Rand, one of the authors of the paper, confirmed that the bubbles of hydrogen on the surface of the solution are very evident indeed. There is a very violent action going on continually

The Chemical Industries of America.*

By Prof Charles E. Munroe

The topic which you have done me the honor to invite me to address you upon appears on first consideration quite specific, but investigation shows that this is not quite the case. Thus we find the popular idea of a chemical industry to be one producing acids, alkalies, salts, explosives, fertilizers, dyestuffs and extracts, pigments, distillation products and elementary substances like bromine, phosphorus, sodium and others, and the officials of the United States Census Bureau in 1880 in fixing a classification, styled in the various censuses "Chemical Production" or "Chemicals and Allied Products," adopted this popular view.

In discussing this, I have said,* "A reason for the variation in the industries included at the different censuses is found in the very general and indefinite title used, for in the strictest technical sense every material thing is a chemical, and accordingly every industry in which the materials used undergo a chemical change in the process of manufacture, as in the smelting of iron from its ores or the production of leather from a hide, may be considered as a chemical industry. It is evident that if this view of the significance of the title were taken "Chemicals and Allied Products" would properly cover every manufacture except those like furniture making, machine construction, or textiles, in which the material remains unchanged in composition during the manufacture, but is turned, or cast, or woven into other shapes. The popular idea of the term limits its application, but admits as chemical industries the manufacture of gunpowder, fertilizers, and similar mixtures, whose ingredients undergo no chemical change during the process of compounding the mixtures. It thus became necessary to decide arbitrarily upon the industries to be included. Those so included at the census of 1905 may be divided into the following classes. Sulphuric, nitric, mixed and other acids; sodas; potashes; alums; coal-tar products; cyanides; wood distillation; fertilizers; bleaching materials; chemicals produced by the aid of electricity; dyestuffs; tanning materials; paints and varnishes; explosives; plastics; essential oils; compressed and liquefied gases; fine chemicals; general chemicals."

These were consequently divided into 19 different classes which were given separate treatment. The combined statistics for these classes for the censuses of 1900 and 1905 are set forth in the following table, the statistics of these two censuses only being compared because they alone dealt with the same materials.

TABLE I—CHEMICALS AND ALLIED PRODUCTS OF UNITED STATES 1900 AND 1905

	Establishments, Number	Wage Earners, Number	Total Wages	Materials Used, Cost	Products, Value
1905	1,786	59,198	$29,515,863	$176,400,680	$282,469,216
1900	1,691	46,700	21,783,335	124,018,044	202,506,076
Increase	95	12,498	7,732,528	52,382,636	79,663,140
Per cent. of increase	5.6	26.8	35.5	42.2	39.3

From Table I it is observed that there was an increase in every item enumerated, but that not only was the actual increase in the number of establishments less than that of any other item, as was to be expected, but that the percentage increase was less. This indicates that the growth of these industries was rather by increased production of existing establishments than by the creation of new ones. In fact, in a more detailed analysis it was found that in some industries the number of establishments had actually decreased, though each of the other items, as enumerated in Table I, showed an increase.

The greater percentage increase in wages over that of the percentage increase in wage earners shows that the lot of the latter was improved and possibly indicates that a better class of labor was employed, and, since the percentage increase in the number of salaried officials for these establishments was 29.6, while the percentage increase in salaries was but 32.4, it is obvious that the wage earners fared, on the whole, better than the salaried officials.

A wholesome feature to be observed is that while the increase in the number of men employed was 12,104, the increase in the number of women employed was but 413, while there was a decrease of over 10 per cent in the number of children employed. I speak of this condition as a wholesome one, because, outside of the clerical and perhaps analytical work, the duties to be performed in these establishments is essentially man's work

* Address delivered, by invitation, on Dec. 9, 1909 at the annual meeting of the American Institute of Chemical Engineers in Philadelphia

* Bull. 92—Census of 1905, page 9, by Charles E. Munroe.

The greater percentage increase in the cost of materials used as compared with the percentage increase in the value of the products shows the growing necessity of intelligent and careful management and skillful workmanship to prevent waste and to increase yields. This is emphasized by examination of the additional item of miscellaneous expenses, which, while less in the total than any of the values given in Table I, showed an increase of 77.2 per cent.

As indicated, the census classification of "Chemicals and Allied Products," which gave the data just discussed, is a purely empirical one, and it deals with but a very few of the true chemical manufactures of the United States. It is not possible to derive from the returns, of the various industries as taken, the data for an exact scientific classification, such as has been referred to above. Yet, in order to arrive at a better conception of the application of chemistry in manufacturing industries and its magnitude, we may follow such a scheme of classification as that employed in many chemical technologies, though here again we meet with the difficulties common to classification and we are compelled to include in our data some of the products of purely physical processes, either because these processes are operated collaterally with, or are related to, the predominating chemical processes, or else because the products are closely associated with the chemical products.

In assembling this data it should be said that in order to compare the data of the different epochs one must first eliminate from the data of 1900 the returns for neighboring industries, for the census of 1905 was a factory census which considered only the results of manufacture as carried out in factories, and not solely for consumption at the point where manufactured, as is generally the case with neighborhood industries. The results of this treatment are set forth in Table II.

TABLE II—CHEMICAL INDUSTRIES OF THE UNITED STATES 1880 TO 1905

	1905	1900	1890	1880
Establishments, number	56,580	53,567	40,451	34,864
Wage earners, avge. number	1,107,714	943,166	677,123	490,776
Wages, total	$575,635,257	$438,404,062	$305,884,278	$176,227,726
Materials used, cost	2,933,660,817	2,215,162,767	1,247,239,582	924,573,773
Products, value	4,716,490,371	3,628,641,475	2,152,490,514	1,357,593,293

Table II, imperfect though it be, both in the industries it includes and those it omits, gives a better conception of the actual magnitude of the industries in which chemical transformations play a part, and which are, therefore, really chemical industries, than Table I does and in so doing it shows the value of the products for 1905 alone to be nearly seventeen-fold greater than is set forth in Table I.

The increase is easily accounted for by noting that items, such as soap, with a product valued at over $68,000,000; glass over $79,000,000; illuminating gas over $125,000,000; dairy products over $168,000,000; refined petroleum over $175,000,000; paper and wood pulp over $188,000,000; bread and other bakery products over $269,000,000; sugar and molasses over $277,000,000; vinous, malt and distilled liquors over $340,000,000; smelting and refining of copper, lead and zinc over $461,000,000; iron and steel over $905,000,000, and many other items have been added to those embraced in Table I.

The simple enumeration of these items indicates how incomplete the statistics usually presented as those of the chemical industries are and how insufficient the popular conception of the chemical industries is. Yet even the data of Table II does not present the case in full, since all agricultural products, amounting in value in 1900 to $4,717,069,973, are really the results of chemical processes and are, therefore, the products of chemical industries, although not factory products.

As with Table I, so with Table II, the deductions are more readily drawn by observation of the increase and percentages of increase for each item at the various epochs. These have, therefore, been ascertained and are set forth in Table III.

Considering now the data of Table II, and more particularly the increases and percentages of increase set forth for each epoch in Table III, while keeping firmly in mind the fact that we are here dealing with two 10-year periods and one five-year period, it is again to be noted that both the actual and percentage increases in the number of establishments are the smallest of all the various increases set forth, and that increase for this item for the 1900-1905 period is not actually less than for 1890-1900 and 1880-1890, as should be expected, but is proportionately less, thus emphasizing what has been deduced from Table I as to the increased production of existing establishments.

TABLE III—INCREASES AND PERCENTAGES OF INCREASES FOR CHEMICAL INDUSTRIES

	1900 to 1905		1890 to 1900		1880 to 1890	
	Increase	Per Cent.	Increase	Per Cent.	Increase	Per Cent.
Establishments, number	3,013	5.6	13,116	32.4	5,587	16.0
Wage earners, average number	164,548	17.4	266,043	39.3	186,347	38.0
Wages	$137,231,195	31.3	$133,519,784	43.3	$129,656,552	73.6
Materials used, cost	718,498,053	32.4	967,923,182	77.7	322,665,809	34.9
Products, value	1,087,848,896	30.0	1,476,150,961	68.6	795,987,221	58.6

Likewise the consideration of the data for this larger number of industries extending over a greater length of time shows that not only is the percentage increase in wages nearly as great at the census of 1905 as those for cost of materials and greater than the value of products, but that, while the proportionate increase in the number of wage earners for the 1900-1905 period is less than that of 1890-1900, the proportionate gain in wages is greater. In fact, all statistics point to markedly improved conditions for the wage earner in the chemical industries, and to his increased participation in the income from the enterprise. This fact is one to be reckoned on by the chemical engineer in making up his estimate for the cost of a projected enterprise, which it is proposed to install.

The statistics of Tables II and III, on the other hand, do not so markedly support the deductions drawn from Table I as to the increase in cost of materials used when compared with the increase in the value of the products in 1900-1905. However, when we consider the larger items included in these later statistics, such as iron and steel; smelting and refining of copper, lead and zinc, and others, we may each of us recall a variety of labor-saving devices which have been invented and introduced for cheapening the cost of production and handling of the raw materials of these industries, and that the inventions have increased in number and perfection with the growth in magnitude of these industries.

An increase in cost of materials is in conformity with the long-recognized natural law of supply and demand. A modification of this law through which labor may get its fair share of increase and capital may get its fair share of increase, while the actual cost may not proportionately be increased, has been brought about in recent times through the increase in the magnitude of the unit of demand, or, in other terms, the quantity handled. As stated, this has to an extent been rendered possible by the introduction of labor-saving machinery, much of which has been invented in this country.

But, in my opinion, and if I read aright the reports of foreign commentators on our chemical industries, in their opinions, the chief modification in the operation of this law has been made

in this country through the development of "team work," though the writers style it organization or systemization.

Entering on my fortieth consecutive year of college teaching I might, from what has been so persistently dinned into my ears, have been led to believe that "team work" originated in the minds of the college youths who flocked to Franklin Field or to the Harvard Stadium. Sitting on the bleachers with practical politicians and presidents, I might be led to suppose that "team work" was an invention of the professional athlete. As a fact, the idea of "team work" is a very old one and military in its essence and original application. It is embodied in our national motto. It is commemorated in the "Charge of the Light Brigade." But this older practice, while greatly promoting efficiency, demanded such unreasoning subordination that the private soldier was properly looked upon as but "food for powder," and when this system was introduced into the factory the operator became but "a cog in the machine."

The modification in this plan of "team work" which has been developed to such advantage in the industrial plants of this country has come through a recognition of the great value of individuality and the necessity for its preservation and development, and it has been demonstrated that the higher the intelligence of the individuals who merge their entities with that of their fellows in a common purpose, and the more complete their comprehension of the means used and the end sought, the more successful is the result, whether gauged by the quality, or the quantity, or the cost of the output. I am happy to say that the chemist has destroyed the older military idea, even in the army, for by his invention of high-powered smokeless powder he has compelled armies to fight in open order so that each individual must exercise his own powers in attack and defense, and be trained to take the initiative in case of necessity.

Naturally the application of labor-saving machinery and of "team work" is most readily made and yields most efficient results in the production, transportation and handling of the raw materials of our larger industries, and it is in these that we find the smaller proportionate increase in the cost of materials

American industries, in which the chemical industries are included, have signalized themselves by the introduction of standards, by the introduction of interchangeable parts into mechanisms, by the wide application of labor-saving machinery, and by the use of "team work." Yet notwithstanding the vast resources of this country, their ease of access, and the cheapening, by methods such as described, of many of costs of production, the cost of "living," not only here, but throughout the civilized world, has steadily increased, and I attribute this largely to the work of the chemist.

At St. Louis, in 1904, I said: "Technical chemistry, then, invades the domains of economics, of politics, and of diplomacy. A striking example of its effects in economics and politics is found in the settlement of the silver question. Gold is a most widely diffused metal. It has, for instance, been shown by assayers at the United States Mint at Philadelphia that if the gold in the clay of the bricks of which the buildings of the Quaker City are built could be brought to the surface, the fronts would all be gilded. In the past our processes for the isolation of this metal have been so costly that only the richer ores would bear treatment. Large bodies of low-grade ores which have been discovered and mountains of tailings carrying values were looked upon as worthless, while enormous quantities of copper, lead, and other metals containing gold were sent into the market to be devoted to common uses, because the cost of separation was greater than the value of the separated products. Eight years ago, when the "silver question" was made the national issue, while the orators were declaiming from the stump, the chemists were quietly working at the problem in their laboratories and factories. Manhe's process for bessemerizing copper ores was combined with the electrolytic refining of the product, so that even traces of gold were economically recovered, while the cyanide processes, such as the MacArthur-Forrest, the Siemens-Halske, the Pelatan-Clerici, and others for the extraction and recovery of gold from low-grade ores and tailings, were successfully worked out and put into practical operation to such effect that by the cyanide processes alone gold to the value of $7,917,129 was recovered in the United States in 1902, which is more than was ever won throughout the whole world by all methods in any one year up to 1661, and probably up to 1701. The data for other purposes is not at hand for 1902, but the returns for 1900 show that gold to the value of $88,985,218 was recovered in the treatment of lead and copper ores in the United States, of which $56,566,971 worth was recovered in refining. It has but recently been publicly proclaimed in this city of St. Louis that the "silver question" is settled, and it is settled, but it was settled largely through the efforts of the technical chemist and metallurgist."

With the improvements in methods and diminution in cost of extraction the Pactolean stream has continued to flow in steadily increasing volume* until the flood of gold has become so great that its purchasing power has become markedly reduced, and costs, measured in terms of gold, have become markedly greater. With this condition well determined the chemist has again stepped in to increase the cost of living by requiring the application of costly methods of inspection of food, drugs, and other articles of consumption; by demanding the elimination of preservatives which permitted the abundance of the harvest being kept until time of need; or the plethora of one locality being sent to the land smitten with leanness; by insisting on the introduction of expensive sanitary arrangements. Pure-food laws are the vogue and all the other needs of man are becoming the subject of special legislation, some wise, but much otherwise. It would prove an interesting exhibit if a statistician were to assemble the actual costs in the administration and execution of these laws in this country alone during the past five years.

I speak with earnestness because I have repeatedly been a participant in these movements, and am even now engaged in an analogous humanitarian enterprise, and I know that a certain result of all such endeavors to improve the lot of man is to put the community to an increased expense.

(To be concluded.)

The Precipitating Effects of Substances Containing Various Forms of Carbon and Cellulose on Cyanide Solutions Containing Gold and Silver.**

By A. J. Clark and W. J. Sharwood.

These notes contain the results of a series of experiments and observations made by the authors at the Homestake Mine, Lead, South Dakota, together with other information on the subject, gathered from various sources.

Relative Activity of Some Precipitants.—Two sets of experiments were made on the precipitating effect of gold-bearing solutions of various precipitating substances, with the results tabulated below under Series A and Series B. In each experiment 300 c.c. of solution were taken. In Series A five grams of the precipitant were added; the mixture was stirred vigorously for a moment and then filtered, with further agitation

*The Production of Gold from 1878 to 1908 Was as Follows

Year	World's Production		Production in United States	
	Fine Ounces	Value	Fine Ounces	Value
1878	5,761,114	$119,092,800	2,476,800	$51,200,000
1888	5,330,775	110,196,900	1,604,841	33,175,000
1898	11,827,866	256,879,700	3,118,398	64,461,000
1908	21,378,481	441,912,300	4,854,140	94,560,000

** A paper read before the Chemical, Metallurgical and Mining Society of South Africa. The complete paper, as printed in the January issue of the *Journal* of the Society, contains a list of references to existing literature.

each time the funnel was refilled with the liquid. After filtration the filtrate was passed through the filter again, and thus exposed to the sediment retained by the filter a second time.

In Series B the conditions were the same as in Series A except that a larger quantity of precipitant (which was not weighed, but was at least 20 grams) was taken, and there was only a single passage through the filter.

Series.	Precipitant.	Original assay of solution.	Assay of filtrate.	Percentage of gold precipitated.
A.	Graphite (Dixon's flake).....	$2.63	$2.65	0
	Bituminous coal (40 mesh)...	"	2.45	7.0
	Coke (40 mesh).....	"	2.11	19.6
	Charcoal, pine (40 mesh)....	"	1.92	27.0
	Calcium carbide.....	"	2.50	5.0
	Zinc dust	"	.02	99.3
B.	Graphite	2.73	2.73	0
	Coal	"	2.13	22.0

Repeated tests have shown similar results. Pure graphite, whatever its source, seems to be entirely without effect on the gold in cyanide solutions, and anthracite coal nearly so. The semi-bituminous or sub-bituminous coals of Western United States (practically bituminous coals, but having the high water content and brownish tinge when powdered, characteristic of lignites) have a feeble precipitating action; well-burned coke is fairly effective and charcoal much more so. Lubricants containing graphite with mineral oil, intended for use in rock drills, and the clean graphite extracted from them, are entirely without effect after several hours' contact with solution.

Either a sufficiently long contact with, or a slow percolation through a relatively large amount of bituminous coal, removed as much, in some cases, as 99 per cent of the gold in a cyanide solution, but the efficiency of the coal as a precipitant fell off with repeated use, as is the case with charcoal.

Effect of Wood Ashes Containing a Small Proportion of Charcoal on the Cyanide Extraction of Gold from Tailings Containing Pyrites.—These experiments were made in percolators upon pyritous tailings, to which various proportions of ashes were added, together with a suitable amount of lime (about 5 lb. per ton), the charges being thoroughly mixed. The cyanide solution was in actual contact with the charges four days, but the charges were drained dry once during their treatment, as well as at the conclusion.

The tailings assayed about $1.75 in gold per ton, and upon them, under normal conditions, without the presence of wood ashes, an extraction of at least from 70 to 75 per cent would have been obtained.

The results of the experiments were as follows:

Percentage of ashes added to tailings.	Percentage of gold extracted by cyanide.
0.17	52.8
0.8	46.0
1.73	35.8
8.0	3.5

At the Homestake cyanide plants, when similar ashes or charcoal have been inadvertently introduced into the treatment vats, a very small percentage has been found to lower the gold extraction from between 70 and 75 per cent to below 60 per cent. As to the origin of these ashes, in one case it had been traced to an old stope, into which they had been dumped from the surface some years before; in another case charcoal had been produced by the charring of the timbers in a former mine fire, and had become mixed with the ore treated. Sometimes charcoal and cinders have been introduced into plants by the dumping into the mill launder of ashes from a heating stove, as a convenient means of disposal; at other times accumulations of ashes and clinker, produced by the burning of chips (collected from the mortars), had been run through the batteries.

Some years ago heavy losses were reported in the cyaniding of ore at an Australian mine, at which crushing and amalgamation were facilitated by the device of heating the ore by burning it like limestone in a furnace with alternate layers of cordwood, and quenching the heated ore with water. A similar kiln treatment has been used successfully at some small California mines, rendering refractory ore containing pyrite and tellurides more amenable, both to crushing and amalgamation in an arraster, but it seems to have proved fatal to cyaniding in the former instance, probably owing to the presence of charcoal. Park (in "The Cyanide Process," 1897, ed.), states that in New Zealand it was found advisable to hand-pick the charcoal from dried ore prior to crushing and cyaniding.

Several instances have been reported where "graphite" ore (usually slate or schist) has caused precipitation. Most likely this is due to the fact that the carbon in such carbonaceous schists is often far from being completely converted into graphite, and may exist largely in the form of hydrocarbons, in a condition similar to that of partially burned coal or wood.

Precipitating Effect of Chips, Charcoal, etc., Upon Gold- and Silver-Bearing Solutions.—Some floating rubbish, consisting of charcoal and charred pieces of wood, as well as some clean chips and twigs (which came from a settling dam and creek bed) and particles of coke (which came from the smokestacks of the locomotive and steam shovel) were skimmed from the surface of the cyanide solution in a slime agitator vat.

The solution contained approximately:

Gold, 0.023 oz. per ton ($0.47).

Silver, 0.28 oz. per ton ($0.14).

Potassium cyanide, 0.025 per cent.

The skimmed rubbish was rinsed twice with water, and dried at 100°C. It was then divided into two portions, one containing the charcoal and charred wood, and the other the clean chips and twigs. The two portions were then weighed, some of each being burned in clean roasting dishes in a muffle. The substances were assayed, before and after burning, with the following results:

No.	Material.	Proportion of total weight of original sample.	Ounces per ton. Gold.	Ounces per ton. Silver.	Values per ton. Gold.	Values per ton. Silver.	Total.
1.	Coke and charcoal charred wood	40%	2.25	7.50	$45.27	$3.75	$49.02
2.	Ash of same....	¼ (or 25% of total ash)..	18.00	60.00	365.40	30.00	395.40
3.	Clean chips and twigs.....	total ash)...	0.96	10.8	10.48	5.40	24.88
4.	Ash of same....	¼ (or 75% of total ash)..	0.96	10.80	19.48	5.40	24.88

As neither of the two classes of material carried more than its own weight of solution, when removed from the vat, obviously a considerable precipitation had occurred in proportion to the weight of the material, the coke, charcoal, etc., precipitating the gold contents of approximately 100 times their weight of solution, and the twigs, etc., precipitating ten times their weight, the corresponding amounts for silver contents being 27 and 10, respectively.

In reply to a suggestion by Mr. Caldecott that these chips, etc., might have been partially burned or charred, it may be stated that they were to all appearance unaltered. It is possible, however, that some of them had lain in the dam or creek for months, or years, so that decay may have commenced.

Decaying Wood.—That decaying wood is an active precipitant for gold in cyanide solution was evident from a series of samples taken from a residue dam, the sand discharged into which contained appreciable amounts of solution as well as traces of dissolved gold, so that the aeration consequent on discharging no doubt resulted in the dissolution of a little more gold. While the average assay value of the sand was lower than that of the residue as discharged, the lower strata were slightly higher in gold contents, as might be expected from the leaching that had since taken place. At two points, however, a decided concentration of values was noted; one of these was found to be in the neighborhood of a pine post, and another around the decaying stump of a tree that had been killed by the residues. Mr. Caldecott's suggestion that the effect of the wood fibre itself is practically nil, and that apparent precipitation by wood is due merely to absorption of solution, is confirmed by our next series of determinations.

Absorption of Gold and Silver by Cellulose.—Several writers have published experimental results showing a large decomposition of cyanide and also a considerable precipitation of precious metals, due to contact with wood, canvas, cocoanut matting and other fabrics used as filters—all forms of cellulose. Some of the writers, working on a laboratory scale, had failed to take into account the ratio of the surface of the precipitating material they used to the volume of solution experimented upon, and did not then compare this ratio with that which would have resulted by working on a large scale. Consequently some operators have been misled with believing that discrepancies were attributable to precipitation by the wood of tanks or filter bottoms, and therefore to hope that, at some future date, the burning of the material would reimburse them for their supposed losses.

As the writers referred to differed considerably from one another, either as regards their opinions, or in experimental results, we give some of our results obtained by examination of various materials that had been exposed to gold-and-silver-bearing solutions for considerable periods.

(1) **Materials from a Sand Leaching Tank.** This was nearly 40 ft. in diameter, the bottom having an area of 1200 ft. It had been in use over a year when the first tests were made. The values given represent the contents of the entire material in the tank, through which many thousands of tons of solution had circulated, carrying up to $6 per ton in gold, and up to $2 silver.

(a) Bottom Filter Cloth of Canvas.—8-oz. duck, 1,200 sq. ft. in use for eight months. Entire cloth: gold, 5 cents, silver 2 cents.

(b) Bottom Filter of Cocoanut Matting.—1200 sq. ft. in use 12 months. Entire filter: gold, 46 cents; silver, 3 cents.

(c) Wooden Slats.—These supported the filter and were of unpainted Oregon pine, 3800 running feet with a cross-section of 2.5 x 1.5 inches, about quarter of the material being crozed out; altogether about 75 cu. ft. In use 12 months. Entire material: gold, $16.60; silver, $5.90. Wood per cu. ft. gold, 22 cents, silver 8 cents.

(d) Fine Slime.—This had accumulated under and between the filters during eight months, and consisted of fine ore, together with precipitated carbonate of lime; altogether, 200 dry pounds. Entire material: gold, 20 cents; silver, 4 cents.

In the last case, and probably in all, most, if not all of the values were contained in solution absorbed and not necessarily precipitated.

(2) **Tank Staves.**—(a) The staves of a similar large leaching tank, after use between three and four years with sand averaging $3 gold and $1 silver value per ton. They were of California redwood, originally painted inside with two coats of P and B. paint, and outside with red oxide of iron in linseed oil. A slight coating of carbonate of lime had formed on the inside. A large sample from these staves, burned and assayed, yielded only traces of gold and silver.

(b) Similar staves from a leaching tank in use at another plant for four years contained 40 per cent moisture. The dried wood contained gold, 47 cents per ton; silver, 3 cents per ton.

(3) **Precipitating Box.**—A precipitating box of 1½-in. lumber was painted inside and out with P and B. paint, and used three years to precipitate low grade cyanide solution with zinc shavings, the slimes averaging about $1 to $5 per lb. Samples were taken by boring completely through the wood at various points in sides and bottom, and were burned and assayed. A little zinc slime was visible on the surface.

The average value of the material, including a trace of adhering precipitate was: Sides, 0.75 cent per sq. ft. bottom, 2 cents per sq. ft.

(4) **Mine Timber.**—A piece of blackened but sound timber (yellow pine) which had been some time underground, and contained ferrous and ferric sulphates derived from mine water was tested. At the time of the test it contained 56 per cent of water. Kept in contact with about three times its weight of cyanide solution, which assayed $1.20 per ton gold value, no precipitation took place in 20 hours. It seems evident, therefore, that wood, when well painted with any suitable protective medium, absorbs little more than traces of precious metal from cyanide solutions of ordinary value.* As dry pine and redwood absorb about their own weight of water, one would expect such wood, if dried after a thorough soaking in solution to assay the same value per ton as the solution taken up—evaporation and precipitation excluded. Alternate soaking and drying or evaporation on the outside of unpainted wooden launders or tanks in a dry climate, might be expected to result in a distinctly higher value in the wood. Adhesion of rich telluride slime or floured amalgam may also cause an increased value.

It is difficult, however, to reconcile with one another the results obtained by the following authors, and only those of Bosqui are at all in line with our own experiments. The other writers named worked entirely on a laboratory scale.

S. H. Williams (*Eng. and Min. Jour.*, Sept. 9, 1905, p. 80, 435) found that an 8-in. wooden launder, 75 ft. long, used seven months at a cyanide plant in New Mexico, having passed 63,000 tons of solution, worth $3 per ton, contained: 2.766 oz. gold and 32.646 oz. silver in 15.5 cub. ft., or 0.18 oz. gold and 2.1 oz. silver per cub. ft., or a total value of about $4.65 per cu. ft. An apparent typographical error in the original article makes the value $60.45 per cu. ft.

A. N. Mackay (*Trans. Inst. Min. Met.*, 1905, p. 14-41; *Min. and Scient. Press*, June 17, 1905, p. 90-387) states that red pine, from a tank used 18 months, in which had been treated only 1200 tons of ore containing 20 dwt. of gold per ton, yielded 1.24 oz. gold per ton of wood.

F. L. Bosqui (*Eng. and Min. Jour.*, Feb. 26, 1898, p. 65-248) experimented with pine and redwood dressed and rough, both with intermittent exposure and drying, and with continuous immersion, for periods up to 12 months, using solution which averaged $5 gold per ton and more or less silver. The highest absorption was with rough pine, exposed altogether about 12 months, this assaying $5.18 gold and 14 cents silver per ton of wood.

On the whole, it seems probable that the precipitative effect of charcoal may be occluded gases, but definite proof is lacking. S. Brussow (*Zeit. Chem. Ind.*, Kolloide, 1909, p. 5-137; *Jour. Chem. Ind.*, Oct. 15, 1909) in a recent article, attributed the precipitation of gold from haloid salts by charcoal to absorption. This seems most unlikely, as he admits that the chlorine of gold chloride remained "almost entirely" in the solution; and we have found that charcoal, which has been in contact with solutions of either chloride or double cyanide of gold, is thoroughly impregnated with metallic gold easily distinguishable with a good lens or low power microscope.

*When launders of unpainted wood are used in the Plattner chlorination process they absorb gold enough to make it worth while to burn them to recover it. The same is true of canvas used for concentration in mills where floured amalgam escapes the plates or where the tailings contain fine gold or rich tellurides. Absorption of silver similarly takes place in hyposulphite leaching. This is preventable by soaking the launders, etc., in paraffin or painting with asphaltum.

On the Determination of the Atomic Weight of a Chemical Valence and on the Mean Chemical Energy Per Unit of the Atomic Weight.

By Dr. Gustavus Hinrichs.

The Weight of a Valence.—After reading the paper in the March issue, it will be readily understood why the ratio *H* to *O* did not come out exactly equal to 1 to 16, but gave for *O* 16 values for *H* slightly in excess of unity. It is this excess that foreshadowed the separate existence of the chemically active portion of the hydrogen atom.

Supposing the slight excess to be due to the hydrogen alone

and due to the smallest weight of matter imaginable (a pantogen atom), we have found the atomic weight of pantogen (the primitive ponderable matter) equal to 1.128 of our unity, as already referred to in the preceding paper.

But this view of valence cannot be restricted to one chemical element, to hydrogen, which, on account of its small atomic weight, would reveal it first; it must be a general property for all elements. We should, therefore, have two such prominent panatoms on oxygen, three on nitrogen, four on carbon.

Calling the weight of this panatom-valence v and retaining the absolute values for the nullovalent mass of the element-atoms we shall have, for example, the entire atomic weight of silver nitrate $170 + 10\,v$.

Hence the atomic ratio of silver nitrate to silver, which for $v = O$ is $R = 1.57407$, will become $R' = R\,(1 + 0.050\,v)$.

This method of calculation is general and now will have to be applied to all cases. Let us see whether we can get a determination for the value of a valence from the existing syntheses of silver nitrate, made so carefully by Berzelius, Stas and Richards. We must, however, remember that the values of Stas are affected by systematic errors which do not show in the work of Richards, because he has used nearly the same weights of silver in all his determinations. For this reason it will be impossible to take the value calculated for v from this synthesis as entirely exact.

The analytical ratio according to the determinations of Stas is 1.57473; according to Richards 1.57479. The atomic ratio for $v = O$ is 1.57407, giving enormous excess of 66 to 72, which has remained a good deal of a puzzle for a long time. We have made various attempts to get at the real reason of this enormous excess without satisfaction beyond the point that the fact stated by Schützenberger fully accounts for the systematic error of Stas and for his two values, the one for dried, the other for fused nitrate.

The above relation gives

$$R' - R = 0.00072 = 0.050\,Rv = 0.079\,v,$$

from which we get $v = 0.009$ to nearest thousandth of a unit.

The synthesis of water by Noyes gives the same value as can easily be verified. It is true these two series of determinations belong to the best we have, but it will not do to accept them as final and *absolutely exact* in this matter, which is close to the limit of precision of analytical work.

From the investigations I have been able to complete I am sure that the value v of nine-thousandths of a unit is *a higher limit*, and that the true value is less than this amount.

It will now be necessary to apply this method to all the 300 reactions used during the past century for atomic weight determinations. For $v = O$ this work has been completed; the results have been put into five such tables as the first, published last July in this journal. The results have been discussed for two of these tables in a recent article in the *Moniteur Scientifique*, and the general results have been represented by diagram and table in connection with my Quelques Letters, page 39, just from the press.

There can remain no doubt but the absolute atomic weights are true as a first and a very close approximation. But it will now be necessary to repeat the reduction of this entire mass of data, the product of laboratory work of a hundred years, for a convenient assumed value of the weight of a valence. From the results of this work the true value of v can then be readily estimated.

This second approximation—which will probably be the final one—we have already carried out in a number of the most important determinations, using the value $v = 0.005$ as the most convenient and practical.

The following more formal statement of the present condition of this great problem may be useful:

I.—The principal part of the atom of any chemically active element is not chemically active, but nullovalent.

II.—The weight of this nullovalent part is strictly commensurable, generally to the unit, not unfrequently to the half, rarely to the quarter, and still more rarely to the eighth of a unit of our scale.

III.—The active part of the valence consists of as many panatoms prominently placed at one end of the prismatic or discoid nullovalent mass as the valence. True At. Wghts., pages 222-223.

IV.—The higher limit of the weight of the unit valence is 0.009; its true value may be 0.005 only, and will have to be determined by a second, final revision of the laboratory work done during the past century.

V.—The atoms of the argonoids or nullovalent elements have a definite geometrical form without polar or prominent panatoms.

Energy of Chemical Reactions.—The combination of 2 gram atoms of hydrogen with 1 gram atom of oxygen sets free an energy equal to 68 kg calories, in round numbers.

The combustion of 1 gram of hydrogen thus produces 34 kg calories.

There is no other element or substance known which by combustion produces as high an amount of caloric energy.

We cease to wonder at this exceptional position of hydrogen when we consider that in a hydrogen atom the chemically active accelerative force is weighed down by the smallest inert atomic mass, namely, of one unit only.

By following this trend of examination for the most prominent oxygen and chlorine compounds, we shall find the caloric energy per unit of mass quite regularly and rapidly diminishes with the increase of the nullovalent mass, as it ought to do, if our above deduction is correct.

The following two tables show that this remarkable diminution of the chemical caloration with the increase of nullovalent matter of the atoms is indeed a fact.

TABLE I.—FOR AO.—Hinrichs' Principles Chem., 1874, p. 148.

Formula.	Caloration in Kgr. Degrees C. For Compound.	For One O.	Weight A.	Per Unit Weight of A.
H_2O	68	68	34	34.0
CO_2	96	48	6	8.0
CO	30	30	12	2.5
SO_2	71	35.5	16	2.22
P_2O_5	381	76	12.4	1.63
P_2O_3	97	32	21	1.52
As_2O_5	222	44.4	30	1.48
ZnO	85	85	65.5	1.30
FeO	69	69	56	1.23
SnO_2	138	69	59	1.17
As_2O_3	154	51	50	1.02
CuO	44	44	63.5	0.69
PbO	55	55	207	0.26
Ag_2O	12	12	216	0.06

TABLE II.—FOR ACl.—Hinrichs' Principles Chem., 1874, p. 148.

Formula.	Caloration in Kgr. Degrees C. For Compound.	For One Cl.	Weight A.	Per Unit Weight of A.
HCl	24	24	1	24
$NaCl$	95	95	23	4.13
$KaCl$	101	101	39	2.59
$FeCl_2$	99	49.5	28	1.76
$ZnCl_2$	101	50.5	32.7	1.55
$SbCl_3$	86	29	40	1.38
$AsCl_3$	75	25	25	1.00
$CuCl_2$	59	29.5	32	0.92
$PbCl_2$	90	45	103.5	0.43
$AgCl$	35	35	108	0.32

This may be sufficient for a preliminary statement of the general mechanism of chemical activity.

St. Louis, Mo., March 12, 1910.

Notes on Chemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

Grading of Pig Iron.

Important Proceedings.

In consequence of the resolution passed at the last autumn Congress of the International Testing Association at Copenhagen, that for the purpose of defining the quality of pig iron more accurately than was possible by the present method of grading by fracture appearance, the committee having the matter in hand should inquire in the various countries concerned as to how far specifications by analysis could be usefully substituted for grading by fracture, a meeting of representatives

of bodies interested in the iron and steel industries was held at the Iron and Steel Institute on Feb. 10 under the presidency of Mr. J. E. Stead, who represented the Iron and Steel Institute and the Cleveland Institution of Engineers.

Mr P. Donaldson, on behalf of the Scotch Pig Iron Trade Association, wrote expressing his opinion that grading by analysis was the correct method to adopt; but, as yet, he questioned if the departure from the grading by fracture would be accepted in Great Britain.

Mr. H. Pilkington (British Foundrymen's Association) wrote pointing out the different conditions existing in England and in America, where practically all the foundry pig was made from hematite. The value of the fracture method of grading had been proved, yet it was somewhat crude when numbers alone were used for classification. An improved method was desirable, because fracture, by itself, was not a reliable index of definite properties. All the suggested chemical specifications were open to objection, and their authors did not always appreciate the commercial aspect of the situation. At present the only practicable course was to keep to grading by fracture numbers and sell on the silicon content.

Mr. J. E. Stead then rose, and after referring to the great difference between conditions of trade in England and those in America or Germany, said fracture appearance had sufficed for many years for large export sales. Grading by analysis only would considerably enhance the cost of pig iron, and it would be better that the change should be gradual, as, in fact, it was coming about now, especially in some consignments to America and Germany. British makers did not object to selling by analysis, but the cost would increase, and at present sales by analysis were not applicable to the trade generally. Another question for consideration was the storing of surplus pig, and he believed that ironmasters, warrant-holders and storekeepers would oppose the change on account of the increased expense and the inconvenience it involved. He objected to any change from the existing practice.

Mr. T. C. Hutchinson (Cleveland Ironmasters' Association) said the system of grading by fracture appearance had proved its efficiency for a period of over 60 years, and fracture appearance did to some extent indicate the analysis. In iron from Cleveland ore the phosphorus, manganese and carbon were practically uniform, and the variation in silicon and sulphur was shown by fracture sufficiently for every practical purpose. In the case of large sales of iron intended for steel making, sale by analysis was desirable and would doubtless become more customary, but for general trade purposes the existing grading met every requirement. He quite agreed with Mr. Stead's observations as to storing by analysis. Cleveland iron was bought in big lots by large firms who sold it in small quantities, and in shipping such large cargoes it would not be practicable to keep small analyzed parcels separate. He was sure that makers, merchants and buyers would all object to the proposed change; and, on behalf of the association which he represented, he had to state that its members would refuse to be bound by any resolution to grade their iron by analysis which might be passed by the meeting.

Mr. H. J. Skelton (British Iron Trade Association) believed that the agitation for the proposed change began in the United States in consequence of difficulties which arose when sandless pig was first produced there. It was well known that almost every parcel of pig made in this country was analyzed. If pig iron were to be sold by analysis a difficulty would generally arise in fixing limits of variation. He dealt largely with special pig, of which analyses were always given regardless of its destination, but the analysis was not guaranteed, in order to provide against possible rejection on account of small variations. Although analysis was becoming more customary, it was not possible for British makers to agree to invariable sale by analysis.

Mr J. E. Stead remarked that the agitation was not coincident with the advent of sandless iron in America. Foundrymen learned the value of making their mixtures in accordance with analyses, and for that reason were now adopting the practice of buying by analysis.

Mr. W. F. Foster (Staffordshire Iron & Steel Institute) said the information as to composition to be obtained by fracture had not yet been completely worked out. The comparison of fracture with analysis would give more knowledge than a superficial observer would credit. Fracture revealed comparatively small differences in phosphorus content.

Mr. W. H. Butlin (British Iron Trade Association) stated that there was an overwhelming majority of the members of the association he represented in favor of the retention of grading by fracture. Those in favor of fracture grading represented an annual output of 1,476,000 tons and those against it only 120,000 tons. Purchase by analysis was progressing in this country. It was certainly beneficial in the case of foundry pig, and might lead to the use of cheaper iron for certain purposes, and this scientific mixing might bring about economy in manufacture.

Mr. J. E. Stead mentioned that purchasers were protected by legal decisions which established certain limits to permissible variations in analysis.

Mr. H. J. Skelton remarked that British pig iron was superior to the product of other countries. Even in Germany, with scientific control, the use of British pig iron was indispensable because the native pig did not yield the desired results.

After some discussion as to the wording of the resolution to to be put to the meeting, Mr. T. C. Hutchinson said he agreed with Mr. Foster that the extent of the information given by fracture appearance was not yet appreciated. Selection by analysis alone was impossible, and fracture appearance gave a very good indication of the composition of the iron.

Mr. G. H. Roberts (Royal Arsenal) proposed: "That the conditions governing the sale of pig iron in Great Britain make it undesirable to recommend any immediate change in the existing practice of grading, whether by fracture or by analysis."

This was seconded by Mr. W. H. Butlin, and carried unanimously.

The resolution will be communicated to the International Testing Association by Mr. G. C. Lloyd, member of the council for Great Britain.

Properties of Alloys of Copper, Aluminium and Manganese.

At the meeting of the Institution of Mechanical Engineers on Feb. 18 the discussion on the ninth report to the Alloys Research Committee (abstracted on page 154 of our March issue) was resumed by Dr. Glazebrook, who said that the results of the high-temperature tensile tests agreed closely with earlier observations and demonstrated that a considerable increase in temperature did not diminish the strength of the alloys, but had a tendency to augment it. With regard to abrasion tests three of the alloys exhibited much less loss than rail steel and tool steel. As to corrosion the degree to which some of the alloys were affected was but slight, and those alloys appeared to be really valuable.

Mr. Sherard Cowper-Coles said that electro-deposited metals might have quite as high a tensile strength as smelted and rolled metals. The study of cold electro-deposited metals and alloys would assist in clearing up several obscure points in metallurgy. Lantern projections of photomicrographs were employed to demonstrate the speaker's contentions. One showed electro-deposited copper in crystals at right angles to the cathode surface, which corresponded to the surface of the ingot mold in casting. Another was a section of a copper strip deposited in a fine V-shaped spiral cut in a mandril. If the groove were rounded instead of angular the arrangement of the crystals would be radial, and weak cleavage lines would occur. A third slide exhibited the setting of the crystals and how the deposited copper was removed in a continuous strip. The effect of particles of impurities was then shown with crystals radiating from a minute nucleus. Such structure might account for the breaking up of metals during rolling. Cast metals might

contain small particles of foreign matters which would similarly cause radial crystallization and weak cleavage lines. The effect of variation in the rate of rotation of the anode was demonstrated by the deposit of copper on a cone. The tensile strength of the copper taken from near the base of the cone was 24 tons per square inch, while the copper near the apex of the cone gave only a few tons. It was conclusively established that metals having all the various physical properties of cast, drawn or rolled metals could be produced by deposition without any heat treatment, rolling or drawing.

Mr. H. F. Donaldson doubted the utility of some of the alloys reported on because of their cost. Equally good tensile strength could be obtained with much cheaper ingredients. He would compare the alloy No. 2 of the author with one containing 36 per cent of zinc, 55 per cent of copper, 2 of iron, 4 of manganese, and 3 of aluminium, having a tensile strength of 47 tons and 30 per cent elongation, which cost but £10 per ton as against £17 per ton for the "No. 2" alloy. He questioned if manganese was the best material to add, and thought nickel might give better results. He had brought a tool made from an alloy consisting of 90 per cent of copper and 10 per cent of aluminium, hardened by hammering. It was used in a cordite factory, and retained its sharpness for fully 48 hours. The speaker produced some bronze knives, used in cordite works, which had thin blades with very sharp edges.

Mr. H. L. Heathcote said he had experimented during the last two years with light aluminium alloys, with the object of producing a non-corroding alloy for cycle rims. He found that pure aluminium corroded far less than an alloy containing 3 per cent to 4 per cent of copper when acted on by solution of sea salt and moist air at 60° C. Alloys of aluminium with 1 per cent to 5 per cent copper, of aluminium with 1 per cent to 9 per cent of iron, of aluminium and nickel, and of aluminium, copper and zinc, all corroded more than commercial aluminium. The only alloy which satisfactorily resisted corrosion was one containing 1.4 per cent of manganese, which when rolled to 15-gage flat strips had a tensile strength of 12 tons, 12 per cent to 15 per cent elongation, and 0.45 per cent to 0.55 per cent contraction. He preferred tensile strength to weighing for measuring the extent of corrosion.

Dr. Rosenhain, in the course of his reply, with reference to Mr. E. F. Law's remarks on the use of a temperature of about 1000° C. and a silicon content for alloys Nos. 10 and 11 of 0.57 per cent and 0.25 per cent, said the heating certainly affected the silicon content, which was always about 0.2 per cent or 0.3 per cent, but the temperature quoted was necessary for the addition of the manganese. He did not think anybody could much improve on his results.

The Boiling Points of Metals.

A paper by Mr. H. C. Greenwood on the "Influence of Pressure on the Boiling Points of Metal" was recently read before the Royal Society. Previously work at reduced pressures had been limited by lack of material capable of maintaining a vacuum at high temperatures, and for similar reasons nothing had been done on the effect of high positive pressures. These difficulties had been avoided by arranging the whole furnace inside an enclosure, in which the desired changes of pressure could be produced. Heating was effected electrically; the temperatures were observed optically, and the boiling point determinations were made by a visual method. Experimental pressures ranged from 100 mm of mercury to 50 atmospheres. The boiling point of bismuth under 102 mm of mercury was found to be 100° C., and under 16.5 atmospheres 2060° C. The boiling points

Market Prices in February, 1910.

Copper was in the neighborhood of £60 15 0 when the month opened, but fell away during the first week to £58; thence recovered uncertainly to £59 12 0, at about which price it has remained.

Tin, after a fall to £145 15 0 on the 4th, rose to £150 15 0 during the next 10 days, and has maintained the price of £151 up to time of writing.

English Lead has been rather depressed, falling steadily from £13 to £12 10 0, afterward recovering up to £13 10 0.

Iron.—Cleveland dropped from £2 10 10 to £2 10 0, afterward recovering to £2 11 2.

Scotch Pig down from £2 16 10 to £2 16 3, following the same movements as Cleveland, closing at £2 11 2.

Hematite dropped from £3 3 10 to £3 3 0, afterward recovering to £3 4 5.

	£	s.	d.
Aluminium, ingots, in ton lots	75	0	0
Alumina, alum, lump, loose	5	12	6
Antimony, black sulphide powder, per ton	20	0	0
Borax, best refined crystal, per ton	16	0	0
Sulphate ammonia, f.o.b. Liverpool	12	2	6
Sal-ammoniac, firsts, delivered United Kingdom	42	0	0
Copper sulphate	19	10	0
Camphor, 1-oz. tablets		1	7
Caustic soda, 48 per cent, ordinary	5	10	0
Bleaching powder, 35 per cent	4	15	0
Shellac, standard T. N., orange spot, per cwt	3	18	0
Sulphur, recovered, per ton	5	0	0
Carbolic acid, liquid, 97/99 per cent, per gallon		1	0
Hydrochloric acid, per cwt		5	0
Creosote, ordinary good liquid, per gallon			2½
Naphtha, solvent, 90 per cent at 160° C		1	3
Rubber, Para, fine, per lb		8	8
Platinum, per oz	6	0	0
Zinc, sheet (Vieille Montagne)	27	10	0

London, March, 1910.

Synopsis of Metallurgical and Chemical Literature.

Smelter Fumes.

The Elimination of Smelter Fume has been a serious and important problem with Western copper and lead smelters. Permanent injunctions against their operation were obtained by the farmers of Salt Lake City, Utah, and it became necessary either to abandon the works or to evolve a scheme for the elimination of sulphuric and arsenious acids from the fume discharged into the air. How well the chemists and engineers of the United States Smelting Company have solved the problem is shown by L. A. Palmer in an article in the March, 1910, issue of *Mines and Minerals*. Not only have the courts permitted the renewed operation of the entire plant treating both copper and lead ores, but the company is recovering a valuable by-product in the arsenic which is refined and marketed. All of the fumes from blast and roasting furnaces, amounting to 350,000 cu. ft. per minute are suitably neutralized and bagged. The bagging of blast-furnace fumes, which carry the sulphur as SO_2, has long been practiced, but no attempt has hitherto been made to bag roaster fumes carrying SO_3. Pot roasters are used for sulphide ores, and reverberatory roasters for blast-furnace matte. The fume from the pot roasters is conducted through a flue to a certain point, adjacent to which is a brick furnace which is fired with slack coal over a layer of lime rock, and to which a small amount of zinc concentrates is added. The fume from this brick furnace passes into the flue mentioned and mingles with the fume from the pot roasters. The zinc oxide generated thus neutralizes the SO_3 in the roaster fume forming zinc sulphate, which is subsequently settled and bagged. The fume from the reverberatories is cooled during its progress through flues leading from the furnaces, and at a suitable point their SO_3 content is neutralized by the introduction of crushed lime into the flue. The bag house ultimately receives all fume and the gases discharged from its stacks are practically colorless and invisible. The bag house is of the usual type, and the bags are shaken by a device operated outside the building. The

dust from the roaster section is returned to the roasters, while the blast-furnace dust is sent to the arsenic plant after it has spontaneously ignited and burned to a loose sinter.

The arsenic plant consists of two Brunton arsenic furnaces and a refining furnace. Clean coke fires are the source of heat in each. The arsenic is volatilized as As_2O_3, and the lead sinters and is resmelted. The arsenic fume from the Brunton furnaces is settled in a long chamber, provided with baffle walls. It is withdrawn at intervals and sent to the refining furnace, which is of the reverberatory type. The settling chambers for this furnace are of the same type and design as for the Brunton furnaces. The product is crystalline and is ground to make a commercial product. The daily output is about two tons of refined arsenic assaying 99.87 per cent pure As_2O_3. Operators at the arsenic plant are compelled to wear a suitable change of clothing and to take a bath before going off shift.

Copper.

The Sampling of Anode Copper with particular reference to the silver content has been made the subject of an investigation at the Washoe Smelter, Anaconda, Mont., and the very interesting results are presented in the March, 1910, *Bulletin Amer. Inst. Min. Eng.* by William Wraith. A disagreement between smelter and refinery led to the investigation. The smelter method of sampling consists in "shotting" the molten stream of copper into water, by "batting" the stream at intervals with a wooden paddle. The shot is then dried, cleaned and screened through a 10-mesh screen of No. 8 wire to remove fines, the oversize being further screened through a 4-mesh screen of No. 20 wire to remove the coarse shot. The undersize of the last screen is taken as the sample. The method used at the refinery is one developed by Dr. Edward Keller, of Perth Amboy, N. J., in which every fourth anode of each lot is drilled, using a 99-hole template. The holes of the template are used in continuous order from 1 to 99, one hole to each anode, regardless of the lot being sampled. For example, if one lot required the drilling of only 90 holes, the next lot would commence with hole 91. These drillings for each lot are ground to pass a 16-mesh screen, the sample being later mixed and quartered. Finally, the quarter selected is screened through a 40-mesh screen, the oversize and undersize being weighed, and the ratio of coarse to fine being carried out in weighing up the assay ton for analysis. Still another method investigated was the "ladle-shot" method, in which a ladleful of copper is shotted by pouring the molten stream over a wooden paddle into water. Samples taken by these various methods were carefully assayed for comparison and to determine the possibility of segregation of the silver in (1) different parts of the flowing stream, (2) the cast anodes, (3) successive portions of a ladleful, and (4) in the "skulls" formed in the ladles due to the solidifying of a shell of copper.

In the smelter shotting method it was proved that there is reasonable uniformity of the charge during the pour, and that different portions of the stream give uniform silver assays. The use of a wooden paddle or of a hot clay paddle seemed to make no difference. In the refinery drilling method it was found that six anodes selected at regular intervals during the casting of a lot, each drilled in 99 places by the template, gave almost uniform assays. Methods of comparing assays from a random anode with those of the lot from which it came were also tried and gave concordant results. A table is given showing the very fair concordance of the shotting and drilling methods when carefully carried out. The investigation of the ladle method showed it to be the least satisfactory due to the segregation of silver toward that portion of the metal solidifying last. The ladle-shot samples gave considerably higher silver results than the furnace shot samples. The "skulls" were impoverished in silver. Successive samples poured from the same ladleful showed a regular increase in silver content from first to last with the "skull" much lower in silver. Under these conditions each of the ladle samples was higher in silver than the smelter shot sample. A ladleful allowed to cool completely showed a remarkable segregation in the top of the button. The conclusions reached are that the shot sample taken from the furnace stream, and the drilling sample taken according to Dr. Keller's template, will check and give the true silver content of copper; and that the ladle-shot method gives higher results due to a segregation of silver toward the portion last solidifying.

The Behavior of Copper Slags in the Electric Furnace has been investigated by Lewis T. Wright, who presents the tabulated results in a paper published in the March, 1910, *Bulletin Amer. Inst. Min. Eng.* Ferruginous copper slags treated in the electric furnace will yield metallic iron. The author has shown that the iron recovered will contain practically all of the copper, gold and silver and more than 50 per cent of the sulphur of the original slag. The buttons are hard and clean, but not brittle. The conditions of the experiments were as follows: Slag used, 700 grams to 900 grams; strength of current 70 amp to 80 amp; time of fusing 1 hour to 1 hour and 20 minutes; weight of metallic button obtained, 153 grams to 218 grams; percentage concentration, 22 per cent to 31 per cent. The slags used carried from 0.25 per cent to 0.55 per cent Cu; 0.18 oz. to 0.33 oz. Ag per ton and from $0.15 to $0.40 Au per ton. The buttons obtained carried from 0.82 per cent to 1.62 per cent Cu; 0.48 oz. to 1.4 oz. Ag per ton; $0.40 to $1.20 Au per ton, and 88 per cent to 94 per cent Fe. The slag resulting from the fusion carried from a trace to 0.1 per cent Cu; trace of 0.025 oz. Ag per ton; trace, Au and 5 per cent to 19 per cent Fe.

Tin.

Flotation Process for the Concentration of Tin Ores.—The Elmore oil flotation process, which we described in its latest development in our Vol. 6, page 187 (May, 1908), has recently found a new application to the concentration of tin ore. This is described in the article by E. S. Elmore, of the *Mining Journal*, of Aug. 28, 1909. There is an old tin mine in Dolcoath, Cornwall, where for many years past those parts of the workings have been abandoned which contain mixtures of the ordinary tin oxide (cassiterite) with sulphides of various metals, such as copper, zinc, iron and arsenic. Such mixed ores were valueless, because if concentrated on water-dressing appliances, the sulphides, or, at any rate, a large proportion of them, were collected with the tin oxide, and the resulting product was too impure for the tin smelter. For the past two years an Elmore vacuum plant has been in operation there with complete success. The complex ore, after crushing, is treated in the vacuum machine, with the result that practically all the sulphides are removed as a concentrate which is almost free from tin, and substantially all the tin is left in the tailings from the Elmore machine, these tailings being passed on to usual water-concentrating appliances for the recovery of the tin, which is found in practice to be practically free from sulphides. Some of the Dolcoath tin ores are free from impurities except copper sulphide. From this ore the vacuum process produces a high-grade copper concentrate, in some cases as high as 10 per cent of copper, which is readily salable to the copper smelter.

Following the success at Dolcoath the process is about to be applied to two other widely separated parts of the world viz., the Straits Settlements and South Africa. With the tin concentrates from South Africa a working trial of the Elmore process on about 35 tons of this material has recently been carried out, the results being given below:

	S %	Cu %	As %	Sn %
The impure tin concentrate treated assayed	1.69	0.26	2.91	72.38
The sulphide product assayed	20.57	2.78	33.0	3.64
The cleaned tin product assayed	0.12	0.03	0.14	74.0

The loss of tin in the sulphur product amounts to only 6 lbs. per ton of material treated, while 95 per cent of the total impurities have been removed.

Zinc.

Flotation of Zinc Ores.—Broken Hill ores have presented some of the most difficult problems in ore dressing and, as a result, a number of processes have had their origin in that district. Donald Clark, director of the School of Mines, N. S. W., describes in the *Eng. and Min. Jour.* for Feb. 26, 1910, a combination roasting and flotation process devised by E. J. Horwood for the treatment of some very finely divided and intermixed sulphides of iron, lead and zinc. The process depends upon the fact that when a mixture of these sulphides is roasted the iron and lead are converted into oxide and sulphate at a low temperature, from 300° C. to 400° C., while the zinc sulphide is unaltered. This partly roasted mixture is then subjected to a heated-acid oil-flotation process, whereupon the zinc sulphide floats and can be collected, while the gangue with the iron and lead will sink. With suitable mechanical equipment in the way of roasters, tanks, spitzkasten, etc., an initial product carrying 32.4 per cent Zn and 22.3 per cent Pb was made to yield a zinc concentrate carrying 50.8 per cent Zn and 0.3 per cent Pb, and a lead concentrate carrying 49 per cent Pb and 2.6 Zn.

Gold and Silver.

Cyanidation of Silver Ores.—The application of cyanidation to silver ores is most largely practiced in Mexico, where the system employed is generally well known. In striking contrast to the usual method is the treatment used at the Zambona mine in the Alamos district of Sonora. As described in the Feb. 26 issue of the *Min. and Sci. Press* by B. B. Kniffen, the unusual feature is in the extreme rapidity of the solution of part of the silver, and the peculiar advantage of concentrating the ore prior to cyaniding. The ore is an orthoclase felspar, brecciated and filled with calcite. The mineral is argentite, distributed throughout the gangue in all sizes down to minute particles. This mineral is easily concentrated on tables, so that 50 per cent of the silver content of the ore can be removed in this way. The tailing is then cyanided. A solution carrying 0.075 per cent free KCN is mixed with the ore in the ratio of 6:1, the pulp being treated in Dorr classifiers. The sand goes to tube mills, the discharge from which is returned to the classifiers which in turn discharge their overflow to Dorr slime thickeners The underflow is stored for delivery to filter presses as needed. In this manner the silver is extracted from the ore by merely concentrating and grinding in solution without agitation of any kind. The silver is reduced from 16 ounces to 1.3 ounces per ton.

As illustrating the sensibility of the ore to cyanide treatment the following table is given showing the reduced silver content at various stages of the treatment:

	SILVER, OZS.
Ore after concentration	8.30
After mixing with solution	4.48
Tube mill heads	5.52
Tube mill tailing	5.28
Discharge from Dorr thickener	2.72
After standing 6 hours	2.28
After a 6-hour agitation	2.18
After a 9-hour agitation	1.96
After an 18-hour agitation	1.82

For a short time the mill was operated without concentration and by cyanidation only. It was found necessary to use a solution carrying 0.15 per cent free KCN and to supplement the other treatment by a 30-hour treatment in Pachuca tanks.

Dredging Beach Sands.—An ingenious manner of overcoming the difficulties encountered in handling Nome beach sands is described in the March issue of *Mines and Minerals*. The richest portion of the beach could not be reached in the ordinary manner, as it was under tidewater. It could not be dredged in the ordinary manner, for sand and water would fill the excavation as fast as made and thus prevent reaching bed rock. Mr. E. L. Michot conceived the idea of a barge carrying a caisson which could be lowered to bed rock and through which small areas could be worked consecutively. As carried out the construction is about as follows: The barge is 75 ft. x 32 ft. over all. Through its center, longitudinally, is a well 50 ft. x 10 ft. Along each side of the well is a track carrying a head-frame mounted on trucks. The head-frame supports the caisson, which is of sheet iron and air-tight, and measures 10 ft. long, 10 ft. wide and 7 ft. high in its working chamber. The caisson is raised and lowered over the head frame by means of a hoisting engine suitably located at one end of the barge. A centrifugal pump connected with the caisson raises the sand to the deck of the barge where it is treated in a long sluice box. Pressure is maintained in the caisson equal to the outer pressure of sand and water, so that the centrifugal pump may more efficiently clean the bed rock. Gravel worth upward from $2 per cubic yard is being worked.

Chemical Engineering.

Decomposition of Soda in Steam Boilers.—It is not generally known that the ordinary soda which remains in boiler water from the softening process is apt to dissociate in the boiler and to render the water acid, owing to the generation of free carbonic acid. Lime and soda are added to water, to precipitate carbonates of calcium and of magnesium, and, further, magnesia hydrate, and in order to secure this object, a slight excess of soda has to be used. The excess must be sufficient, and yet as small as possible. This soda accumulates to a certain extent in the boiler, even when the boiler is blown off, and it is this soda which is decomposed. A case of this nature was brought before the Mannheim Branch of the Verein Deutscher Ingenieure last November. The firm of Messrs. C. F. Böhringer & Söhne, of Waldhof, noticed that their distilled water, obtained by the condensation of boiler steam, was acid. This acidity—it was not overlooked—disappeared in the course of time, and was correctly ascribed to carbonic acid. It was at first thought that some bicarbonate had found access to the boiler, in which it would be decomposed. The softener was, therefore, overhauled. When the trouble was repeated, it was suspected that the temperature in the water softener, 80° C., might not be sufficiently high to decompose all the bicarbonate; the temperature was hence raised to 100° C., boiling point, but the carbonic acid still appeared in the boiler water. After having been informed of these observations, Dr. Grünewald ascertained that Basch had proved some time ago that boiler water contained the soda, not all as carbonate, but largely as caustic soda, and that carbonic acid was given off in the boiler. Now, the presence of carbonic acid undoubtedly favors the rusting of iron, though it is not indispensable for starting the oxidation of iron in the presence of water, and this point, while not important, deserves some attention, because caustic alkali is regarded as preventing the formation of rust. (Lond. *Engineering*, March 11.)

White Lead.—By an act of the French Parliament, which took effect July 20, 1909, the manufacture and use of white lead for either interior or exterior painting and decorating is to be prohibited after five years from the date of the act. London *Engineering*, of Jan. 14, observes that this is the result of agitation started some years ago by the French Socialists, who made the most extravagant claims regarding the ill effects of white lead on the painters, causing a high death rate from lead poisoning. The bill against the manufacture and use of white lead was introduced in the Lower Chamber and was passed. The Senate sought to amend the act by providing indemnity for the manufacturers who were thus to be deprived of their business, but the Lower Chamber would not accept the amendment. The Minister of Labor was also against the amendment, thus establishing as a principle the right of Parliament to destroy private property without compensation. The entire procedure is against industrial progress. By observing suitable precautionary measures, the manufacture and use of white lead are no more dangerous than many other industries. In fact, there is said to be more lead poisoning among employees of accumulator and storage-battery works than among

painters, but no provision is made for the protection of the health of these men. The French law has evidently been passed wholly as a measure alleged to be in the interest of labor, and without regard to industrial progress and property rights. Engineers have already begun to seek for a suitable substitute for white lead as a pigment. Some experiments already carried out indicate that nothing will take its place satisfactorily and that zinc white is not at all adapted to certain uses where lead was formerly employed.

Specific Heat and Chemical Equilibrium of Ammonia Gas.—An interesting apparatus for determining the specific heat of gases is described by Prof. Walter Nernst in *Zeit. f. Elektrochemie*, Feb. 1. He used this for determining the specific heat of ammonia gas, which is of special interest for the calculation of the NH_3 equilibrium. For the molecular heat of ammonia he finds the following formula to hold true up to 700° C.:

$$C_p = 8.62 + 0.0020\,t + 7.2 \times 10^{-9}\,t^3$$

For the formation heat of ammonia he finds at 850° C.

$$Q = 30{,}220$$

and states that it should be correct within 1500 calories. On the basis of his new determinations he finds that the application of his new thermodynamical theorem to the ammonia equilibrium gives results better in agreement with the facts than when older and less accurate figures were used.

Gas-Washing Bottles.—A recent Faraday Society paper of Dr. N. T. M. Wilsmore (read Jan. 10) describes gas-washing bottles with a very slight resistance to the passage of a gas. Three forms are described. In one, a short side tube is joined to the inlet tube of an ordinary washing bottle and bent upward so that the gas bubbles passing through it carry some liquid with them, causing constant circulation of same. In a second and more efficient form, the gas passes through a length of 5-mm tubing, slightly inclined to the horizontal. Below this, and connected to its ends, is a bulb of liquid, so that here again the passage of gas through the tube causes a circulation of liquid. The third form described is a modification of the Richardson wash-bottle, in which the pressure is reduced by lengthening the nozzle through which the gas enters the washing bulb. (From advance sheets.)

Storage of Coal.—The results of an investigation into the deterioration of stored coal, now being carried on by the Technologic Branch of the United States Geological Survey, are given by H. C. Porter and F. K. Ovitz in the March, 1910, *Jour. Ind. and Eng. Chem.* Coal absorbs oxygen from the air during storage without forming CO_2 and the amount of oxygen absorbed accords approximately with the deterioration in heat value. Oxidation may be largely prevented by immersion in water. Methane is evolved from freshly mined coal in quantities of importance as bearing on mine explosions, but of no importance as a loss of fuel value. Outdoor tests are being conducted to determine the extent of the loss during storage in the open as compared with immersion under water. Laboratory tests are being carried on to determine changes in the yields of gas, tar, and ammonia, through deterioration in storage.

Enameling Cast Iron for industrial uses is discussed in a general way by Harold Holcroft in the *Jour. Soc. Chem. Ind.*, for Feb. 15, 1910. The object of enameling, the testing of enameled articles, the preparation and requirements of various enamels, and the principles of their application are the heads under which the subject is presented.

Electrochemistry.

Electrode Losses in Electric Furnaces.—In an appendix to his paper on electrode losses, just published in Vol. XVI of the *Transactions* of the American Electrochemical Society, Mr. Carl Hering gives briefly the results of his tests of carbon and graphite between 100° C. (cold electrode end temperature) and 900° C. (furnace temperature). Higher furnace temperatures had not yet been used in the tests. In general, the curves have but a slight slope at the highest temperature of the test (about 900°) and all appeared to tend toward becoming nearly horizontal. Hence the error would probably not be great to use these values even for somewhat higher temperatures until the actual determinations for these temperatures have been made. This would surely be far more correct than to use the older data. Reduced to a terminal temperature of 100° C. the values for about 900° C. furnace heat are as follows in the inch units specified in Mr. Hering's article in the October issue of our last volume (Vol. VII, page 442): Graphite $r = 0.000323$; $k = 0.740$. Carbon $r = 0.00148$; $k = 0.329$. They are presumably accurate to within a few per cent, and were determined from a number of readings taken with considerable care. For graphite, the electrical resistivity falls nearly rectilinearly from about 0.000330 at 390°, hence only slightly, the curve being nearly horizontal. The thermal conductivity falls faster, also nearly rectilinearly, being about 0.861 at 390°, but it seems also to tend to approach the horizontal. For carbon the electrical resistivity falls faster, from about 0.00166 at 360°; the curve is slightly concave upward and tends to become horizontal. The thermal conductivity rises quite fast from about 0.226 at 360° and also tends rapidly to the horizontal, being concave downward. The much-discussed ratio of the heat conductivities of graphite and carbon, therefore, turns out to be only about 2 to 1 instead of 8 to 1, 10 to 1 and 18 to 1 as have been claimed by others. This, at least, is the case for 900° C. furnace heat, and applies to the mean values under operating electrode conditions, and to properly proportioned electrodes.

According to these constants for 900°, the so-called "electrode voltage," for 1°, for graphite is 0.0447 and for carbon 0.0639. This means that the minimum loss for carbon is about 50 per cent greater than for graphite, and not slightly less, as has been thought before. On the other hand, the difference in cross-sections between carbons and graphites, for the same furnace and electrode length, is less than that determined from the older approximate figures, the carbons being now only 3.21 times as large as the graphites. There is another matter of interest brought out by these tests. The mean electrical resistivity (under electrode conditions) falls more rapidly for carbon than for graphite. This means that an appreciably larger half of the heat due to electrical resistance will be generated in the colder half of the electrode, hence nearer the terminal. This heat will, therefore, have a shorter path for escape to the terminal than if the heat were generated more evenly throughout, as with graphite. In other words, a carbon electrode, it seems, would be hotter a short distance from the cold terminal than one of graphite would. This, of course, is an unfavorable feature and may account in part for the increased minimum loss for carbon. It seems to explain why the "mean" thermal conductivity turns out to be so unexpectedly high.

Quite recently two interesting papers have appeared in the *Proceedings* of the Am. Inst. Elec. Eng. of March. One is a paper by Mr. Carl Hering on "the proportioning of electrodes for furnaces" (giving the results of his tests in numerous tables and diagrams), the other is a mathematical paper by Dr A. E. Kennelly "on the modifications in Hering's laws of furnace electrodes introduced by including variations in electric and thermal resistivity." Both will be abstracted in our next issue.

Composition of Electro-Deposited Alloys.—A recent Faraday Society paper of S. Field (presented Jan. 18) discusses the conditions which determine the composition of electro-deposited alloys. Continuing the study of the quantitative composition of electro-deposited alloys, those of copper and silver have been examined. For the electrolyte, a mixture of the double cyanides was again used. Ordinarily silver is more negative than copper, but the difference in their character is such as would not seem to indicate any difficulty in depositing them simultaneously from a cyanide solution. The composition

of the deposits was obtained by dissolving in nitric acid, and estimating the silver as chloride, except where the copper was obviously small, when that metal was determined. When a copper coulometer is in circuit the ratio (R) of the weights of alloy and copper deposited in the coulometer gives a figure from which, under ideal conditions, the composition of the alloy may be determined thus:

$$\text{Percentage of silver} = \frac{241}{R}(R-2)$$

Experiments soon proved that simultaneous deposition only occurs within very narrow limits. Thus when the metals in the solution are in the proportion of even 1 equivalent silver to 34 equivalents of copper no deposition of copper occurs under ordinary conditions of current density, but when in the proportion of 1 equivalent silver to 100 equivalent copper, simultaneous deposition proceeds more readily. To examine the effect of such varying conditions as current density, free cyanide, and temperature, the following conditions were necessary:

1. Large volume of solution, and, therefore, little change of composition during each experiment.

2. Constant addition of silver to maintain a uniform content of that metal.

3. Solution containing not more than 0.4 grain silver per litre.

Under such conditions it was found that the percentage of the more positive copper varies as follows:

1. Percentage of copper increases with C.D.

2. Percentage of copper decreases with rise of temperature of solution.

3. Percentage of copper decreases with additions of free cyanide.

Thus the results follow the usual course as previously shown for brass, and the two metals become more widely separated in the presence of potassium cyanide. In no case were the deposits of any value physically, except where the silver very largely predominated. (From advance sheets.)

Rapid Electrodeposition of Metals.—A recent Faraday Society paper (read Jan. 10) of Dr. F. Mollwo Perkin and Mr. W. E. Hughes described a rotating cathode employed for rapid electrodeposition of metals. It consists of sheet platinum spun up so as to form a narrow thimble, the upper end being open and having a stout iridium wire fused to it. This electrode, which has an active surface of 16.3 cm, is rapidly rotated within a cylindrical platinum gauze anode. In other cases, particularly when graded potential methods are employed, a spiral anode of platinum is rotated within the gauze cylinder, which then functions as cathode. In the potential measurements a cylindrical glass funnel with a glass tube sealed in the side is employed for holding the electrode.

Dr. Henry J. S. Sand pointed out that the form of electrode ultimately found most suitable by the authors very nearly approached the one used by him, and if the investigation had been extended to this a further improvement would have been found. He considered the authors' electrodes were inferior in smallness of resistance to the electrolyte, stirring efficiency, and ease of manipulation. His electrodes, again, could be used with ordinary beakers and far less wash water was required. He had discarded the use of NH_4NO_3 solution as a connecting liquid, as on one occasion the auxiliary electrode gave completely wrong results. Dr. F. M. Perkin, in reply, said the electrodes described in the paper were simpler and less costly than Dr. Sand's. He preferred the tap-funnels to beakers, and found no difficulty in raising the temperature of the electrolyte before pouring into funnel. (From advance sheets.)

Electrochemistry of Organic Compounds.—A very useful summary of recent progress made in organic electrochemistry based on papers printed between May, 1907, to September, 1909, is given by Prof. K. Elbs in *Zeit. f. Elektrochemie*, Feb. 15. The article deals with the electrolysis of organic compounds and electrochemical reductions.

Recent Metallurgical and Chemical Patents.

Iron and Steel.

Treatment of Iron Before Tinning or Galvanizing.—Tinned plates are frequently imperfect by reason of the presence of minute uncovered areas, perforations, or so-called "pinholes," in the coating metal, such imperfections being highly detrimental by reason of the tendency of the coated metal to become corroded or pitted at these points or areas. While all surfaces of iron or steel which have been thoroughly cleaned and freed from grease or oxide are very susceptible to oxidation or corrosion, such imperfections are particularly liable to occur in plates which have been subjected to an acid pickling process, and their presence may in this case be partly or wholly attributed to the persistence in metal to be coated of acid conditions arising from the baths employed for the removal of the scale. According to a method patented by Mr. Henry Howard, of Boston, Mass., the iron or steel articles are first freed from scale by the usual methods, as by immersion for a suitable time in an acid pickling bath. The articles are then immersed, preferably after a brief washing or rinsing with water or a suitable alkaline solution, in an alkaline electrolyte, being connected therein as the cathode. As electrolytes, solutions containing a carbonate or hydroxide of an alkali are must suitable. Since the sheets under treatment are the cathodes, caustic soda is formed at them and hydrogen evolved. By this procedure the cleaned surfaces are rendered far less liable to corrosion, any residual acid conditions are quickly eliminated and the metal may immediately receive a coating of tin or zinc by immersion in the molten metal in the usual way. (949,575, Feb. 15, 1910.)

Utilizing Steel Scrap.—Steel scrap, rich in silicon, etc., is usually treated by melting it in a furnace in which its silicon or other alloy content is burned away and goes into the slag and becomes a waste product. Mr. J. P. Woods Beckman, of Parnassus, Pa., takes the scrap containing silicon or other alloy content and heats it in a closed furnace whose temperature is below the temperature required to fuse the scrap in such a manner as to prevent any substantial oxidation, bringing the scrap up to a white heat. He then adds to the scrap molten steel which has been melted in a separate furnace, and thus utilizes the excess heat of this steel to melt the scrap, which has been preheated to a point slightly below its melting point. In this manner he is able to utilize practically all of the alloy content of the scrap without loss and at a very slight additional cost. (947,031, Jan. 18, 1910.)

Electrolytic Iron.—Electrolytic iron is produced from ores by the following process of Mr. Alexander S. Ramage, of Newark, N. J. (the patent being assigned to the Electro Steel Company of Canada, at Welland, Ont.). Magnetite containing titanium is heated in a rotary cylinder in presence of water-gas with an excess of hydrogen at a temperature below the melting point of iron until the iron content is reduced to a metallic sponge. The reduction of the iron occurs at a dull red heat, whereas the titanium oxide is unaffected by this treatment. The iron powder, or sponge, containing unreduced titanium oxide, but substantially free from carbon, is discharged into air-tight vessels, where it is cooled, and is then electrolytically refined in an electrolyte containing ferrous sulphate and ammonium sulphate, together with some organic material, such as glycerin or glue. The cathodes are sheets of iron or revolving tubes. The current density is 10 amp per square foot and the temperature 140° Fahr. The cathodes are heated to expel the occluded hydrogen. The titanium oxide passes into the anode sludge in the electrolytic vats. (949,004, Feb. 15, 1910.)

Gold and Silver.

Filtering Device.—Continuous filtration of slimes is the subject of a patent granted to George Ridgway, of Kalgoorlie, Western Australia, who has perfected a mechanical appliance for the purpose mentioned. The process is continuous, but is

quite different from any of the half dozen or more continuous slime processes recently devised. The inventor retains the "basket" type of filter, and alternately lowers and raises the baskets into exhaust, washing and discharge vats in his machine. The construction will be apparent by reference to Figs. 1 and 2. The annular trough is divided into sections *A*, *D*, *E* and *F*. The first is filled with the pulp to be filtered, the second with water or wash solution, the third devoted to drying and pumping, and the fourth to the discharge of the cake. The various sections are divided by suitable partitions over which the baskets are raised and lowered in their continuous path around the entire tank. The filter frames are made of iron pipe bent into a V-shape and covered with a suitable filtering medium. All of the frames in a basket are properly connected. On each set of filters is a device for engaging a track at certain intervals around the tank, whereby they are lowered and raised at the proper time. Agitators keep the pulp in a homogeneous condition during filtration. When the machine is started all of the filters in tank *A* are in connection with the vacuum apparatus and a cake of slime is formed on the filter, while the

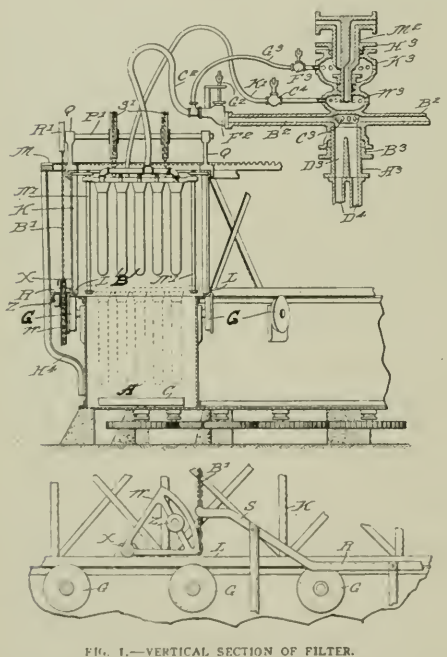


FIG. 1.—VERTICAL SECTION OF FILTER.

liquid is conveyed to a suitable chamber. As each set of filters approaches the partition between tanks *A* and *D* it is automatically raised from *A* and again automatically lowered into *D*, where the cake is washed during the progress of the filter through that tank. Arriving near the partition between *D* and *E*, the filter is again raised and kept in this position, with the vacuum still on until it arrives over tank *F*, where it is lowered and the cake discharged by compressed air admitted to the inside of the filter frame, and by a water jet directed against the outside. The filter is then ready to receive another cake on being automatically lowered into tank *A* containing the agitated pulp. (945,193, Jan. 4, 1910.)

Agitation.—The agitation of pulp or solutions in the treatment of ore by the cyanide process is effected by J. E. Rothwell and R. P. Akins, of Denver, Col., in a cylindrical tank with conical lower portion by means of pumping. The solution supplied to the pump comes from an overflow at the top of the tank and thus contains no large particles of pulp to wear the linings and working parts of the pump. The discharge from the pump enters the apex of the conical bottom, and carries with it any sands which may have settled at that point. The path of the discharge is upward through a vertical pipe placed in the center of the tank, from which it issues against a baffle or distributing plate. This causes the pulp solution to spread out evenly over the area of the tank, the heavy particles settling to the bottom and the fine slime again overflowing to the pump, to be forced upward through the

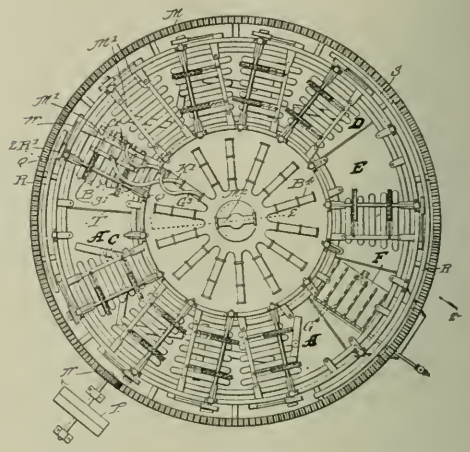


FIG. 2.—PLAN OF PLANT.

central tube, carrying the settled sand with it. A series of tanks may be used for continuous agitation and overflowing, the overflow from one tank to the next taking place through a special shape vane at the top of the tank which cuts out a uniform part of the tank content and carries it to the next tank. (951,742, March 8, 1910.)

Sand Treatment.—While the above two patents referred, one to the treatment of slimes only and the other to a mixed charge of sand and slime, a third interesting patent of a similar nature is that granted to Charles E. D. Usher, of Johannesburg, South Africa, for the treatment of sand only. Such material has hitherto usually been treated in vats by percolation, which is a slow process, and it is the design of the inventor to provide a more rapid means of treating this material. This is effected by providing in the bottom of a vat a number of radial perforated pipes connected with a central chamber. The ends of all the radial pipes are curved in the same direction with the circumference of the vat. A charge of sand having been placed in the vat, a solution of cyanide, for example, is forced in through the pipes and out of the perforations. The action of this solution is to stir the sands and to keep their surfaces bright and clean by attrition. The mass finally reaches a state of homogeneous semi-suspension. As soon as a sufficient quantity of solution is forced through the charge to collect in a layer at the top of the vat, the pump is disconnected from the sump from which the solution was being drawn, and connected with a decanter which rests in the upper surface of the liquid in the vat. The supply of solution is now continuously circulated through the mass of sand being injected through the radial pipes at the bottom and drawn off through the decanter placed in the clean layer at the top. The inlet of the decanter is usually so arranged that it draws in a quantity of air with the solution, thus providing an efficient aeration of the

mass in the vat. The process of solution and aeration is continued until the values are dissolved from the sand, whereupon the solution is drawn off and conducted to a suitable tank. The sands are then washed in the same manner as they were leached. (949,455, Feb. 15, 1910.)

Antimony.

Treating Arsenical Antimony Ores.—H. L. Herrenschmidt, of Paris, France, proposes to produce a clean antimony regulus from arsenical antimony ores by a combination wet and dry process. A preliminary roast of the coarse ore at a temperature of about 400° C. is the first step. The volatile oxides of arsenic, antimony and sulphur given off in this step are condensed for later use. The fine ores are not roasted, but are mixed with carbonate of soda and lime, with soda waste to which has been added carbonate or sulphate of soda, or with sodium sulphide. This mixture is then boiled with water under a pressure of four or five atmospheres. This operation forms antimoniates and arsenates of soda, or sulphoantimoniates and sulphoarsenates of soda, or all four. The liquor containing these salts is drawn off and is now caused to react with the oxides formed in the roasting operation before mentioned. Oxides and sulphides of antimony comprise the precipitate, while the arsenic dissolves in the solution of the arsenate or sulphoarsenate of soda. The antimony oxides and sulphides are collected and reduced to a clean regulus free from arsenic. Any antimony left in the liquor containing the arsenic may be precipitated by the sulphur dioxide from the roasting process, or by any other suitable means, while the arsenic remains in the solution. The inventor describes special apparatus for carrying out the process. (948,545, Feb. 8, 1910.)

Zinc.

Smelting Zinc Ores.—The recovery of zinc from lean zinciferous material, such as zinciferous slag or lean ores, may be effected by fine pulverizing and mixing with coal dust, briquetting the mixture and then charging these briquets into a furnace with small coal or coke. Zinc is volatilized and the other materials are slagged off. The object of a process patented by Hermann Pape, of Hamburg-Billwärder, Germany, is to eliminate as far as possible the necessity for grinding the material. He has determined that the quantities of reducing material contained in the briquets and the quantity of small coal or coke fed into the furnace with the briquets is more than sufficient to effect the removal of zinc from the briquets. Consequently he has found it possible to add to the ordinary furnace charge a considerable proportion of the zinciferous material in lumps, thereby avoiding the expense of crushing and briquetting. The manner of charging the furnace is, first, small coal or coke, second briquets of zinciferous material with carbon, third lumps of the zinciferous material. (948,740, Feb. 8, 1910.)

Alloys.

Improvement in Copper Alloys.—The manner of introducing certain other constituents into brass for the purpose of increasing its tensile strength and ductility is the basis of a patent granted to S. C. Peck and W. R. Hodgkinson, of London, England. Small proportions of chromium, manganese, nickel, cobalt or zinc may improve the quality of the alloy. The preferred manner of introducing these elements is by the addition of the phosphate of the metal with a reducing agent such as carbon, to a small quantity of copper or one of its alloys. This gives a metallic solution of the phosphide of the metal to be added, and this metallic solution may then be used to alloy molten brass in the desired proportions. In the second operation some of the zinc in the brass is volatilized as zinc phosphide, leaving the brass alloyed with the metal added. An actual test was as follows: 2600 grams of brass (Cu. 70 per cent, Zn. 30 per cent) were melted in a crucible with 200 grams of a mixture of equal parts of chromium phosphate and powdered charcoal. The temperature was about 1000°C. The addition of charcoal and chromium phosphate was repeated twice at intervals of an hour. After four hours heating the cast was made and this metal containing chromium phosphide was used to improve the qualities of brass as follows: 1516 grams copper and 655 grams zinc were melted together and to the melt was added 82.5 grams of the chromium phosphide brass. The resulting metal tested 19 tons per square inch tensile strength, and showed an elongation of 72 per cent on two inches. The same brass without the chromium phosphide added showed tensile strength of 14.4 tons per square inch and elongation of 38.5 per cent on two inches. (943,159 Dec. 14, 1909.)

Paper Mill Liquor.

Regeneration of Spent Liquor of Paper Mills.—The economical treatment of the spent liquor of paper mills for the recovery of the alkali and resinous matter contained forms the basis of a process devised by Isidor Kitsee, of Philadelphia, Pa. Heretofore, it has been the custom to concentrate the liquor to a syrupy consistency, after which the resinous matter is burned or carbonized. This residue is again dissolved in water, the carbonized mass filtered out and the caustic recovered. The process not only involves the loss of the resinous matter, but is inefficient in the recovery of the alkali. To overcome the objections to this process, the inventor causes the spent liquor to be electrolysed in a diaphragm cell. The spent liquor is placed in the anode compartment of the cell, while the cathode compartment contains water to which a little sodium chloride has been added. Through the action of the current, the sodium chloride is decomposed, the chlorine going to the anode compartment, while caustic soda is formed in the cathode compartment. The presence of the chlorine in the anode compartment acidulates the solution and causes the resinous matter to separate and coagulate. The result of the operation is the concentration of the alkali in the cathode compartment and the accumulation of the resin in the anode compartment. If little resinous matter is contained in the spent liquor and the chlorine is generated too abundantly, it may be led from the closed compartment to a tank of water, thus forming a suitable bleaching agent. (942,207, Dec. 7, 1909.)

Various Metallurgical Patents.

Roasting and Sintering Ores.—Metallurgists continue to give considerable attention to the preparation of fine ores prior to charging in the blast furnace. One of the latest patents on a process of this kind has been granted to Walter G. Perkins, of London, England, and Mark L. Requa, of San Francisco, Cal. The process resembles somewhat those patented by Dwight and Young in that there is a perforated hearth on which the charge is placed, the hearth lying above funnels and a tube connected with the exhaust apparatus. There is a marked difference, however, in that the hearth is stationary instead of movable as in the other patents mentioned. The ignition of the charge is effected by a blast of gas directed onto the surface of the charge under a hood. The hood is movable and as soon as ignition has commenced at any one point of the hearth, the hood is moved to another area. The exhaust continues until the entire charge is sintered. The hearth is hinged along one of its sides and in order to discharge the sintered mass the hearth is moved to an inverted position on its hinge and the entire sinter falls on to a perforated or slotted grating, through which the fine material falls, while the large lumps are discharged at a point convenient for smelting. (951,199, March 8, 1910.)

Still another sintering device is the subject of patent granted to George G. Vivian, of Tintic, Utah. This furnace is of the reverberatory type, and is operated by hand. Ore is fed in hoppers placed along one of the long sides of the furnace, the hoppers being water-jacketed. The furnace operator rabbles the mass over the hearth of the furnace and as soon as it becomes sintered withdraws it onto the cooling floor. To control the operation of the furnace means are provided for admitting cold air above the charge. (950,798, March 1, 1910.)

Joseph Weatherby, of New Cumberland, Pa., has been granted a patent for agglomerating fine iron ores. In a rotary kiln the fine material is not heated to the point of agglomeration until it reaches a point near the discharge end, where coke or charcoal is introduced and mixed with the ore. As the mass agglomerates at this point the carbon particles are consumed, leaving the ore in a porous condition suitable for smelting. The refuse coke dust discarded at the blast furnace has been successfully used. (949,387, Feb. 15, 1910.)

In order to overcome the difficulties arising from the formation of annular rings in kilns, August Brăutigam has made use of a blast of air directed at one point of the ring until it disintegrates. The operation may be performed while the furnace is either stopped or in slow motion, the effect being to so chill the ring at a certain point that it will crack, due to contraction. When the furnace is again placed in operation the balance of the ring falls off and mingles with the ore. (13,062, Dec. 28, 1909.)

An improvement in roasting furnaces of the superposed hearth type has been made by Utley Wedge, of Ardmore, Pa. The improvement relates to the construction of the furnace whereby the furnace arches and their superposed hearths are supported on an outer structure, so that the enclosing wall of the furnace is relieved of weight and strain. Economical construction and greater facility in making repairs constitute the main features of the improvement. (946,178, Jan. 11, 1910.)

Cooling Roasted Ore.—The economical cooling of roasted ore is of importance as an intermediate step in some metallurgical operations. Frank D. Baker, of Denver, Colo., accomplishes this in a rotary cooler with cone-shaped ends, revolving in a vat of water and having a spray of water directed along its top. The buoyancy of the water is of assistance in permitting the use of a longer cylinder than might otherwise be practicable on account of the necessity of bearings between the extremities of the cylinder. The cooler is rotated by any suitable power connection. At its discharge end it is equipped with internal spirally arranged blades which advance the ore to the opening. (943,294, Dec. 14, 1909.)

Electrostatic Ore Separation.—A patent has been granted to Frederick G. Cottrell, of Berkeley, Cal., on improvements in effecting interchange of electric charges between solid conductors and gases. One of the interesting applications of the principle involved is the electrostatic separation of ore. A metallic revolving roller, connected to earth, receives a steady supply of finely divided ore from a suitable hopper. In front of the roller, parallel with its axis, and slightly above its center, is a pubescent electrode in the form of a piece of cotton-covered magnet wire. This takes the place of the metallic discharge points usually used in machines of this type, and is the main feature of the invention. The inventor claims that all the advantages claimed for metallic points are obtained in higher degree by the use of fine filaments or hairs of a textile substance, even though it may not ordinarily be considered a good conductor. For example, such materials as asbestos, cotton, wool and silk are used. Suitable receptacles are provided beneath the roller to catch the conductive and nonconductive particles. (945,017, Jan. 11, 1910.)

Various Electrochemical Patents.

Copper Plating.—Dr. E. F. Kern patents a new electrolyte for copper plating. It is a solution of copper fluosilicate and ammonium fluosilicate (or,in some cases, aluminium fluosilicate) in water, and a further improvement may be obtained by the additional use of ammonium fluoride with an alkaline tartrate. For plating iron, steel, or zinc with copper, the bath preferably consists of 100 parts water, 12 copper fluosilicate, 8 ammonium fluosilicate, 6 ammonium tartrate, and 5 or more parts of ammonium fluoride. The bath is maintained neutral or slightly alkaline by means of ammonia. For coating brass, copper "or alloys of other metals," the bath preferably consists of 100 parts of water, 12 copper fluosilicate, 8 ammonium fluosilicate, 8 ammonium fluosilicate, 0.06 gelatine or tannin, and 5 or more parts of ammonium fluoride. In this case the bath is acidified by free fluosilicate acid or is maintained neutral by employing aluminium fluosilicate instead of ammonium fluosilicate. When the bath is used in acid condition, it is advisable to add gelatine or tannin to the bath in the proportion of one part of solid gelatin to one part of the concentrated extract of tannin to 2000 parts of the bath. A copper anode is employed. (946,903, Jan. 18, 1910.)

Nitric Acid from Air.—Mr. Francis I. DuPont, of Wilmington, Del., who has paid considerable attention to the production of nitric acid by electric discharges through air, has been granted another patent for a furnace for this purpose. It is shown in Fig. 3. The air is passed through a silica tube *a* within which are the electrodes *b*, between which the arc *k'* is formed. The silica tube is surrounded by the glass tube *h* through which a cooling liquid is passed from *j* to *k*. The field magnets or rings *l* are energized by a polyphase current, thus producing a rotary magnetic field which causes the arc to

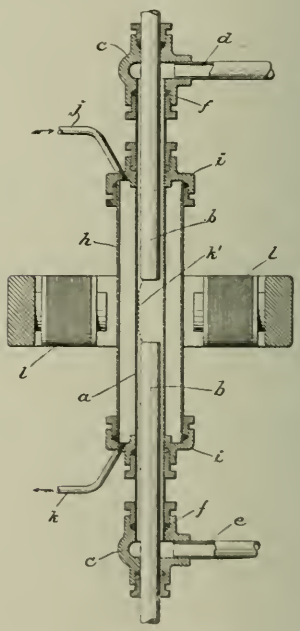


Fig. 3.—Nitric Acid from Air.

rotate around the inner surface of the silica tube. Air is forced through the passage *e* into the zone between the electrodes; it is heated by the rotary arc (so as to form nitrogen oxide), but as soon as the rotating arc passes from any portion of the air, this air is immediately subjected to the cooling action of the walls and its temperature is so rapidly reduced as to prevent the formed nitrogen oxide from being again largely decomposed. (950,703, March 1, 1910.)

Ozone.—Mr. Robert W. Rice, of Cleveland, Ohio, patents an ozonizer in which glass plates serve as dielectrics, being placed between aluminium plates. These are punched so as to provide a series of discharge points. A transformer is provided in the lower part of the apparatus. (930,347, Feb. 22.)

A large number of electrochemical patents had to be reserved on account of lack of space for our next issue, in which all electrochemical patents will be brought up to date. Several abstracts of patents of an electrochemical nature will be found above among the patents on Iron and Steel and Copper.

Smelter Fumes and Damage Suits.

By R. G. Burton and Taft Reed.

A most interesting conflict of interest between the agricultural and lumbering industries on the one hand and the metallurgical and chemical industries on the other is now being brought into more and more prominence. Ever since sulphuret ores have been smelted there have been complaints from the surrounding agriculturists, and the early law cases of England are replete with decisions on the respective rights of the parties. In the majority of cases the issue turned upon the strict right of the agriculturists and the preservation of their property against even a technical invasion by the gases and smoke from the smelters. The same conditions have prevailed for many years in Germany, and now in America the question is becoming one of greater importance than ever before. The clash has become almost a battle for very existence.

The increase of population has greatly increased the value of foodstuffs so that the agriculturist has the better right—where actually injured—to bitterly complain; where the forests a few years ago were regarded as practically worthless, having in same cases really nominal values, they have now suddenly become enormously valuable on account of their fast disappearance and a real injury to them represents a real loss of value. On the other hand, the great increase in population has given a great impetus to all metallurgical and chemical developments, with the result that any interference with these industries must inevitably mean not only serious loss to the operator, but to the public as well. Civilization demands electricity, electricity demands copper, as expanding industry demands steel, which in turn requires coke, and as the copper furnace or coke oven is interfered with in its operation, from whatever cause, so civilization must be inconvenienced and must in the end sustain a loss. What is true of the loss or injury occasioned in the manufacture of copper or coke, is, of course, true of all those commodities which in their manufacture cause to be thrown off gases deleterious to vegetation, so that on the side of copper and coke are arrayed zinc, lead, wood-pulp, commercial fertilizer and a dozen others.

"But," says the farmer or lumberman, "I am at least entitled to be paid a fair price for my timber damaged or my crops destroyed." True, but the question will arise, "Where is the dividing line. What has been killed or injured by the gas or fumes and what by the weather or other natural causes?"

It is just at this point that the real trouble begins. It is no longer a fight between agriculture on the one side and industry on the other. The matter has narrowed down to one of lawyers and courts, and most important of all, to the question of evidence. The plaintiff may be honest, his crop or his woodlands may have been injured, but in practically every case, honest though he be, his injury, to his mind, has been done solely by his neighbor who mines and smelts and cares nothing for farms, natural causes of death are lost sight of, prospective crops would have been enormous, the stand of timber and its value are prodigous and the damage to it is most pitiful to see. And those orchards! They were not infested with scale and blight and diseases innumerable and they received the best of care and attention and were of the hardiest and most valuable varieties.

So long as the crops, the orchards and the timber were of little value, the evidence required usually went only to the point of showing the fact of death and the fact of the proximity of a smelter. But now that the values of crops and particularly of timberlands have increased and will increase so enormously, the question of cause of injury will become more and more vital. What was once in a haphazard way almost conceded is now coming to be analyzed, and the true conditions so long really incomprehensible to the courts, and to the defendants as well as to the plaintiffs, are coming to be understood. Scientifically trained men have been investigating the conditions and have studied those natural causes either wholly responsible or acting as contributing factors in the damage complained of and the following surprising condition has been found: That in almost every case insects, fungi, and worst of all, fire and dessication, have been working a great part of the injury for which the smelters have been blamed and for which they have had to pay handsomely.

Damage to the forest by insects is caused either by those which kill the tree outright, or by those which weaken the tree and reduce its commercial value. Millions of feet of timber are killed each year by bark-beetles of the genus *Dendroctonus,* which attack conifers, in certain parts of the country destroying whole forests. To another beetle, *Agrilus bilineatus,* is attributed the death of most of the mature chestnut trees in the United States. One of the Scolytids, *Scolytus quadrispinosus,* has caused most of the death of hickory in the past few years in the southern Appalachians. These are only a few prominent examples of insects causing serious damage to the forest trees and could be greatly multiplied.

The ravages of fungi are seen everywhere and in all parts of the country. A striking example is the "white pine blight" which causes a discoloration of the leaves exactly similar to that caused by smelter fumes and like them causes the death of the tree. Another fungus is *Diaporthe parasitica,* which is now killing off the chestnut in the vicinity of New York City.

Worst of all, however, is the damage, direct or indirect, done by the forest fires which are more or less prevalent everywhere. In the Appalachian system, for instance, the soil is gradually but surely reaching a condition of barrenness due to the constant firing at the wrong time of year by the farmers under the mistaken belief that the value of the land for grazing will be enhanced. The common idea that fires must be severe in order to do serious damage is erroneous. A light ground fire running over the ground year after year dessicates the soil by destroying the ground litter and humus, paves the way for the inroads of insects and fungi and for the beginning of erosion; in short, disturbs the normal balance of things in the forest to such an extent that the trees are killed just as surely as though girdled by severe fires.

In view of the foregoing facts, it will be readily seen that the best and only defense for the smelting company to adopt is based on the collection of evidence in the field, section by section and township by township, which will show the relative damage done by smoke, by fire, insects, fungi and other causes. The criteria of smoke damage in such a field examination are at once simple and complex. The different causes of death, including smoke, and how each acts, may be perfectly understood, but it is almost always the case in the forest that death is due to no one cause, but to a combination of several causes. Thus, fire may scar a tree and afford entrance to insects or fungi, which in turn so weaken the tree that it falls a victim to smelter fumes, which it would otherwise have resisted.

Over large areas perhaps no one cause can be fixed upon as acting alone, and this fact would at first seem to negative a defense along these lines. It will be plain upon brief consideration, however, that it is only necessary to show that there are other causes than smoke playing a large part and the burden of proof immediately falls upon the prosecution to show how much of the damage has actually been done by the smoke, and unless he can accomplish this, his case at once falls to pieces. The judge is justified in instructing the jury, as has been done in recent cases, that unless they can settle upon the actual amount of damage by smoke, beyond all reasonable doubt in their own minds as to the accuracy of the figures that they thus fix upon, that they shall bring in a verdict of nominal damages only. This method of procedure denies to no one his opportunity for recovery by law for any damage traceable to the influence of smelter fumes. But it forces the plaintiff to substitute facts for theories and a detailed examination of the conditions for exaggerated ideas of his injury. Not even the defendants themselves have usually any idea of how small a part of the injury they are perhaps responsible.

To sum up, the financial importance to the smelting company of collecting evidence in the field of every cause other than smoke, which has killed or damaged the forest, cannot be overestimated. What special form these other causes of death will take varies greatly with the locality and can only be determined by a field examination of the particular region in question.

Automatic Gas Calorimeter.

The automatic gas calorimeter, which is built by the Sargent Steam Meter Company, of Chicago, has recently been improved in various details. A description of the improved instrument should, therefore, be of interest.

Fig. 1 shows the complete calorimeter. It consists of a wet test meter in which the gas consumed is accurately measured.

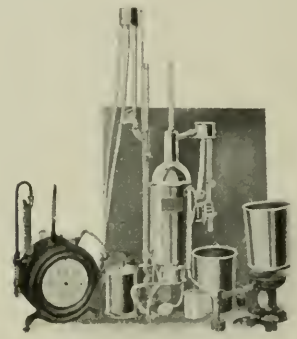


FIG. 1.—COMPLETE CALORIMETER.

From this meter it flows to a governor which maintains a uniform pressure of the gas at the burner. In the calorimeter proper the accurately measured gas is burned and its calorific value is manifested in the rise of temperature of measured quantities of water flowing through. From the calorimeter the heated water for each unit of gas burned is automatically discharged into one of the pails in which it is weighed on the decimal scales. The pounds of water, times its rise in temperature in degrees Fahrenheit, times the quantity of gas in cubic feet consumed gives the b.t.u. direct.

Meter.—Fig. 2 is a sectional elevation of the meter. The rotor is accurately balanced, and is mounted on a heavy shaft to which is attached the integrating train. The meter rests on three legs, each equipped with a leveling screw. A circular sensitive level permits of accurate adjustment. The water level is indicated in the glass gage by a pointer set in the laboratory of the makers, when the meter is calibrated, and when it is operated under similar conditions it will give absolutely correct results. A thermometer indicates the temperature of the gas metered and a manometer its pressure, both of which are necessary in reducing metered to standard gas.

One revolution of the needle equals one-tenth of a cubic foot and as the dial, made of white porcelain, is divided into 100 parts, the meter has a range before repeating from one-thousandth of a cubic foot to 100 cu. ft. A filling funnel and drain facilitate its manipulation. When the meter is used with the calorimeter, the electrical attachment closes a circuit for every tenth of a cubic foot measured, which through the dumping bucket automatically segregates the water whose temperature is raised by the unit of gas burned.

Pressure Governor.—In order to maintain a non-pulsating flame at the burner, a pressure governor is required between meter and calorimeter. The governor shown in Fig. 1 is of the inverted bell displacement type and can be adapted for any gas by the adjustable weights.

Calorimeter.—In the calorimeter every heat unit given out by the combustion is manifested in the rise of temperature of the water flowing through while a unit of gas is burned. It is mounted on three wide spreading legs, insuring stability, each of which is provided with a leveling screw so that it can be operated on a irregular surface.

Fig. 3 is a section of the instrument in which the inlet water, having a constant head at the wier *A*, the temperature of which is taken at *B*, passes down the tube *C*, and enters the calorimeter at *D*. Here the water is spread around the bottom of the instrument by the baffle plates *E* and *F*, and then flows upward around the tubes *G*, through which the products of combustion pass. The partially heated water on leaving the tubes spreads out over the dome sheet *H*, where it is heated by the hottest gases and then passes to the wier *K*, through the baffle plate *I* and around the thermometer *J*, where the outlet temperature is taken. From this wier *K* the water is discharged into the automatic tipping bucket, which delivers the heated water alternately to the copper pails.

The combustion of gas takes place in the central flue at *L*, the products of combustion passing upward to the dome sheet and then down through the flues in a contrary direction to that taken by the water. The gases are discharged at *M*, where the temperature is maintained the same as that of the surrounding air by the damper *N* and the quantity of water flowing through.

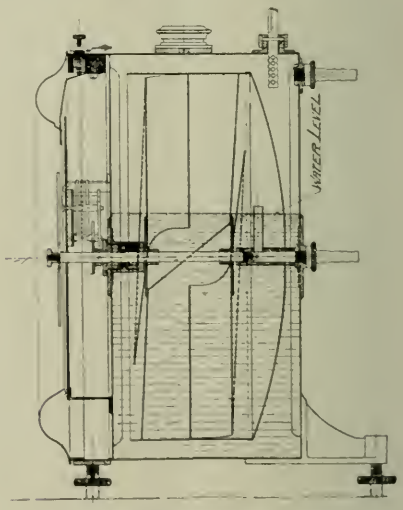


FIG. 2.—SECTIONAL ELEVATION OF METER.

Accessories.—The special Bunsen burner, adjustable for any gas, extends well up into the internal flue and the closed base prevents downward radiation. By means of a fixed mirror the operator can always see the color of the flame through the air admission opening. Two dry cells furnish the electric current for operating the releasing magnet and the weight of the water in the bucket tips it. The scales have a capacity of 10 lb., are graduated to a hundredth of a pound and when balanced with the empty pails indicate the net weight of water, whose temperature has been raised by the combustion of a 1/10 cu. ft. of gas.

With this calorimeter determinations may be made for every 1/10 cu. ft. burned, covering an interval of time from one to three minutes, and for an unlimited duration. With such a log, continuous curves of a calorific value of the gas may be plotted and an absolute record of the operation of the gas-generating apparatus may be obtained and preserved.

When extreme accuracy is required in reducing metered to standard gas, a special draft gage is used to get the pressure of the gas at the meter. It is used in connection with producers, furnaces, and ventilating apparatus.

The foreign matter in gas is determined mechanically when it is tested for its calorific value or by a separate operation.

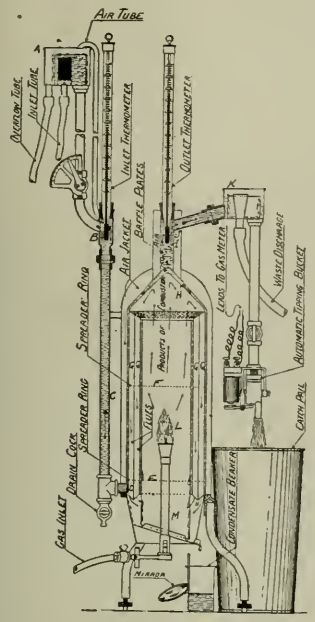


FIG. 3.—SECTION OF CALORIMETER.

In either case the unit of gas analyzed is determined by the meter. The tar in producer gas, the dust in blast furnace gas, the moisture in any gas or air, and the dust in the atmosphere is determined by special filtering apparatus, built by the Sargent Steam Meter Company.

Centrifugal Compressors for Oil Burning.

By S. A. Moss.

The use of crude or partially refined petroleum as fuel for industrial purposes is rapidly extending. Taking into account convenience and cleanliness, oil is a cheaper fuel than coal for many industrial purposes. For this reason oil is being used to an increasing extent for forging, hardening and tempering in blacksmith shops; baking cores and melting in brass, iron and steel foundries; smelting, melting and refining in metallurgical establishments, and for a great variety of other processes where high temperatures are desired.

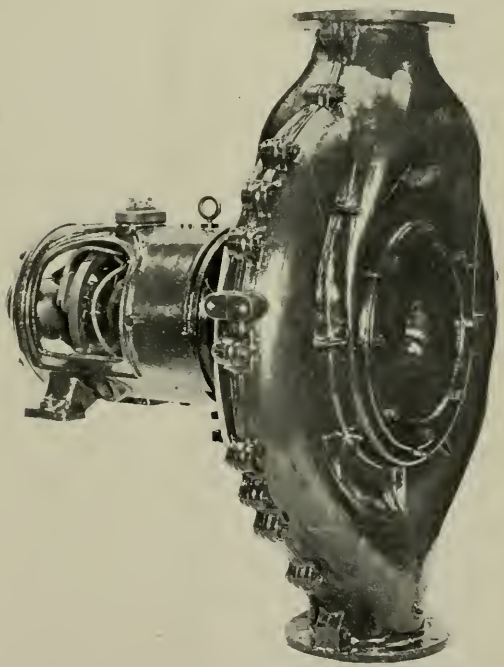
Oil is also used under steam boilers in many cases. However, space will not be taken for detailed discussion of this application.

The furnace usually consists of a chamber (often of very large dimensions), lined with fire brick, upon the floor of which is placed the object to be heated. The oil, together with some of the air for combustion, enters at one side from one or more jets, and the burning blast impinges on the object to be heated. Sometimes the chamber is rectangular, and the jets are directed immediately upon the object. In other cases there are bays or side chambers to give opportunity for complete combustion previous to the main chamber. There must, of course, be an outlet for the products of combustion after they have given up their heat. As can be seen, a furnace of this type is much simpler than one where coal is used as fuel, for with coal there must be a grate, a combustion chamber and an ash-pit. There is, in addition, the expense of firing, clinkering, and removal of ashes. Oil fuel is usually stored in tanks and delivered by pumps, and the storage and delivery is usually a simpler problem than that of coal.

So far as cost of a heat unit is concerned, coal is possibly a cheaper fuel at the present price of oil. However, taking into account all of the incidentals, the total cost with oil is in many cases much less.

Two kinds of oil are in use at the present time. One is crude oil just as it comes from the well, while the other is a residue remaining after some of the more valuable products have been extracted. The nature of this residue varies considerably at different places and at different periods.

In order that the oil may burn in a proper manner and in a reasonable space, without smoke, it must be intimately mixed with the air necessary for combustion. The oil is, therefore, reduced by a "burner" to very small drops, each surrounded by air for its combustion. This is called atomization. The drops must be of very small size, since a drop of appreciable size,



CENTRIFUGAL AIR COMPRESSOR.

even though it is surrounded by the entire amount of air necessary for its combustion, will proceed for a considerable distance from the burner, as a small ball of yellow flame, before it is completely burned.

The burners for producing the proper type of jet are of many shapes. There is always a pipe carrying oil under pressure leading through the burner. There is usually an additional pipe carrying compressed air or steam for the atomization. In the case of oil burners for steam boilers, the jet is usually steam, while for industrial purposes, the jet is usually compressed air. Some of the air for combustion is entrained or sucked in by the velocity of the jet issuing from the burner. In the case of steam atomization, the entire amount of air

must be supplied in this way. With air atomization the amount supplied under pressure and the amount entrained differ with the type of burner. There are many standard burners on the market and in addition many of the plants which burn oil have special devices of their own. The arrangements are, therefore, numerous and no general description can be given. There are the following general classes, however:

The compressed air may be supplied at a pressure from 15 lb. to 30 lb. per square inch. In such cases, the compressed air usually forms a comparatively small portion of that necessary for combustion, and the main air is entrained by the high velocity or sucked in by the chimney draft. In one such system, the main air is supplied by a forced draft given by a fan blower delivering the air through a pipe ending near the burner, at a comparatively low velocity.

In other cases, the compressed air is supplied at a pressure of from 1 lb. to 4 lb. per square inch. In these cases, probably the greater portion of the air is supplied, and a comparatively small amount is entrained. However, it is difficult to make a positive statement in this matter. The designers of the various burners usually have an idea regarding the amount of air to be supplied, although this idea is often very approximate.

The General Electric Company have a standard line of single-stage centrifugal compressors from 5 hp to 350 hp for pressures of from 1 lb. to 4 lb. per square inch, and these are very well adapted for use with oil burners. Multi-stage machines are also made for pressures from 15 lb. to 30 lb. per square inch, and these have also been furnished for the same purpose.

These compressor sets are self-contained and are direct-driven by Curtis steam turbines, induction motors or direct-current motors. They have no wearing or rubbing parts and consequently, no matter how long they are in operation, there is no falling off in pressure and output due to leakage. The pressure is steady and free from pulsations.

Notes.

Colorado Iron Works Company's Impact Screen is the subject of a pamphlet, 9-B, issued from the Denver office. The screen is designed to take the place of trommels and hydraulic sizers in certain work. Recent minor improvements make the device thoroughly dependable for hard work.

Vanadium steels are now made in a variety of types suited to special purposes. Vanadium toughens steel directly by solid solution in the carbonless portion, and indirectly by removing oxides. The American Vanadium Company, Pittsburgh, has set forth qualities and uses of vanadium steels in a booklet recently issued. Test tables and microphotographs show the strength and structure of the steels. To extend their market, they have also recently published interesting pamphlets in French and German entitled Les Aciers au Vanadium and Vanadium Stahl respectively. Interesting information is given on the heat treatment of vanadium steels, the methods of employing vanadium in iron and steel, and results. Copies of these nicely illustrated books are sent to interested parties on request.

Power from Exhaust Steam may be obtained through turbine engines and greatly increases the efficiency of the power plant. The American Regenerator Company, Chicago, has explained in a pamphlet recently sent to us the operation and use of its regenerators for heat storage between the exhaust of reciprocating engines and the turbines.

Metal Statistics are neatly presented in a small volume of that title issued by the American Metal Market Company, New York. Iron and steel, copper, tin, spelter, lead, antimony, gold and silver are considered.

International Exhibition of Railways and Land Transport.—The Argentine Republic will celebrate the 100th anniversary of its political emancipation by an industrial exhibition to be held at Buenos Aires, May to December, 1910.

Ferroinclave, a sheet steel reinforcement for concrete work, is illustrated and described in a new catalog issued by the Brown Hoisting Machinery Company, Cleveland, Ohio. Ferroinclave is a sheet steel with dovetail corrugations, and is used in the construction of roofs, floors, walls, stairs, bridges, tanks, coal bins, etc.

Pneumatic Separators.—By the proper arrangement of air currents, deflectors and chambers all light, powdered materials may be handled with uniform results and remarkable precision. It is the design of Osborne pneumatic systems to accomplish this. These systems are adapted to removing waste from grinding machinery, separating and grading crushed products, washing dust from fuel gas, and neutralizing the fumes from chemical operations. A catalog of the Griscom-Spencer Company, 90 West Street, New York, explains in detail the operation of the Osborne system.

The Mineral Production of Canada for 1909 has been compiled and is being distributed by the Minister of Mines, at Ottawa.

Fire Brick.—A catalog received from the Scioto Fire Brick Company, Sciotoville, Ohio, describes high-grade firebrick manufactured for various purposes. The analysis of the clay from which the brick are made shows it to be almost pure silicate of alumina.

Transite Asbestos Wood is a fireproof building material made by the H. W. Johns-Manville Company, New York. The material is mineral in character, but can be handled and worked just like wood.

The American Lava Company, Chattanooga, Tenn., manufactures machined insulating articles known to the trade as lava, but being in reality a talc which hardens upon being subjected to a high temperature. Quite a variety of insulators, gas burners, etc., are made in this manner.

Eimer & Amend, New York, have issued their 1910 edition of the catalogue of chemical apparatus. This is a very full and exceedingly interesting publication which should be in the hands of all users of chemical apparatus and chemicals.

The Ammonia-Cyanide Process for treating copper, gold and silver ores is a development by Western metallurgists who have organized the Ammonia-Cyanide Engineering Company, of San Francisco. The principle of the process is the leaching of ores, precipitating the dissolved copper by boiling out the ammonia in tight apparatus and then precipitating gold and silver by zinc dust or shavings in the usual manner. The ammonia boiled out of the solution is condensed for re-use.

The Control of Combustion in the boiler plant is the aim and object of the Sarco CO_2 recorders, made by the Sarco Fuel Saving & Engineering Company, of New York. A recent catalog describes their device for automatically recording the CO_2 in the flue gases. Other catalogs deal with the company's pyrometers, calorimeters, recording draught gages and recording steam meters.

The New York Metal Exchange has issued its statistical review of the domestic and foreign metal markets for 1910. The production and prices of tin, copper, lead, spelter, iron and steel, antimony and silver in the various markets of the world are given.

Water Gage Glasses break from contact with cold drafts of air or liquids while the glass is heated. A new German gage glass, known as "Sonderglass," is made by fusing two separate tubes into one, thereby giving increased strength and allowing for expansion and changes of temperature. A. W. Chesterton Co., Boston, are importers into this country.

Ozone. As an instance of the progress which ozone is now making in water sterilization it may be mentioned that the city of St. Petersburg, in Russia, has installed an ozone plant for the treatment of 10,000,000 gal. of water a day. The system of ozone generation of the Siemens & Halske Company is being used. Further progress is also reported from several German and Austrian cities. Ozone ventilators for the purification of air in rooms are stated to find increased favor.

Cupel Machine.—The Denver Fire Clay Co., of Denver, Col., has sent us a description of Iler's cupel machine, which with the improvements recently made, has a capacity of 500 cupels per hour. It is operated by foot power, and the density and thickness of the cupels are regulated by one small set-screw on the pedal. The Iler cupel machine is made in two sizes, one making 1¼-in. and 1½-in. cupels, and the other making, 1¼-in. 1½-in. and 2-in. cupels.

Liquid Chlorine.—While in former years the liquid chlorine used in this country for various chemical and metallurgical purposes was imported from Germany, various American manufacturers have taken up the manufacture of liquid chlorine during the last year. Among them are the Castner Electrolytic Company, the Hooker Electrochemical Company, the Electro Bleaching Company and the Goldschmidt Detinning Company. From a statement of the Electro Bleaching Company we learn that they are entering the market prepared to supply any quantity of liquid chlorine from 100 lb. up to carload lots—18,000 lb. minimum. It is shipped in steel cylinders or drums holding approximately 100 lb. each.

Concrete Mixer.—We have received from the Chain Belt Company, of Milwaukee, Wis., their illustrated catalog 39, giving details of construction of their chain belt concrete mixer. This is a "batch" mixer of the "non-tilting" or chute-discharge type. A power loader is employed instead of the standard charging hopper. While the Chain Belt concrete mixers are furnished with steam, gasoline or electric power as desired, steam is thought to be the most reliable and economical under general conditions.

Combination Induction Furnace.—The firm of Pilger & Neidhart, in Frankfurt-Bockenheim, Germany, is erecting a combination induction furnace of the Roechling-Rodenhauser type for the manufacture of steel castings from electric steel.

New Blast Furnace.—The new furnace of the Andrew & Hitchcock Iron Company, Hubbard, Ohio, was successfully blown-in on Feb. 19, this year. This is the first of the new furnaces to go into operation this year. It is the latest type of furnace, designed and built by Mr. Julian Kennedy, of Pittsburgh, Pa., and Mr. Arthur E. L. Dette, engineer in charge of construction. The William B. Pollock Company, Youngstown, Ohio, furnished the iron work, and the Stowe-Fuller Company, of Cleveland, Ohio, all the firebrick. Niles brick were used for furnace lining, National brick for the stoves, and Empire brick for pipes and connections. The furnace, stoves, and connections make one of the best and most complete blast-furnace plants in the Mahonning Valley.

Industrial Enginering and the **Enginering Digest** have been consolidated into one paper under the name *Industrial Engineering and the Engineering Digest.*

The **Schaeffer & Budenburg Manufacturing Company,** 963 Kent Avenue, Brooklyn, N. Y., have sent us their catalog on indicating and recording pressure and vacuum gages, ammonia gages, hydraulic gages, draft gages, etc. Among the gages described is a new draft gage which indicates a pressure or draft of 1/100 in. of water column. It is fitted with a large, easily read dial.

California Mining Bulletin.—The California State Mining Bureau has just issued a bulletin dealing with the mineral productions and mining laws of that state. There is also included a set of county maps.

Pacific Smelting and Mining Company has issued to its stockholders the first annual report, showing the property owned by the company and the status of development. The company's office is 42 Broadway, New York, and the mines and smelter are located at Fundición, Sonora, Mexico.

The United Iron Works Company, Springfield, Mo., has recently issued a catalog of concentrating machinery. General milling machinery for ore-dressing is described and illustrated. The company makes a specialty of electromagnetic separators and manufactures the Cleveland-Knowles separator.

Electrostatic Ore Separation is the title of a booklet which we have received from the Huff Electrostatic Separator Company, Boston. A brief explanation of the principles and application of electrostatic separation of minerals is given. This interesting new separator will be described in detail in our next issue.

Testing and Research Laboratories of E. S. Lincoln, Brookline, Mass., are described in a booklet which he has just issued. Testing and research work of any description in the electrical field is undertaken.

The Pacific Tank Company and the **National Wood Pipe Company** announce a business consolidation under the name of Pacific Tank & Pipe Company. The company has factories on the Pacific Coast, and offices in the Equitable Bank Building, Los Angeles, Cal. A new factory has just been completed at Portland, Ore., which is the largest of its kind.

The Year-Book for Colorists and Dyers, published by Herman A. Metz, New York, has been received at this office. The book contains much very useful information of interest to the trade.

Oxygenite is an electrolytic product in the form of a gray pulverulent mass resembling fine sand and possessing the property of releasing when heated all of its contained oxygen. This amounts to about 5¼ cu. ft. of gas per pound of oxygenite. The material burns slowly, without a flame, and can be kept almost indefinitely, even if exposed to the atmosphere, without deterioration. It is valuable for generating oxygen for oxy-acetylene welding plants without having recourse to buying compressed oxygen in tubes. The oxygenite process and the machinery used in its application are controlled by the Industrial Oxygen Company, New York.

Davis-Bournonville Company reports a great advancement in cutting steel by the oxy-acetylene process. A 6-in. steel shaft requires 3 minutes to cut at a cost of about 37 cents. A 3½-in. steel plate has been cut 18 ft. in 26 minutes, and a 6-ft. cut was made in ⅝-in. steel in 2 minutes and 29 seconds. Other excellent performances consisted in cutting 84 ft. of ¼-in. steel in one hour, and a circle of 19½ in. circumference out of 1-in. plate in 45 seconds.

Export Office Opened.—The General Fireproofing Company, of Youngstown, Ohio, has opened an export office at 396 Broadway, New York City, in charge of Mr. A. D. Level, who has been for several years with the United States Steel Products Export Company.

Cyanide Plant Opens.—The old mill of the Natividad y Anexas Mining Company in the Sierra Juarez district, State of Oaxaca, Mexico, has been closed down and the ores will be treated by the new cyanide plant, which has just been finished.

Forge Shop Furnaces.—The W. S. Rockwell Company, of New York City, have sent us their catalog No. 8, on their forge shop furnaces operated with oil or gas fuel, for forging, flanging, plate heating, angle-bending, spring-fitting, case-hardening, tempering, annealing, etc. The catalog gives data on the various sizes in which these furnaces are made, with diagrams showing their construction.

Nationality of Inventors of Electric Steel Furnaces.—In a recent lecture on electric steel furnaces the following somewhat amusing remark was made by Mr. Viktor Engelhardt, chief engineer of the Siemens & Halske Company: "The inventors of the most important arc furnaces for steel making are Frenchmen like Heroult and Girod, or Italians like Stassano. The inventors of the induction furnaces are of the Teutonic or Anglo-Saxon races: Kjellin, Frick, Gronwall are Swedes; Colby, an American; Roechling and Rodenhauser are Germans. One might make the paradoxical remark that the types of the furnaces are in conformity with the race characteristics of the inventors. The arc furnaces spark with a noise and brilliant flame suggesting more fire than there is really in them. The induction furnaces behave in the reverse way. Their cooler outside appearance does not suggest at once how much heat is available in their interior."

Personal.

Dr. **Richard Amberg**, formerly manager of a carborundum company in Southern Switzerland, is now connected with the Indiana Harbor, Ind., plant of the American Steel Foundries, where an electric steel furnace is in operation.

Mr **A. J. Bettles**, of Salt Lake City, has resigned his position as consulting metallurgist to the Utah Copper Company, and expects to devote his time to private interests. He has been closely identified with the progress of metallurgy, and particularly ore dressing, in Utah for a quarter of a century.

Mr. **J. W. Coyle**, who was connected with the Best American Calorific Company until they retired from business, is now with the Rockwell Furnace Company, making a specialty of oil and gas furnaces for railroad work. Mr. Coyle is an experienced railroad man, having formerly been master blacksmith for the Lehigh at Wilkes-Barre, and later in charge of the drop-hammer and machine department at the forge shops of the Reading, at Reading, Pa.

Mr. **J. V. N. Dorr**, metallurgical engineer of Denver, and the inventor of the classifier and pulp thickener known by his name, has recently been in San Francisco on professional business.

Mr. **L. C. Hammond**, formerly at the head of the sales staff of the Denver Engineering Works, has accepted a similar position with the Central Colorado Power Company. Mr. P. D. Grommon has been placed in temporary charge of Mr. Hammond's work.

Mr. **Lafayette Hanchett**, formerly of Colorado, but later identified with the Newhouse interests in Utah, has left for an extended world tour, having sailed from San Francisco last month.

Mr. **H. W. Hardinge**, the inventor of the Hardinge conical-tube mill, of New York City, is on a professional trip and will visit the Lake Superior copper district and the famous Cobalt, Canada, silver district.

Mr. **C. A. Heberlein** has recently returned to the City of Mexico from an outlying district, where he has been erecting a smelting plant.

Mr. **A. B. W. Hodges**, who has for some years been general manager of the Granby Consolidated Company in British Columbia, has recently resigned his position, and will assume the general managership of the Cerro de Pasco Mining Co. in Chile

Mr. **D. C. Jackling**, manager of the Utah Copper Co., has left Salt Lake City on a trip to Colorado Springs, from which place he will go to Ray, Ariz., to inspect some copper properties.

Mr. **E. M. Johnson** has recently resigned his position as metallurgist at the Collinsville, Ill., plant of the St. Louis Smelting & Refining Company.

Mr. **F. H. Nickle** has resigned his position as engineer for the Pennsylvania Salt Manufacturing Company, at Wyandotte, Mich., and will specialize in vacuum evaporating work and salt-plant engineering. Mr. Nickle is associated with the firm of McDonald & Nickle, with headquarters at Detroit.

Mr **W. H. Oldham** has recently been appointed general blast-furnace superintendent for the Birmingham Coal & Iron Company, Birmingham, Ala.

Mr **William Scallon**, who formerly resided at Butte, Mont., where he was president of the Anaconda company and Western manager of the Amalgamated company, has removed to New York, where he will be permanently located at Hotel Netherland. Mr. Scallon will engage in the practice of law and will have no connection with the companies mentioned.

Mr **Charles F. Shelby**, formerly with the Phelps Dodge Company in Arizona, and later superintendent at the Greene-Cananea, in Mexico, has accepted a position as smelter manager with the Cerro de Pasco Company, in Peru.

Mr. **E. R. Wooley**, president of the Utah Ore Sampling Co., has returned to Salt Lake City, from an Eastern trip

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matters and in chronological order

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

Electric Furnaces (*Continued*)

657,202, Sept. 4, 1900, H. C. McBrair, of Middletown, N. Y. Resistance type. A dental furnace for firing porcelain, in the form of a box having a hinged cover. The joint between the base and cover lies in a plane inclined downward toward the front. The base and cover are of metal lined with fire-clay. A platinum resistance-wire is embedded in the inner face of each part. Contact springs and pins connected with the hinges complete the circuit.

657,911, Sept. 18, 1900, G. D. Burton, of Boston, Mass. Resistance furnace for reducing ores containing gold, silver, sulphur and arsenic. The current used is from 6000 amp to 7000 amp at from 4 volts to 16 volts. The furnace has a vertical resistor chamber surmounted by a short stack, both consisting of carbon or of an alloy of copper, three parts; antimony, two parts, and tin one part. The upper and lower ends of the chamber are surrounded by massive clamps of copper, serving as terminals. The lower clamp has a plate closing the lower end of the furnace, and is hinged to drop down. The chamber is surrounded between and above the electrodes with clay, and may be lined with clay. An inclined feed-pipe opens into the stack. This pipe and the stack are closed by caps. Pipes serve for the introduction of steam and air, to assist combustion and chemical action. A pipe leading to an eduction-pump and water condenser serves to draw off sulphur and arsenic vapors. The furnace has steam and air gages and a pyrometer. Combustible matter may be mixed with the ore charge. By gradually raising the temperature of the charge, lead, silver and gold, or lead, antimony, zinc, etc., may be successively smelted and separately drawn off. The furnace may be used for separating bullion and for producing calcium carbide.

658,315, Sept. 18, 1900, A. H. Cowles, of Cleveland, Ohio. Arc or resistance type. The electric furnace proper is a chamber with solid carbon walls, constituting one electrode, and receiving a depending carbon electrode. The depending electrode is adjustably carried by a water-cooled stem passing through a stuffing-box. Ore-hoppers with double valves open into the top of the chamber. At each side of and connected with the smelting chamber are chambers with carbon walls, receiving coke supported on a grate beneath. Each side chamber communicates with hot-blast stoves or regenerators for supplying heated air and gas. Valved fuel-hoppers open into the upper ends of the side chambers. The walls and hoppers are water-jacketed and cooling coils may be embedded in the walls and cover. In operation, heated air and gas are introduced into one side chamber and pass through the body of coke therein, effecting combustion. The products of combustion then pass through the charge of ore in the intermediate electric furnace, preferably not through the zone of highest temperature, and thence through the body of coke in the other side chamber, escaping to the regenerators. The flow of gas and air is periodically reversed. The furnace may be used to reduce "lime, clay, sand, aluminium, etc."

658,536, Sept. 25, 1900, G. M. Westman, of New York, N. Y. Resistance type. Furnace for treating ores containing arsenic and precious metals. The furnace chamber is of fire-clay, with a removable lid carrying a valved hopper. The smelting chamber is a trough, the bottom of which is recessed at each end to receive water cooled cast iron electrodes. The exposed upper ends of these electrodes may be curved to the shape of parabolas, to increase contact with the ore. The charge, which

may be an arsenical ore containing iron, e. g., mispickle, carries the alternating current and is heated thereby as a resistor. The liberated arsenic escapes through a pipe to settling chambers and a water condenser. The melted ore free from arsenic, molten iron sulphides, or mattes, are tapped off. Much of the gold and silver sinks downward and may be absorbed by pools of lead over the electrodes.

658,698, Sept. 25, 1900, C. S. Bradley, of New York, N. Y.

Modification of furnace shown in patent 597,945, especially designed for the production of cyanides from carbides. The furnace is of the wheel type, carbon electrodes of opposite polarity depending into one side of the annular rim constituting the receptacle. Pipes for supplying nitrogen to the carbide extend outward from the hollow shaft to the rim at different points. A valve in the hollow shaft, or separate valves in the several pipes actuated by fixed tappets, supply nitrogen to the carbide from the time of its formation until it cools below the point at which further absorption will take place.

NEW BOOKS.

Allen, Alfred H. Commercial organic analysis; treatise on the properties, modes of assaying, and proximate analytical examination of the various organic chemicals and products employed in the arts, manufactures, medicine, etc.; with concise methods for the detection and estimation of their impurities, adulterations and products of decomposition. Vol. 1, fourth edition, entirely rewritten. Edited by H. Leffmann and W. A. Davis. Illustrated. Price, $5.00 net Philadelphia: P. Blakiston's Sons & Co.

Armes, Ethel. Story of coal and iron in Alabama. Illustrated. Price, $5. Alabama Chamber of Commerce.

Ball, W. V. Law affecting engineers. 300 pages. Price, $2 net. New York: D. Van Nostrand Company.

Ballinger, W. F., and Emile G. Perrot. Inspector's handbook of reinforced concrete. Price, $1.00. New York: Engineering News.

Bardwell, Fred Leslie, and Elwood B. Speer. Laboratory experiments in inorganic chemistry. Compiled from various sources for the use of students of the Massachusetts Institute of Technology. Price, 75 cents. Boston: F. L. Bardwell.

Bartley, Elias Hudson. Text book of medical and pharmaceutical chemistry., Seventh edition, revised 1909. 749 pages, illustrated. Price, $3. Philadelphia: P. Blakiston's Sons & Co.

Blaine, Robert Gordon. Calculus and its applications Practical treatise for beginners, especially engineering students. Price, $1.50 net. New York: D. Van Nostrand Co.

Breckenridge, Lester Paige, and Paul Diserens. Fuel tests with Illinois coal. (Comp. from tests made by the Technological Branch of the United States Geological Survey, at the St. Louis fuel testing plant, 1904, June 30, 1907). Fifty-four pages, illustrated, tables and diagrams. Urbana, Ill.: University of Illinois.

Cambria Steel Co. Cambria Steel; handbook of information rel. to structural steel manufacture by the Cambria Steel Co., containing useful tables, rules, data and formulae for the use of engineers, architects, builders and mechanics; prepared and compiled by George E. Thackray. Ninth edition. 484 pages. Price, $1.00. New York: Cambria Steel Co.

Durkee, F. W. Experiments in general inorganic chemistry. Second edition, revised. Price, 50 cents. Tufts College, Medford, Mass.

Gillette, Halbert Powers, and R. T. Dana. Cost keeping and management engineering; treatise for engineers, contractors and superintendents engaged in the management of engineering construction. Price, $3.50. Chicago: M. C. Clark Publishing Co.

Gissing, F. T. Commercial peat, its uses and its possibilities. Price, $2.00 net. Philadelphia: J. B. Lippincott Co.

Gregory, R. Arman and H. E. Hadley. Class work of Physics. Price, $1.00 net. New York; Macmillan Company.

Grover, F. Modern gas and oil engine. Fifth edition. Price, $2. New York: Spon & Chamberlain.

Grünwald, Julius. Enameling on iron and steel. Price, $2.00 net. Philadelphia: J. B. Lippincott Co.

Henderson, J. Electrotechnics. (Physical and electrical engineering laboratory manuals.) Price, $1.20. New York, Longmans, Green & Co.

Latta, M. Nisbet. American producer gas practice and industrial gas engineering. Price, $6.00 net. New York: D. Van Nostrand Co.

Lunge, G. Coal, tar and ammonia. Two volumes. Fourth edition. 1210 pages, illustrated. Price, $15.00 net. New York: D. Van Nostrand Co.

Lunge, G. Manufacture of sulphuric acid and alkali; theoretical and practical treatise. Vol. 2: Sulphate of soda, hydrochloric acid, Leblanc soda. Third edition, much enlarged. 1044 pages, illustrated. Price, $15.00 net. New York: D. Van Nostrand Co.

Morgan, J. J. Blast furnace practice. Price, 75 cents net. Philadelphia: J. B. Lippincott Co.

Morsch, Emil. Concrete steel construction (Der Eisenbetonbau); authorized translation from the third (1908) German edition, revised and enlarged by E. P. Goodrich. Price, $5.00. New York: Engineering News.

Munroe, C. E., and Clarence Hall. Primer on explosives for coal miners. 61 pages. (U. S. Dept. of the Interior, U. S. Geological Survey Bull.) U. S. Supt. of Docs.

Normandy, Fk. Sea Water Distillation. Price, $2.00 net Philadelphia: J. B. Lippincott Co.

Sharpe, T. Modern Foundry Practice, including molding, materials used, machinery and appliances, etc. Second edition. Price, $8.00. New York: Spon & Chamberlain.

Shelford, F. Pioneering for engineers; including most suitable camp equipment, rations, medicines, clothing, etc. Price, $1.25. New York: Spon & Chamberlain.

Sudborough, J. J., and J. C. James. Practical organic chemistry. 394 pages, illustrated. Price, $2.00 net. New York: D. Van Nostrand Company.

Stewart, R. Wallace. Elementary text-book of physics. Price, $1.50 net. Philadelphia: J. B. Lippincott Co.

Suplee, H. Harrison. Gas turbine. Progress in the design and construction of turbines operated by gases of combustion. Price, $3.00 net. Philadelphia: J. B. Lippincott Co.

Tutton, A. E. Howard. Crystalline structure and chemical constitution. Price, $1.60 net. New York: Macmillan Company.

Tyrell, H. Gratton. Concrete bridges and culverts for both railroads and highways. Price, $3.00. Chicago: M. C. Clark Publishing Company.

BOOK REVIEWS.

Producer-Gas Fired Furnaces. By **Oscar Nagel.** Ph.D. 8vo., 192 pages, 237 illustrations. Price, $2 net. New York City: The Author (P. O. Box 385).

Intended as a hand-book giving detailed descriptions of these furnaces, it only measurably achieves its object. While the book is in parts commendably written and illustrated, yet other parts are unsatisfactory both in manner of presentation and illustration, while the style, as written English, is faulty throughout. The scientific explanations, where ventured upon, are mostly of doubtful accuracy.

The author is most at home in describing the furnaces used in the lime, cement, brick, ceramic, glass and chemical industries, and in respect to these the book will certainly be found useful; the furnaces used in non-ferrous metal industries are fairly well treated; the furnaces used in the iron and steel industry are poorly handled, the author often showing igno-

rance of the purpose or mode of operation of some particular furnace (e.g., furnace shown in Figs. 77-81).

In describing producers themselves, some blunders occur which decrease the value of the information. The Bildt automatic feed, for instance, is labeled the "Morgan Producer." We cannot consider this chapter as of much value; it is rather a heterogeneous compilation.

Altogether, the subject is neither exhaustively nor equably treated; the book might have been much better. Parts contain useful information which make the book worth while; it is, however, neither scientifically accurate nor otherwise always reliable.

* * *

Modern Coking Practice. By **T. H. Byron** and **J. E. Christopher.** 8vo. 109 illustrations and diagrams, 168 pages. Price 8/6 (in New York, $3.50). London: Crosby, Lockwood & Son.

This is an eminently practical, condensed, well-written work, thoroughly up to date. The authors are chief chemist and submanager, respectively, of the Wigan Coal & Iron Company, and in practical charge of an extensive Semet-Solvay coking plant. The sampling and valuation of the coal and coke, and working up of the by-products, are treated in the same business-like manner as the main subject—the construction and operation of the ovens. The book is recommended to all interested in this subject.

* * *

The Chemistry of Cyanide Solutions. By **J. E. Clennell,** B.Sc. Second edition. 8vo., 202 pages. Price, $2.50 net. New York: McGraw-Hill Book Company.

A second edition of this useful book will be received with pleasure by all who have used the first edition of the work; it is an eminently practical laboratory guide to the metallurgical chemist in charge of a cyanide plant. We would recommend the chemist, however, to use the book as a nucleus about which to accumulate notes of other analytical cyanide methods as they appear from time to time. In that way a complete laboratory guide can be constructed which will put the metallurgist abreast of the best practice of his profession in this line.

* * *

Die Herstellung von Hochprocentigem Ferrosilizium im elektrischen Ofen. By Dr. Ing. **Waldemar Pick** and Dr. Ing. **Walter Conrad.** 8vo., 144 pages, 26 illustrations. Price 6.60 marks. (Retail price in New York, $2.15.) Halle a. S., Germany: Wilhelm Knapp.

This monograph on the electric furnace production of high-percentage ferro-silicon is an amplification of the doctor's thesis presented by the authors at the University of Vienna. It contains more detailed and more exact information upon this subject than is to be found anywhere else. The student of electrometallurgy, the electric furnace expert, the manufacturer of ferro-silicon, and the user of the same, will all find the work full of interesting and valuable information.

A short introduction gives the older metallurgical literature on "silicon pig" iron or "glazed pig." A longer chapter describes the recent researches on the silicides of iron, and gives the phase diagram or equilibrium diagram of the various mixtures of iron and silicon, followed by a description of the commercial ferro-silicons.

The spontaneous crumbling of ferro-silicon containing 30 to 65 per cent silicon is next discussed. This is an important topic, because of the decreased market value of such powdered material. Sometimes two samples from the same furnace, with identical analysis, will behave differently, one crumbling and the other not. Their conclusion is that the crumbling is due to a crystalline decomposition, caused primarily by two rapid or too slow setting from the melted condition.

Another interesting subject discussed is the explosiveness of some ferro-silicons, and the poisonous gases given off by them. The explosions are ascribed to gases, like acetylene or silicon hydride, given off by the action of atmospheric moisture and ignited by sparks caused by the attrition of the pieces against each other as the material is moved. The gases are given off particularly by 40 to 60 per cent ferro-silicon, and can be avoided by soaking the material in water or passing steam through it, and then not packing it until it has been thoroughly dried.

Another remedy suggested is to forbid the shipping of 40 to 60 per cent ferro-silicon.

Eight pages are given to the chemical analysis of these alloys, in which the authors give the methods which they have found most reliable.

A sixteen-page chapter describes the various electric-furnace processes which have been proposed and used for making ferro-silicon. The connection of the rise of this industry with the historic "slump" in the calcium carbide business is dwelt upon. The patents of Price and Tone are not very appreciatively mentioned.

There then follow the three most valuable chapters in the book: One of 37 pages on the construction and operation of an electric furnace ferro-silicon plant, another of 8 pages on the theoretical work done in the production of these alloys, and a third of 21 pages on the useful effect and practical efficiency of furnaces in actual operation.

In the chapter on construction and operation of plant, the details are minute and satisfactory; the person following them, however, would need to carefully consider to what extent he might be infringing patented apparatus and methods.

In the calculation of the theoretical work necessary for producing different grades of ferro-silicon, Dr. Henry Noel Potter's determination of the heat of oxidation of silicon, 7595 calories per gram, is taken as the most reliable. With this we agree, but we cannot agree to take Thomsen's value for the heat of oxidation of carbon to monoxide, 2183 calories per gram, which is in error at least 10 per cent; Berthelot's value of 2430 is much more probably correct. The gases escaping from the furnaces are assumed at 700° C, the alloys at 1500° to 2000°, the specific heat of silicon as averaging 0.29, its latent heat of fusion (probably 128 calories) is neglected. With these conditions, a minimum power requirement of 321 kw-days is calculated per ton of silicon. Working with partly un-enclosed arc, the escaping gases are assumed to be at 2000°, and the minimum power requirement is 358 kw-days. For various grades of ferro-silicon the minimum power requirement using metallic iron in the furnace is calculated per ton of product in kw-days as:

	With Enclosed Arcs.	Un-enclosed Arcs.
25 per cent ferro-silicon.....	88 (130)	97 (150)
50 " " " "	165	184

The kw-days given above in parenthesis indicate the minimum power requirement calculated for using iron oxide in the furnace instead of metallic iron.

The investigation of actual furnaces to determine their efficiency gives very interesting results. The net efficiency is only 40 to 50 per cent, the 60 to 50 per cent list being accounted for by radiation and conduction and the vaporization of silica, in the ratio of about 20 per cent to the former and 30 per cent to the latter. The heat required to heat cold SiO_2 to the vaporizing point and vaporize it is taken as 843 calories per gram. One conclusion reached is that at least 2 kg of silicon can be obtained per kw-day, and better results than this have been obtained.

The concluding chapters deal with the influence of the voltage, current used, and impurities present, on the output of silicon and on the purity of the alloys obtained.

As a whole, the book is a highly interesting contribution to electrometallurgical literature, and we congratulate the authors upon their splendid work.

J. W. Richards.

Metallurgical and Chemical Engineering

A Consolidation of Electrochemical and Metallurgical Industry and Iron and Steel Magazine

Vol. VIII. New York, May 1910 No. 5

PUBLISHED MONTHLY BY THE

Electrochemical Publishing Company

239 West 39th Street, New York.

Pittsburgh Office, 201 Oliver Building.
Western Office, Denver, Colo.
European Office, Hastings House, Norfolk St., Strand, London, Eng.

J. W. Richards, Ph.D., President.
Curtis E. Whittlesey, Treasurer. E. F. Roeber, Secretary.
Telephone Call: 4700 Bryant.

E. F. Roeber, Ph.D., Editor.
H. C. Parmelee, Western Editor.

Yearly subscription price for United States, Mexico and United States dependencies, $2.00; for all other countries, $2.50 (European exchange, 10 shillings, 10 marks, 12 50 francs).

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Entered as Second-Class Matter at the Post Office at New York, N. Y., under the Act of Congress, March 3, 1879

New York, May, 1910.

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Pittsburgh and Electrochemistry.

Electrochemical engineering is not a strange or new art in Pittsburgh. On the contrary, the early history of electrochemical industries in this country—Hall's work on aluminium and Acheson's work on carborundum—is intimately connected with the Pittsburgh district. Only after these two pioneer electrochemical industries had been there founded on a commercial scale were they transplanted to the new power center, Niagara Falls, which then grew into the electrochemical center par excellence. Now, however, with electric steel refining furnaces increasing almost daily in importance, it requires no particular flight of fancy to think of Pittsburgh as another coming electrochemical center. But how different such a statement sounds now from what it would have sounded a few years ago. Then iron men would have thought of the electric furnace trying to replace the blast furnace. Now it is universally understood that the big field of the electric furnace in iron and steel metallurgy is as an adjunct to the Bessemer converter and to the open hearth to produce that better steel so urgently needed.

* * *

The American Electrochemical Society has wisely selected Pittsburgh for the place of its annual meeting in May. It can no longer be said that the society enters thereby the enemy's country. Progressive steel men now recognize the helper and friend in the electric furnace man. To reach a perfect mutual understanding of all the various phases of the problem, nothing can be more effective than social intercourse and heart-to-heart talk between both classes of engineers, and for this the convention will offer abundant opportunities. So we look forward expectantly to one of the most successful and pleasant meetings, especially since electrochemists are now returning for the first time as a professional body to the place of their early triumphs. On revient toujours à ses premiers amours.

Oxygen and Nitric Acid from Air.

Mr. Ernest A. LeSueur, who 20 years ago was the pioneer of the diaphragm cell for sodium chloride electrolysis, has contributed to this issue a most interesting article on "commercial oxygen," produced from the atmosphere by a simple mechanical process. The subject attracts just now great attention. If it would be possible to produce "commercial oxygen" cheaply enough, many applications of it in large-scale metallurgical and chemical engineering processes could be worked out. As Mr. LeSueur points out, to be cheap the method should be a mechanical or physical rather than a chemical one; further, it should not be necessary to produce pure oxygen. If an atmosphere enriched in oxygen compared with the ordinary atmosphere is sufficient, he claims it can be produced at a very reasonable figure by a sort of turbine or "steam engine."

* * *

In this connection attention may be called to an abstract of a patent of Mr. Henry Howard, also given in this issue, for the continuous production of nitric acid from air. In a very clever

way the machinery is so designed as to fulfill the desiderata of that theory according to which the oxidation of the nitrogen is a purely thermal phenomenon. First, compression, and, by electric discharges, establishment of a gas mixture comparatively rich in nitrogen oxides, and then an exceedingly quick cooling of the mixture so as to "freeze" the equilibrium corresponding to the higher temperature and prevent the dissociation of the nitrogen oxide. This cooling effect is obtained by sudden expansion of the gas mixture while performing work in an engine or turbine. Mr LeSueur's article and Mr. Howard's patent are interesting examples of the novel application of mechanical engineering principles in chemistry.

Friedrich Kohlrausch.

In Friedrich Kohlrausch, Germany has lost one of its most prominent and influential physicists, a past-president of the Physikalisch-Technische Reichsanstalt and the one man who has done more than anyone else to cultivate experimental laboratory work in physics in German universities. As to his research work along manifold lines, attention may be called here only to his very extended and minutely accurate investigation of the electric conductivity of salt solutions at varying concentrations, which, taken together with Hittorf's determination of transference numbers, enabled him to establish the law of the "independent migration of ions" for infinitely dilute solutions. Qualitatively at least, this was the first statement of the electrolytic dissociation theory. What has been added later is Arrhenius' method of determining by different methods the degree of dissociation in solutions which are not infinitely dilute and the great generalization of the theory by numerous investigators whereby it became possible to apply it to many problems of the theory of solutions and of general chemistry.

The Passing of the Argo Smelter in Denver.

The last days of March witnessed the drawing of the fires from the furnaces at the smelting plant of the Boston & Colorado Smelting Company at Denver. The Argo smelter, as the plant was commonly known, has been one of the most interesting and important factors in Western metallurgy for almost 40 years. Originally established at Black Hawk, 30 miles from Denver, at a time when the rich surface free-milling ores of that famous camp had given place to sulphides, the Argo smelter played a leading part in the fortunes of the miners who were discouraged over their rapidly diminishing gold recoveries from ore known to contain high values. The leading spirit in establishing the smelter was Nathaniel P. Hill, at that time a young Eastern college professor. The process used resulted in the production of a rich copper matte which was shipped to Swansea for refining. Later Mr. Richard Pearce introduced the Ziervogel process for recovering the silver, and introduced a method for treating the residues for gold which was kept secret for almost 40 years. When finally disclosed it was found to be simple, and a number of metallurgists confessed to having guessed the principles involved some years before their disclosure. The gold was concentrated in copper bottoms which were subsequently granulated and smelted in a reverberatory furnace with iron pyrites. Successive meltings and granulations with final treatment in the refinery in crucibles yielded a bullion 950 fine in gold.

This system of smelting and refining, using copper as a base, was continued to the end, although lead smelting developed far beyond the possibilities of copper smelting in Colorado. In 1906 the Argo refinery was destroyed by fire and since that time the matte has been sold to the American Smelting & Refining Company. The diminishing fortunes of the Argo plant were coincident with the discovery of rich lead ores and the genesis of lead smelting in Colorado. Copper ores have never been abundant in that State, and it has always been possible to handle them in lead smelting plants, producing a low-grade matte which was subsequently concentrated to a grade suitable for refining in the converter. The Argo company at one time established a second plant at Alma, Col., but it was not profitably operated and was finally abandoned. It is an interesting speculation to consider what would have been the effect on the lead smelting industry of the West had the Argo company entered the lead smelting field instead of sticking to its own process, which finally became antiquated. The ingenuity of its metallurgists and the prestige of its financial backers should have made it a formidable factor in present-day developments. The history of the company shows it to have been very successful, and it retires with the good will of many miners who had dealings with it.

Pittsburg's Supremacy in Iron and Steel.

It is only among those who are unaccustomed to think in quantities that the notion has been entertained that Pittsburg loses any material part of her prestige as an iron and steel producing center by the building of the steel plant at Gary, Ind. The Gary plant exemplifies an important economic principle, but by no means a new one. Nearly 20 years ago it was common talk in the industry that a location on the Great Lakes had an advantage over the remainder of the Central West, because it was better for the fuel to go to the ore than the ore to the fuel, but instead of a general exodus only a few stragglers have wended their way to a lake site. The Johnson Company moved from Johnstown, Pa., to Lorain, Ohio, making its first steel at the new site April 1, 1895, but owing to financial stress did not obtain the real benefit of the lake site for more than four years, since it was not until July 5, 1899, that the company made pig iron at Lorain. A few years later the Lackawanna Iron & Steel Company packed up its goods at Scranton, Pa., and moved to Buffalo. The Gary plant, at the south end of Lake Michigan, is, of course, different in that it does not in any sense represent a removal, for the United States Steel Corporation has clearly had no intention of moving any equipment or transferring any trade. The Gary plant is built from the ground up, with fresh capital, and its object is to take care of the increased requirements of the Western market.

* * *

The Gary plans, as outlined in the steel corporation's report for 1907, comprised eight blast furnaces and 56 60-ton open hearth furnaces, the blast furnaces being rated at 450 tons per day each, and the 56 open-hearth furnaces at 225,000 tons of ingots per month. This much was authorized at the time, and the statement was further made that an increase in the blast furnaces to 16 was ultimately contemplated, with a corresponding increase in the open-hearth and other equipment. In either case there would have been a large preponderance of steel mak-

ing over blast-furnace capacity. Apparently the plans have since been modified, for the present status of the work is that of eight blast furnaces completed, with six in blast and 28 open-hearth furnaces completed and in operation, with 14 to be completed this summer, eight blast furnaces and 42 open-hearth furnaces being expected to balance. It does not seem probable that the project will be expanded beyond these limits for some time. The function of Gary is to make steel by the most modern and approved methods, rather than to make an enormously large tonnage. The eight furnaces may be rated at 1,200,000 tons of pig iron annually, the steel ingot production being larger, through the reuse of scrap, while the finished steel output will doubtless be approximately commensurate with the pig iron tonnage.

* * *

From a tonnage standpoint, Pittsburg need not consider such an operation as presenting serious rivalry. One can arrive at Pittsburg's tonnage potentialities better by a study of what has been accomplished than by a mere survey of rated capacities. To do this it is necessary to go back to 1906, the last year of full employment, and then allow for new construction since that time. This, by the way, will show quite forcibly that Pittsburg has not stopped growing. Statistics of the production of Allegheny County, of which Pittsburg is the county seat, are given for 1906 as follows by the American Iron & Steel Association:

	Gross tons.
Pig iron	5,702,721
Bessemer steel	3,255,064
Open-hearth steel	3,799,907
Other steel	50,530
All steel	7,105,501
Rails	851,419
Structural shapes	1,054,747
Plates and sheets	1,300 873
All rolled products	5,686,017

The Mahoning and Shenango valleys, midway between Pittsburg and Lake Erie, and the Wheeling district, as well as various scattered plants in Western Pennsylvania, might perhaps be included with Pittsburg, but drawing a strict line we include with Allegheny County only a few plants just outside the county and distant less than 30 miles as the crow flies from the heart of Pittsburg. Thus we add to the above tonnage an allowance for the Midland and the two Donora furnaces, about 450,000 tons of pig iron annually. Since 1906 there have been built and put in operation within this circumscribed district: In 1907, two Carrie and one McKeesport furnaces; in 1909, two Dusquesne and one Aliquippa; in 1910, two Aliquippa, these eight furnaces having a rated capacity of 1,370,000 tons a year. Altogether, then, the present pig iron capacity of the immediate Pittsburg district is fully 7,500,000 tons a year, more than six times the prospective capacity of the Gary plant as at present outlined.

* * *

In steel capacity there is an increase since 1906. The Bessemer department at Duquesne, two 10-ton vessels, has been dismantled, but in place are 18 open-hearth furnaces with greater capacity; at Homestead there is additional open-hearth capacity, and at Aliquippa the Jones & Laughlin Steel Company is completing four 200-ton Talbot furnaces. With minor improvements at other plants, and considering the steel capacity at Vandergrift, Butler and Donora, just outside the county, the prospective steel ingot capacity is well in excess of 8,000,000 tons annually. With 7,500,000 tons of pig iron capacity, the Pittsburg district makes only about a quarter-million tons of merchant pig iron a year. It is not a center for the manufacture of the finer products of the foundry, nor, indeed, does it surpass other districts so much in the finer forms of steel as it does in tonnage. There is room for Pittsburg to grow along these lines, as the necessity is reduced for it to grow in the production of the heavy forms of steel.

Refractory Brick.

The subject of refractories is of immense importance for metallurgical and chemical works. But the knowledge of this subject is scattered and illusive or at least indefinite. By their contact with the whole of chemical and metallurgical industries the makers of firebrick have acquired some practical data of undoubted general value on the relation between methods of manufacture and properties of firebrick, while chemists and metallurgists in special fields have valuable and definite ideas on what kind of special refractories they want or need. But the trouble seems to be that there is not yet the right co-operation or understanding between the manufacturer and consumer of refractory brick. That a refractory brick may be excellent for one purpose and wholly unsatisfactory for another goes without saying. An acid brick is destroyed by a basic charge or slag. But besides the chemical nature of the reaction, the purely physical factors are of equal importance. The consumer should know the maximum temperature which his brick has to stand, and the firebrick manufacturer should adjust his baking process accordingly so as to produce a brick that will not shrink at the highest temperature in use. To indicate how contradictory the requirements are in different cases, we may simply mention that in most cases firebrick is expected to be a poor heat conductor in order to reduce the heat loss, while in other cases—in zinc retorts, in muffle roasting, crucible melting, etc.—the refractory is expected to transport the heat from the outside to the inside with high efficiency. It seems that a discussion of the subject of refractories from the standpoint of both the consumer and producer should do much good for both parties concerned. It is with pleasure that we call attention to an interesting article on defects in refractory brick and their causes, contributed to this issue by Mr. Gilbert Rigg, in charge of the experimental work of the New Jersey Zinc Company. It is the result of a large number of studies and tests and is contributed in the hope of starting a discussion on this subject. For this purpose and in the hope of doing something useful we place our columns gladly at the disposal of both users and makers of refractory brick.

Irregularities of Blast-Furnace Running.

Our January issue contained an article on regularity of blast-furnace operation, by Dr. Joseph W. Richards, and in the present issue will be found an article, more or less supplementary to the former one, on the chief irregularities of blast-furnace running by Mr. Edward R. Cook, of the Warwick Iron Company, written from the blast-furnace manager's standpoint.

That these two authorities should agree on the main points under discussion is highly satisfactory from the standpoint of blast-furnace literature; that they should lay the chief emphasis on different phases of the problem is perfectly natural and no less instructive than their points of agreement. In brief, Dr. Richards assumes a furnace in good running order, not near the close of a campaign, and discusses the factors of regularity in its running such as are concerned with its immediate management and treatment. On the other hand, Mr. Cook starts with the tacit assumption that all these factors are inferior in importance to combating those secular changes which mark declines in output steadily operating through months of running and which finally compel the furnace to go "out of blast." Mr. Cook very aptly says: "Take a blast furnace built on proper lines and anybody can run it; take one built improperly and no one can run it." Dr. Richards' discussion applies, therefore, to the best and most regular running of a furnace assumed to be built properly; Mr. Cook's mainly to the irregularities and difficulties met in a furnace built improperly, causing secular changes which finally stop the furnace.

* * *

Both discussions are, therefore, valuable, and we do not hesitate to designate Mr. Cook's as the more fundamental, because it is certainly imperative to have, first, a furnace capable of being kept "in blast" before it becomes worth while to discuss how to run it most regularly and most efficiently. Mr. Cook has discussed the great fundamental difficulty of keeping the blast furnace in operation; Dr. Richards the fine points and finishing touches, or refinements concerned in its steady operation at highest efficiency. Some of Mr. Cook's statements will hardly pass the scrutiny of the blast-furnace profession unchallenged. The statement that no one knows how to build the blast furnace right should provoke discussion. His statement that the three largest producing blast furnaces in the United States, in 1909, were three dry-blast furnaces, certainly proves the very great importance and value of the items of regularity discussed by Dr. Richards. In fact, we cannot but regard the questions raised as "open" questions, and their further discussion by technicians and theorists is likely to prove profitable all around.

Bessemer Roasting.

Evolution of technical terms is coincident with the changes in practice, and final usage is usually dictated only by a full understanding of the principles of a method or process. As long as the latter is in a transitional stage, the descriptive terms used are likely to be numerous, local in their origin and meaning, and not wholly satisfactory. The developments of the past few years in the practice of roasting sulphide ores have given rise to a number of terms, no one of which seems wholly adequate to express concisely the principles involved and the result obtained. Various terms such as sintering, pot roasting, layer roasting, blast roasting, lime roasting, etc., have been proposed and used. An analysis of these terms will show that each of them has some special significance, referring to the mechanical device used, the nature of the mixture to be roasted, or the general result obtained, but none of them can be considered a generic term complying with the conditions mentioned, so that it is worth while to look for a better one.

A recent metallurgical writer has suggested the use of the term "Bessemer roasting," as fulfilling all the conditions. The Bessemer process implies the use of a blast of air to effect certain chemical changes without the use of extraneous fuel. In steel making the Bessemer process is based upon the heat evolved by the oxidation of certain elements in pig iron. In "Bessemerizing" copper matte we have similar conditions and results, applying a blast of air to oxidize sulphur and iron and to liberate copper. Finally the analogy is apparent in what it is proposed to call Bessemer roasting, for in that process we resort to a blast of air to oxidize the sulphur, and the process is carried on with the fuel contained in the ore itself. The suggested term is properly descriptive from a metallurgical point of view. It is independent of the details of the mechanical application of the process, which, indeed, are the most unsettled features of it. Roasting in pots was an early application, and only a short time ago it appeared that roasting in thin layers would supersede the pot process, but that has not yet come to pass. The term "lime roasting" is no longer tenable, since it lays stress on a feature now known to be unessential. Finally, sintering can be accomplished in a variety of ways and furnaces, and the term has no reference to the Bessemer feature of the roasting process as now carried out. On the whole it seems as though the term "Bessemer roasting" is the most acceptable that has been proposed, and we are glad to endorse it and bespeak for it general usage.

Pittsburgh and By-Product Coking.

It would appear that the iron and steel industry has experienced sufficient expansion in tonnage capacity to cover the requirements of the market for a time and is destined to devote its energies chiefly to the introduction of economies and improvement of product. In the last dozen years three periods of high pressure have alternated with periods of depression. The spectacular boom of 1899 was followed by a sharp and short readjustment in 1900; the next period of high pressure lasted longer, and was followed by the depression extending from late in 1903 to late in 1904, this giving way to three years of high pressure followed by a depression of a year and a half. Each period of high pressure caused plans to be adopted for new erection, much of the new capacity not becoming effective until the pressure for material had subsided. The next period of high pressure would show the new capacity to be needed, and would give rise to a fresh burst of new erection. The longer the period of high pressure, the greater the amount of new erection undertaken, and thus it came about that in the most recent period of high pressure the crop of new blast furnaces and steel works was by far the largest of all. The depression is past, the new erection is not all completed, yet existing capacity is not fully engaged. In other revivals it required but a few months to take up all the slack and produce a position in which mills were falling farther and farther behind in their deliveries.

* * *

The iron and steel industry is always improving its methods and appliances, but the rate of progress varies from time to time. We do not assert that if it were possible to list all the improvements made in the past quarter century it would not be found that more improvements were introduced in periods of extreme prosperity than in periods of relative depression, but

if so the showing would be made so by there being much more new erection in periods of prosperity than in periods of adversity. Relative to the amount of money spent, the greatest progress is undoubtedly made when profits are small. Nothing could be more natural. When existing capacity is taxed and producers are falling behind in deliveries it is justifiable in new erection to build as others have built before; when existing capacity is not fully engaged it is necessary to justify the new erection by improvements over existing practice. In these circumstances the next few years in the iron and steel industry promise to constitute a period of improvement in methods and appliances. Along these lines by-product coking looms large. In our March number we pointed out that the development of the large gas engine greatly increased the feasibility of by-product coking by furnishing a new and unlimited field of usefulness for the gas, and we do not wish here to traverse any of that ground again. Rather, we take as our theme the Pittsburgh district as a locus for by-product coking. The suggestion as to the unwisdom of "carrying coals to New Castle" may arise in the minds of some of our readers, but let us look into the matter a little. The Pittsburgh district is one of the greatest consumers of power in the world, and most naturally, for once Pittsburgh was only the junction of two rivers surrounded by forests and the industries were attracted chiefly by cheap fuel; naturally they were industries which require a great deal of it. Pittsburgh was the home of aluminium manufacture until the Niagara Falls power became available.

* * *

Pittsburgh, it may be said, is a natural gas center; hence, why try to make there the much poorer gas which by-product coking yields? Let us see. From a natural gas standpoint the State of Pennsylvania is practically the Pittsburgh district. The Geological Survey reports show a rapid increase in Pennsylvania's natural gas production, until 1904, since which year the value has been almost stationary, and we know perfectly well that the quantity must therefore have been decreasing, since values per 1000 ft. have been steadily increasing. Statistics of quantity are available only for 1907 and 1908. They are as follows for Pennsylvania, in thousands of cubic feet:

	1907.	1908.
Production	135,516,015	130,476,237
Consumption	164,541,179	147,790,097

The decreased consumption in 1908 was, of course, due to the industrial depression. Pennsylvania, as a matter of fact, imports a great deal of natural gas from West Virginia, and is destined to import more and more, if no substitute is found. Fortunately for our purpose, the Geological Survey has divided the consumption for domestic and industrial purposes, and in view of the industrial depression we cite only the figures for 1907 as follows:

	1000 cu. ft.	Value—Cents per 1000 cu. ft.
Domestic	44,840,748	24.2
Industrial	119,700,431	10.1

Inasmuch as the domestic consumption in Pittsburgh is steadily increasing and as West Virginia gas can be piped to many towns for domestic consumption, it would be absurd to expect that the gas would continually be sold for industrial purposes when it will bring more than twice as much if sold for domestic consumption. Indeed, a large part of the gas used for industrial purposes is produced by the manufacturers, who are not in the business of distributing for domestic consumption. Year by year the cost of finding and developing new wells increases, so that the use of natural gas for industrial purposes in the Pittsburgh district is necessarily decadent.

* * *

An oblique view is frequently taken of the hardiness, both of Connellsville coking coal and of Lake Superior iron ore. In both cases the reputation of the region was made by the best of the deposits, but in summaries of the "life" of the region the conception seems to be the product of the quality of the best by the quantity of the total. Much of the best Connellsville coking coal is gone, just as much of the best Lake Superior ore is gone. Pittsburgh industries can use an almost unlimited amount of by-product gas, and the coke requirements are equally large. Connellsville coke takes a freight rate to Pittsburgh of no less than 75 cents per ton of 2000 lb. for a haul perhaps averaging 50 miles. When blast furnaces were put in operation at Aliquippa only 20.5 miles below Pittsburgh, a rate of 99 cents was made on Connellsville coke, 24 cents being added for 20.5 miles additional haul. There is coal everywhere about Pittsburgh, most of it adapted to by-product coking, and in many cases there need be no freight charge at all. In other cases the rate would be very low, compared with the preposterous rate on Connellsville coke. There would even be an advantage in making by-product coke at Pittsburgh for consumption elsewhere, since Pittsburgh could use the gas and allow the coke to go. In cases in which the coal had to be transported to the by-product ovens a low freight rate could undoubtedly be obtained, on the principle well recognized by the railroads of permitting manufacture in transit. Observing this principle, the railroads are at the present time making very low rates on Connellsville coal shipped for coking to beehive ovens whose coal seams are worked out.

* * *

While the railroads maintain very high rates on Connellsville coke, they could be counted upon to make the lowest possible rates in connection with the establishment of by-product ovens, for the railroads are committed to the fostering of this industry in order to increase the supply of creosote obtained from coal, since they need the material for tie preservation and expect to require very large quantities. In the report of the special committee on "Conservation" presented to the annual meeting at Chicago, last March, of the American Railway Engineering and Maintenance of Way Association, one of the recommendations was: "That efforts be made to secure such improvements and economies in the construction and operation of by-product coke ovens as will lead to their general substitution for the beehive type." The immediate Pittsburgh district promises to consume regularly approximately 8,000,000 tons of coke, chiefly in blast furnaces. Surely there is reason to expect that ultimately most of the coke will be made in by-product ovens, the gas being used for various industrial purposes. Of course, the blast furnace is itself a gas producer and it is claimed that blast-furnace gas is sufficient to furnish all the power necessary to finish steel in the heavier rolled forms. There remains, however, the heating of the steel and as electric steel refining grows, employment will be furnished for all the gas from both coke oven and blast furnace.

Pittsburgh Meeting of the American Electrochemical Society.

The seventeenth general meeting of the American Electrochemical Society will be held in Pittsburgh from May 4 to 7 (that is, Wednesday to Saturday of the first week in May). Hotel headquarters will be at the Fort Pitt Hotel, quite near the Union Station. The registration of members and guests will begin at the Fort Pitt Hotel after 12 m. Wednesday, May 4. Tickets for the all-day Friday excursion and the Friday evening dinner should be obtained from the registrar when registering.

The program is as follows:

Wednesday afternoon, May 4. A visit, starting from the Fort Pitt Hotel, at 1:30 p. m., will be made to the technological testing plant of the U. S. Geological Survey. This is perhaps the largest Government testing plant in the world and its inspection will be of the highest interest to the scientific technologist, especially as the management is arranging special tests and exhibitions of particular interest to the visitors.

In the evening at 8:30 p. m. there will be an informal reception in the English Room of the Fort Pitt Hotel by the officers of the society and the local committee to the visiting members, ladies and guests.

Thursday morning, May 5. The annual business meeting of the society will be held at 10 a. m. in the Convention Hall of the Fort Pitt Hotel, where the result of the annual election of officers will be announced and the report of the board of directors will be presented.

This is to be followed by a professional session at which the following papers will be read and discussed:

Determination of the constants of materials for furnace electrodes. By Carl Hering.

Empirical laws of furnace electrodes. By Carl Hering.

On Hering's alleged laws of electrode losses in electric furnaces. By C. J. Reed.

A study in furnace conductors for heavy alternating currents. By K. C. Randall.

A new electric steel furnace. By A. L. Queneau.

On the presence and influence of gases in steel. By P. L. T. Héroult.

Thursday afternoon, May 5. Excursions and visits to the following plants have been arranged for the afternoon, all proceeding from the Fort Pitt Hotel, after lunch, each under the direction of one or more guides familiar with the plants to be visited.

Pennsylvania Smelting Company, Carnegie, Pa. (Smelting of lead silver ores in blast-furnace, refining and desilverizing base bullion, cupellation of silver, etc.)

Park Steel Works of Crucible Steel Company of America. (Puddling furnaces producing muck-bar, open-hearth furnaces producing basic open-hearth steel, crucible furnaces producing high-grade tool steels.)

Jones & Laughlin Steel Company. (Bessemer steel process, basic open-hearth process, Talbot continuous process in tilting furnaces. Rolling of merchant bars, structural shapes, plates, sheet bars, cold-rolled shafting, etc.)

Isabella Furnaces of Carnegie Steel Company, Sharpsburg. (Modern iron blast furnaces equipped with Gayley's original plant for drying the blast.)

Universal Cement Works. (Manufacture of cement from granulated blast-furnace slag.)

Pittsburgh Reinforced Brazing & Machine Company. (Brazing and repairing broken castings by special methods. Oxy-acetylene welding of heavy and light work.)

Nernst Lamp Company. (Manufacture of commercial Nernst lamps.)

Thursday evening, 8 o'clock. A session will be held in the Convention Hall of the Fort Pitt Hotel.

The retiring president, Dr. J. H. Baekeland, will deliver his presidential address on "Science and Industry," whereupon the following papers will be read and discussed:

Possibilities of cheap power in the Pittsburgh district. By F. Crabtree.

Induction furnace progress. By T. Rowlands.

Ductile tungsten and molybdenum. By C. C. Fink.

A new process for the treating of cobalt-nickel ores rich in silver and arsenic, by combined melting and electrolytic operation. By C. C. Cito.

Evolution and present status of the dry cell. By D. L. Ordway.

A new radiation pyrometer. By C. E. Foster.

Friday, May 6. All-day excursion, leaving Union Station at 8:20 a. m. on special train. Lunch will be served on the train, and the following plants visited in the order given, returning to Pittsburgh about 5:30 p. m.:

Allegheny Plate Glass Works, Glassmere.

Westinghouse Electric Works, East Pittsburgh.

Westinghouse Machine Works, East Pittsburgh.

Firth-Stirling Works, Demmler (Héroult electric steel furnace in operation).

Carnegie Steel Works, Homestead.

Tickets for the excursion may be obtained from the registrar at a cost of $1.50.

Friday evening, at 7 o'clock, an informal subscription dinner will be held at the Fort Pitt Hotel. Ladies are particularly invited.

Saturday morning, May 7, 9 o'clock, a professional session will be held in the Engineering Building of the University of Pittsburgh. The following papers will be read and discussed:

On the separation of oil from condenser water by electrolysis. By H. M. Goodwin.

A new method for electrolytic winning and refining of metals. By E. M. Chance.

An electrolytic method for the reduction of casiterite. By W. O. Snelling.

Refining of tin dross in an electric furnace. By R. S. Wile.

The conductivity and electrolysis of cuprous chloride solutions. By M. deKay Thompson.

The electrolytic reduction of nitric acid (third paper). By H. E. Patten and W. J. McCaughey.

The effect of moisture and of solutions upon the electrical conductivity of soils. By R. E. O. Davis.

The electrochemical potentials of calcium-aluminium alloys. By Jas. Breckenridge.

A rotating graphite anode. By J. W. Turrentine.

Comparison of rapid electrochemical methods. Part 1.—The Determination of Copper. By F. C. Frary and A. B. Peterson.

The electrolytic deposition of platinum—II. By Wm. J. McCaughey and H. E. Patten.

The use of perchloric acid in metal-plating baths. By C. F. Mathers.

The Whiting electrolytic cell for the production of caustic alkali and chlorine. By Jasper Whiting.

The Townsend cell; its behavior under various current densities. By H. K. Richardson and R. L. Patterson.

Lunch will be taken in the Hotel Schenley.

Saturday afternoon, 2:30, a general public meeting will be held in Carnegie Music Hall, when an illustrated address on the present status of the electrochemical industries will be delivered by Dr. J. W. Richards, secretary of the American Electrochemical Society.

This will be followed by Dr. John A. Brashear on the advantages of Pittsburgh as an electrochemical center.

Finally, an address on the conservation and utilization of natural sources of power will be made by Mr. John H. Finney.

This meeting may be conveniently followed by visits to the Carnegie Institution and Museum, Carnegie Technical Schools,

University of Pittsburgh, and other points of interest in the vicinity.

For the evening arrangements will be made for a social gathering of members and guests who do not leave by the early evening trains.

A ladies' committee will take care of the entertainment of visiting ladies. Ladies are invited to attend the session of the society on Thursday morning and evening and Saturday morning, and are specially invited to the general public meeting in Carnegie Music Hall on Saturday afternoon. Ladies are also invited to attend the visit to the U. S. Testing Plant on Wednesday afternoon and any of the visits scheduled for Thursday afternoon.

Ladies are particularly invited to be present at the informal reception on Wednesday evening, at the informal dinner on Friday evening and also to the lunch at Saturday noon.

Ladies are not invited to accompany the all-day excursion on Friday; on this day the Ladies' Committee will provide a pleasurable program consisting of automobile rides, visits to art galleries, lunch, etc.

San Francisco Meeting of the American Chemical Society.

The summer meeting of the American Chemical Society, to be held in San Francisco from July 12 to 15, 1910, promises to be one of the pleasantest outings ever enjoyed by the members of the society.

A special train made up of the Santa Fé's finest equipment will leave Chicago on the evening of July 4, arriving at Colorado Springs on the morning of July 6. About six hours will be allowed for a trip to Manitou, the Garden of the Gods or to the top of Pike's Peak. Leaving about 1 o'clock the train will reach Adamana on July 7 and a half day will be spent in a visit to the Petrified Forests, two of which and possibly three may be examined. Leaving Adamana that night the party will arrive at the Grand Canyon of the Colorado on the morning of July 8, where the day will be spent. Leaving the Grand Canyon that evening the train will arrive at Redlands and Riverside, Cal., the following afternoon and about two hours and a half will be given to the semi-tropical scenery of each of these two cities.

Sunday, July 10, will be spent at Los Angeles, leaving there in the evening and arriving at Lang, Cal., on the following morning. At Lang the borax mines will be visited on invitation of Mr. S. T. Mather, of the Thorkildsen-Mather Company, where the party will be their guests until about 1:30 o'clock, when the train will leave for Santa Barbara, giving us about five hours in that unequalled seaside resort. During the night the train will leave for San Francisco via the coast route of the Southern Pacific, probably reaching our destination about 12 o'clock of the 12th.

The meeting will follow and entertainment by the California Section. The tentative program for the entertainment includes: First, a steamer trip around the bay and out through the Golden Gate; second, a trip to the top of Mt. Tamalpias and to the Muir Woods, the first giving an extensive view of the ocean, the bay and the surrounding mountains and hills, while the second contains fine specimens of the coast redwood (*sequoia sempervirons*); third, an excursion on the Ocean Shore Railway to Pescadero with a possible return via the Santa Clara Valley; fourth, an excursion to the vineyards and wineries of the Italian-Swiss colonies in Sonoma County; fifth, a visit to the University of California at Berkeley; sixth, a visit to Stanford University with a possible automobile trip through the orchards of the Santa Clara Valley; seventh, a camping-out trip for one night and parts of two days into the Big Basin, the State Park, where some of the biggest redwoods are to be seen; also it is hoped to visit some of the local manufacturing plants.

Following the meeting the party will dissolve as a whole, returning as they desire, either via the smelters in Utah and Colorado, via the beautiful scenery of the Canadian Pacific, or via the National Yellowstone Park.

Unusually low rates have been obtained from Chicago; namely, $62.50 for the round trip from that city with $15 extra if the party returns via the northern routes. There will be $6.50 extra railway fare on the side trip to the Grand Canyon. The berth rate from Chicago to San Francisco will be $14, with an additional charge for the four extra days in transit in lieu of hotel expenses, as the Pullmans will be used throughout the trip. This additional charge will approximate $7 on the berth rate.

The Puget Sound Section are hoping that a considerable number of the members may decide to return via Seattle.

In view of the efforts that are being made by the California members and of the unusual attractions of the trip, it is hoped that a special effort will be made by Eastern members to be present at the meeting. Reservations for the special train will be made in the order of their receipt. Any members of allied societies going West at this time who may wish to share in the privileges of the special train should address the secretary, Dr. Charles L. Parsons, New Hampshire College, Durham, N. H.

Successor to Dr. Chandler Appointed.

The trustees of Columbia University have announced important changes in the department of chemistry made necessary by the announced retirement of Prof. Charles F. Chandler. The administrative duties of the department will be assumed by Dr. Marston T. Bogert and a new chair of industrial chemistry has been created to be filled by Mr. Milton C. Whitaker. Dr. Bogert, a past-president of the American Chemical Society, has been associated with Dr. Chandler as professor of organic chemistry, and will now be administrative head of the department. Mr. Whitaker has been superintendent of the Welsbach Company's works at Gloucester, N. J. In 1897 he graduated from the University of Colorado and entered Columbia University as a graduate student, where he engaged in research work in the department of chemistry until 1902, when he entered the employ of the Welsbach Company. Mr. Whitaker's work in the department will begin next school year. The trustees also announce the acceptance of the fund raised by the alumni of the University to establish a lectureship in honor of Dr. Chandler.

Enlargement of Ingot Iron Works.

Elsewhere in this issue will be found an article by Mr. James A. Aupperle, chief engineer of the American Rolling Mill Company, of Middletown, Ohio, on their American ingot iron. In this connection the following notes on considerable enlargements of the works of this company should be of interest.

The American Rolling Mill Company is erecting a new plant at Middletown, Ohio, the cost of which when completed is estimated at $3,300,000. It is on a lot of 475 acres, one mile east of the present 50-acre plant, and connected by private right-of-war. The P. R. R., Big Four, and C. H. & D. offer excellent shipping facilities.

The work being done at present is largely grading, and a couple of steam shovels are busily engaged making the change of some seven feet, which is the average for the entire plot.

One of the unusual features of the new plant is the elimination of the narrow-gauge "dinkey" service. All trackages throughout will be of standard gauge.

The plans of the proposed equipment call for four 65-ton, open-hearth furnaces; one, 40" blooming mill; one 24" billet mill; two jobbing mills; eight sheet mills, together with a complete cold rolling department for the manufacture of uniform color sheets. For the galvanized sheets, eight pots will be provided, the hot process, of course, being employed.

At present the company buys pig iron, but the ultimate plan calls for the erection of blast furnaces to furnish all necessary stock.

The entire new plant, with the exception of the blooming and billet mills, will be electrically driven, and Westinghouse apparatus is to be installed throughout.

Three turbo-generators, each of 2150 kilovolt-ampere capacity (1500 kilowatts at 70 per cent. power factor), will furnish the power. The turbines are high-pressure and low-pressure, the former taking steam direct from the boiler plant and the latter using the exhaust from the engines on the blooming and billet mills. In case of shortage in the latter, automatic interconnection of the units permits the admission of live steam direct to the low-pressure turbine. This device also gives a very great overload capacity. The low-pressure turbine installation offers obvious advantages, as the power for the largest rolling mill engines can be obtained at a very high total thermal efficiency.

The sheet and jobbing mills are to be driven, respectively, by one 1500-hp and one 1000-hp alternating-current motor, each operated at 2200 volts and 25 cycles, the current coming direct from the alternators.

The cranes and tables will all be operated by direct current, the well-known Westinghouse "mill-type" motors being used and current furnished at 250 volts by two 750-kw rotary converters.

Shears, conveyors, and such other machinery as is in continuous operation will be driven by alternating-current, 220-volt induction motors.

The electrical installation is in charge of Mr. W. J. Beck. It is hoped to have the new plant running in less than a year, or about February, 1911.

New Laboratory.—The care and rigid inspection necessary in the manufacture of "American ingot iron" has resulted in the erection, now under way, of a new laboratory of research engineering.

This building is 40 x 60 ft., and will be provided with the most modern equipment on the market for conducting electrical and physical tests, as well as complete apparatus for the chemical and microscopical examination of irons and steels.

One section of the basement will be used for making accelerated corrosion tests, in which samples of iron will alternately be subjected to salt water and heated air. The automatic device for accomplishing this, which has been designed by Mr. J. A. Aupperle, chief engineer, consists essentially of a slowly revolving Ferris wheel about seven feet in diameter. A companion wheel will dip into fresh water.

Another feature of the corrosion investigations will be the carbonic-acid moisture tests in which a measured volume of illuminating gas is burned to produce moisture and CO_2 as suggested by Professor Charles F. Burgess, of the University of Wisconsin.

We hope to be able to give a more detailed description of the laboratory later. With its equipment it will cost $40,000, and is to be ready for service about August 1, 1910.

A New By-Product Coke Oven Plant.

The Bethlehem Steel Company has recently closed a contract through Messrs. Frederick J. Mayer and David B. Carse, of New York, with the Didier-March Company, a concern whose stock is largely held in Germany for the construction of a by-product coke oven plant which will be capable of coking about 4300 tons of coal per day, and an ultimate capacity of about twice this amount.

It is understood that the ovens used will be of the double underflue type, so arranged that there will be no reversing of the direction of the gases or the products of combustion. The ovens will have a capacity of about 10 tons each, there being in the first installation four batteries of 75 ovens each, or a total of 300 ovens.

This contract was made in the interest of the Berlin Anhaltische Maschinenbau Aktien Gesellschaft of Berlin, Germany, and the Stettiner Chamotte Fabrik Aktien Gesellschaft of Stettin, Germany. These concerns are the largest manufacturers in Europe of gas works, transmission machinery, by-product apparatus, and refractory materials. The Bethlehem plant will, therefore, be of the most modern type, with the latest machinery for the handling of all the materials.

It is understood that there will be installed a new process for the direct recovery of ammonia, a large briquetting plant, and apparatus for the distillation of tar. The coal storage will take care of 30 days' consumption at the rate of 8600 gross tons of coal per day, or sufficient for the doubling of the plant, which will soon be built.

It is reported that the interests building this plant are now active competitors for by-product plants in this country.

The United States vs. Anaconda Copper Mining Co.

Apropos the smelter fume problem which has occupied the attention of smelters and farmers for several years, it is interesting to note that the Federal Government has taken a hand for the purpose of protecting national forest reserves from alleged damage, and has sued the Anaconda Copper Mining Company, praying for an injunction to close the Washoe smelter in Montana. The Government's intention is to force the company to make such disposition of sulphur and arsenic fumes as shall render them innocuous. Experts for the Government claim that the outcome of other suits in California and Georgia has been an improvement in smelting methods, and will endeavor to force similar improvements in Montana. The Department of Justice has announced that the Montana suit will be pressed for a decree, unless Government experts can cooperate with the Anaconda company in terminating present conditions.

Institute of Metals.

The Council of the (British) Institute of Metals have decided to initiate what is hoped will be an annual series of May lectures.

The first of these will take place in London on Tuesday, May 24, when Professor Gowland, F.R.S., vice-president of the Institute, will deliver a lecture on "The Art of Working Metals in Japan," which will be illustrated by diagrams and lantern slides.

Chemists' Building in New York.

The Chemists' Building is progressing rapidly. The excavation of the site at 50-54 East Forty-first Street is nearly completed, and the bids of W. L. Crowe Company for the general construction, and of Chas. L. Eidlitz & Company for the electrical work have been accepted. A pamphlet will soon be ready for distribution among the membership of the Chemists' Club and other chemists, which will contain floor plans and other exact information. Barring unforeseen hindrances, the building should be ready for occupancy in January or February, 1911.

The chief features of the club's quarters have been previously detailed (see, especially, page 4 of our January issue), perhaps, however, some agreeable surprises will await the visitor. It has been found possible to carry out the architects' original plans in all essentials, and the building promises to be both useful and attractive.

Chemists should consider carefully the advantage of renting laboratories in the Club Building. So far, every profession has profited by concentration in a definite locality. The new Chemists' Building may seem to some rather distant from their old-time haunts; but within a year, their clients will be moving into its vicinity. The new Chelsea docks, as well as the Penn-

sylvania Station are causing merchants to consider Thirty-fourth Street the probable wholesale center; manufacturers are moving to Long Island and New Jersey; even the courts must inevitably find an up-town site.

The rentals will be moderate, and the arrangement of the flues, supply mains and chemical waste lines makes it possible to equip the laboratory according to one's fancy or needs, with the least expenditure of time and money. Then there is the proximity of a large consulting library. The possibilities of social and intellectual intercourse with men in the same line of work are multiplied to such an extent that the commercial chemist would find himself in a truly academic atmosphere.

A project will be submitted to the New York Section of the American Chemical Society at its next meeting, which, if adopted, will greatly enlarge the usefulness of the Chemical Museum for research, and here, again, proximity to a scientific center will prove advantageous to every tenant.

The Subscription Committee is not yet able to report the sale of all the stock, as $25,850 are still to be disposed of, and it is the duty of all those who desire to see the new building a success to relieve the burden of its managers as far as possible, and to do their best to interest friends in the enterprise, if they feel themselves unable to take a considerable number of shares.

The directors of the building company are Dr. Morris Loeb, Dr. C. F. Chandler, Mr. Albert Plaut, Mr. W. H. Nichols, Jr., and Dr. Leo H. Baekeland.

New York Section of the Society of Chemical Industry

At the meeting of March 25 of the New York Section of the Society of Chemical Industry (Mr. Maximilian Toch, chairman) an interesting lecture on color photography was delivered by Frederic E. Ives.

Messrs. L. H. Baekeland, Charles Baskerville, David Wesson, C. A. Doremus and Gustave W. Thompson were elected by ballot to serve on the executive committee for the ensuing year.

News from Niagara Falls

While the old established electric furnace industries of Niagara are in flourishing condition, interesting progress is also reported from the youngest of the large electrochemical works, the Hooker Electrochemical Company, which operates the Townsend process for the production of caustic soda and chlorine by electrolysis of brine. The chlorine is used in various ways, the chief product being bleaching powder; but electrolytic chlorine is now also attracting attention for metallurgical purposes.

Of special interest is the successful use in continuous commercial operation for several years of cell units with 2500 amp capacity and of very compact construction, and a recent large addition of units of 5000 amp capacity, illustrating the practical applicability of the principles of the Townsend cell. It may be mentioned that very favorable comment has been made on the cell by technical experts from Europe, who have visited the plant of the Hooker Electrochemical Company.

From the Canadian side it is reported that the works of the American Cyanamid Company in Niagara Falls, Canada, are in satisfactory commercial operation. The manufacture of commercial fertilizers from calcium cyanamide is stated to differ in various respects from European practice. The process of the American Cyanamid Company has been worked out to suit exactly the conditions of this country, especially the demand for a so-called complete fertilizer.

From Welland, Welland County, Ontario, it is reported that at the works of Electro Metals, Ltd., where ferrosilicon is principally made, extended experiments have recently been made on the baking of very large amorphous carbon electrodes. As is well known, this is a problem the solution of which increases in importance and difficulty with the increase of size of electric steel furnaces now being erected in various parts of the country.

The Iron and Steel Market.

April has brought fresh disappointment to the iron and steel trade, not because it did not bring an improvement in the market, but because it did not bring as great an improvement as was expected. While in the preceding months of the year actual engagements fell short of production, it is probable that on the whole fully as large a tonnage of finished steel was sold in April as was produced. In a sense the industry was "breaking even," but the spring is naturally the time for heavy buying, and unless general business conditions improve, buying is likely to be lighter in June, July, and August.

Upon the markets for wire and pipe there rested a heavy burden upon the opening of spring. In the closing months of last year buying in these lines, particularly by jobbers, was very heavy, and very large stocks were accumulated by reason of the general promise of the steel industry of higher prices this year. Through the winter the mills continued at full pace, and accumulated stocks of their own, so that the spring trade had two sets of stocks to liquidate. The movement into actual consumption was probably fully up to normal, taking care of the jobbers' stocks, but as there has been no particular prospects of price advances the jobbers have not bought heavily, and the mills are left with stocks. In no other branch of the finished steel trade has there been any curtailment of production, so far as can be ascertained, but in pipe and wire there has been curtailment, the Lorain pipe plant being closed during the early part of April and running at reduced output the remainder of the month, while the largest independent wire plant in the Pittsburg district closed April 9 for a short period to reduce stocks.

Other branches of the finished steel market have had sufficient vitality to maintain full production and the present outlook is for a continuance of full output for several months. At the same time the market entirely lacks the snap which has characterized the recent bursts of activity. In the three years of continued high pressure, from late in 1904 to late in 1907, mills were steadily falling behind in deliveries for more than two years, and prices were advancing, although not rapidly. Although business suddenly collapsed at the end of October, 1907, there was an advance in wire products on September 3 preceding, and a few months prior to that, on May 14, there was a large advance in pipe. At present the market shows none of that advancing tendency, although demand and production are 25 per cent. greater than in that period. The difference is made by capacity having increased just a trifle more than demand.

So far at least there is no evidence that the market is suffering from the fact that all efforts at artificial control were abandoned over a year ago. It is quite likely that if there had been artificial control during the second half of last year prices could not have been advanced as rapidly as they were by the eagerness of buyers to take what they considered bargains. They would probably have doubted the permanence of the advances, and this year, with demand less than was expected, they would probably have felt that prices were being artificially maintained, and would have bought less freely than they have in the actual circumstances, when it is known that each producing interest is working by itself.

Finished steel prices are substantially the same as at the opening of the year. From time to time there has been shading in some products, notably sheets and wire products, but the market has not become quotably lower. In structural shapes there has been a gradual weakening, by $1 a ton, and this concession, made early in the year only by Eastern mills, has in the past month become quite general.

Pig Iron.

From November 1 to April 1 pig iron production has averaged

each month at a greater rate than 31,000,000 tons a year, while October fell only a trifle below that rate. February showed the highest rate, 31,600,000 tons; March showing a drop of 400,000 tons to 31,200,000 tons. In no calendar year has as much as 26,000,000 tons been made, so that the industry has been running at a stupendous pace. The decrease in production in March was wholly by steel works, the merchant furnaces increasing their output. This was unnatural, for the merchant furnaces have been accumulating stocks, and should have reduced production. They have been doing so of late, several stacks having gone out in April, while more are certain to go out in the next two months. Prices have continued to weaken, although very slightly. Prices at valley furnaces (90 cents higher, delivered Pittsburg) stand quotable at $16 for basic, $17.50 for Bessemer, $15.75 for No. 2 foundry, $16 for malleable and $15.25 for forge. The Southern market has dropped 50 cents since last report, being quotable at a minimum of $12. At this figure there is the first definite resistance to decline. The Southern furnaces, however, have a larger margin than have the Northern furnaces, which must buy ore and coke. Some Central Western furnaces have not bought ore for the season, and perhaps will not unless conditions improve, preferring to go out of blast. For the first time the competition of furnaces owned or controlled by ore interests promises serious results to the purchasing class, for there is now a large production by them, and more are being built by ore interests, for no less than eight merchant furnaces at Chicago, Detroit, Cleveland, Josephine and Buffalo, are being built by Lake Superior ore interests, two of these, at Detroit and Cleveland, being practically completed.

Steel.

Bessemer steel has weakened, while open-hearth steel is, if anything, stronger than it was a month or two months ago. The weakness in Bessemer steel is directly attributable to the light demand for rails. A Bessemer steel interest just east of Pittsburg has lately become a producer of sheet bars and entered the Pittsburg market early in April with cut prices on both billets and sheet bars. Prices are roundly quotable as follows: Bessemer billets, $27 to $27.50; sheet bars, $27.50 to $28; open-hearth billets, $28.50 to $29; sheet bars, $29 to $29.50; all f.o.b. maker's mills, Pittsburg or Youngstown districts. Rods are nominally $33, Pittsburg.

Wages.

Early in April the Pennsylvania Steel Company, for its own operations at Steelton and the Sparrows' Point operations of its subsidiary, the Maryland Steel Company, announced a wage advance of about 6 per cent. Shortly afterwards the United States Steel Corporation announced an advance, for May 1, averaging about 6 per cent. for its blast furnace and steel works employees in the North. There had been an advance in Connellsville coke wages January 16 and an advance in the Lake Superior ore region at the beginning of mining. The question of advancing wages in the South, and of advancing wages upon its railroads, was left undecided. Other iron and steel interests in the Central West will presumably be moved by the incident to make corresponding advances. Many of the railroads have made wage advances averaging about 6 per cent.

These wage advances are based upon the unrest of labor and the "increased cost of living." As shown by the Index numbers of commodity prices, there has been a decided increase in the cost of living since 1907. The steel industry, however, is obtaining from $3 to $5 a ton less for its products than in 1907, and while prices were slightly higher, on an average, than in 1906, there was a general wage advance at the beginning of 1907. There has been no wage reduction since 1907, except that in the spring of 1909 there was an abortive movement to reduce wages, but former wage scales were soon restored.

Thus the steel industry is paying higher wages and getting less for its product. The trend is a very unfortunate one. The cost of steel production is altogether too high for the circumstances in which the industry finds itself.

Finished Products.

Regular prices, f.o.b. Pittsburg, are as follows:

Standard steel rails, $28 for Bessemer, $30, for open hearth; f.o.b. mill, except Colorado.

Plates, $1.55 for tank quality, ¼ in. and heavier.

Shapes, $1.50 to $1.55 for I-beams and channels, 15 in. and under, zees and angles, 2 x 3 and larger.

Steel bars, $1.45 to $1.50, base.

Iron bars, $1.60, Pittsburg; $1.50 to $1.60, Philadelphia; $1.50, Chicago; $1.40 to $1.45, St. Louis.

Wire nails, $1.85, base; plain wire, $1.65, base; galvanized barb wire, $2.15.

Black sheets, 28 gage, $2.40; galvanized, $3.50; blue annealed, 10 gage, $1.75 to $1.90; plainted corrugated roofing, $1.70 per square; galvanized, $3.

Tin plates, $3.60 to $3.75 for 100-lb. cokes.

The Non-Ferrous Metal Market.

Practically no interesting features have characterized the base metal market for the past month, and no transactions worthy of note have been reported. Temporary improvement has been noted on one or two occasions, but there has been a general evidence of weakness. The feeling prevails that the delay in the anticipated spring buying has contributed to the lack of firmness in the market.

Copper.—The only feature in the copper market in the last few weeks was the sharp decline in lake on April 11. This followed a period of dullness, and was due to the decision of the Calumet and Hecla to offer copper at 13c. instead of 13.75c. The effect was to stimulate sales, both foreign and domestic, and some good orders were booked. The action of the Calumet and Hecla may have been influenced by the American statistics for March, which were regarded as unfavorable on account of the increase in the world's visible supply. Electrolytic has shown the same features as lake.

Tin.—The market for tin has been unsteady and unsatisfactory, with practically no demand. Domestic sellers recently offered the metal below the import basis, but buyers did not take much for future delivery. Prices have varied a little during the month, but at the close of the market on April 13 tin could be bought at 32.40c.

Lead.—This metal has shown a steady decline for several weeks. The reluctance of the consumers to buy at the prices offered has resulted in a dull market. The reductions made by the American Smelting and Refining Company have been met promptly by other sellers, but no large sales have been made. At the close of the market April 13 lead was easy at 4.35 to 4.40c. New York, and 4.20 to 4.22½c. St. Louis.

Zinc.—Nominal prices have been the rule for some weeks, and liberal offers failed to find ready buyers. Prices have dropped steadily until the close of business April 13 showed spelter offered at 5.50c. in New York and 5.35c in St. Louis. Other metals except aluminium have shown little or no feature. Business has been active in all branches of the aluminium trade and the production is being bought up steadily.

CORRESPONDENCE

Electric Steel Refining.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—The article in your April issue on the South Chicago Electric Furnace Plant of the U. S. Steel Corporation is of considerable interest. After we have heard so much as to what could or might be done with the electric furnace in the electrometallurgy of iron and steel, it is refreshing to read a simple statement of what is being done in South Chicago with the 15-ton Héroult steel-refining furnace on a thoroughly commercial scale in every-day work.

While the object of your article was evidently to state

simply facts, without any theoretical discussion, some interesting deductions can be drawn directly from the facts given in your article. The question has often been asked what is the rationale of the success of refining steel in the electric furnace and the answer has usually been that two causes are contributory to success—the possibility of producing a very high temperature and the neutral or reducing atmosphere of the electric furnace.

Now, the figures of temperature given in your article show conclusively that the possibility of producing a very high temperature in the electric furnace is not utilized for steel refining, at least not in the practice of the Illinois Steel Company at South Chicago. The steel in the electric furnace is not much hotter than when it comes from the Bessemer converter. Hence the success of refining steel in the electric furnace should be attributed essentially and primarily to the neutral atmosphere alone, which permits reactions (deoxidation and the specific reactions in the Héroult furnace by which sulphur and phosphorus are removed) which are impossible in oxidizing furnaces.

Chicago, Ill. M. N.

* * *

Rapid Electro-Analysis with Stationary Electrodes.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—I notice in your April issue an abstract of an article by Perkin and Hughes in which they describe some new forms and details of rotating electrodes for the quantitative determination of metals by electrodeposition. A number of articles have appeared lately dealing with this subject, interest being aroused to a great extent no doubt by the advantage, and in many cases by the necessity, for more rapidity.

This being the case it seems strange that an article by Dr. John T. Stoddard, which appeared in the *Journal* of the American Chemical Society for March, 1909, in which he brings out the merits of the stationary wire gauze cathode, should have received apparently so little attention. This article has been abstracted and, I believe, reprinted in full in the *Chemical News*. Also, other experimenters, for instance L. S. and R. C. Palmer, in a paper read in May, 1909, before the American Electrochemical Society, have confirmed to a large extent the results obtained by Dr. Stoddard, but in spite of this publicity many investigators, notably Perkins and Hughes, do not appear to be aware of the method as suggested in that article, or, if they are, do not consider it of importance.

In the opinion of the writer it is worth very serious consideration, especially as Dr. Stoddard has worked out the details as they occur in actual laboratory work quite thoroughly, and in adapting it to the work in this laboratory few changes were made, and these only in some comparatively unimportant details.

Some time previous to the appearance of this paper the increase of work in this laboratory necessitated some more rapid method for the determination of copper, and the rotating anode was tried with both a platinum dish and a mercury cup for the cathode. In spite of the time saved by the rapid precipitation it was soon recognized that in a laboratory connected with a manufacturing plant where "results" are wanted with as much rapidity as is consistent with accuracy, rather than "material" for papers, the apparatus necessary to the rotating method was not, to say the least, desirable.

Dr. Stoddard's article appeared about this time and was recognized as a step forward, provided it proved practical and accurate, which it has.

In this laboratory the gauze cathode is the same as described by him, and is a cylinder 1¼ in. in diameter by 1¼ in. in length, made of 52 mesh platinum wire gauze. At each end of the cylinder the gauze is folded over a platinum wire 0.02 in. or 0.03 in. in diameter, and the whole soldered to a platinum wire stem about 0.05 in. in diameter and about 6 in. long, the cylinder complete weighing about 10 grams.

The anode is a platinum wire (0.05 in. in diameter) spiral with seven or eight turns in a length of ¾ in., with enough extra wire to make a straight stem 2 in. or 3 in. long. The outside diameter of the spiral is 7/16 in.

This anode is different from that described by Dr. Stoddard. It was intended to follow his description and use a platinum foil cylinder for the anode, but through some oversight none was on hand when everything else was ready for a start, so in order to get quick action a platinum wire was coiled up in a spiral form and pressed into service. It was so successful, appearing to possess the slight advantage of allowing better circulation of the electrolyte, that it has been retained.

At first, naturally, strong currents and rapid deposition were in order. With a current of 8 amp (furnished by a storage battery), or even more sometimes, the copper in a 1 gram sample of brass containing 85 per cent copper was precipitated in 12 or 15 minutes. It was found after a time, however, that this speed carried disadvantages—disadvantages, by the way, which are not necessarily overcome by the rotating electrode. The last traces of copper were apt to be left in the solution, and it required close attention, on the other hand, to see that the strong current did not decompose all the acid, indicated by a tendency to foam, and in this way spoil the work.

The current was dropped to 6 amp and the time at this rate was about 20 minutes, with better results. Gradual changes were made, based on more experience, until now the start is made with 4 amp (increased to five or six for the last few minutes), and the time allowed is 30 minutes.

At this rate, working on a 1 gram sample, it is very rare indeed that the last traces of copper are not removed, whether it is a yellow brass containing 60 per cent of copper, or a sample of ingot copper containing practically 100 per cent, and on the other hand, there is no danger of the acid being all decomposed. It is just as simple as, and requires no more attention than, the old over-night method, and yet is comparatively rapid.

The deposition of the copper is made in a volume of 60 c.c. to 70 c.c. and at a temperature nearly or quite boiling. In the regular course of a brass analysis the electrolyte would contain 5 c.c. of concentrated sulphuric acid minus what would be lost in evaporating for the lead determination, and to this is added ½ c.c. strong nitric acid. Under these conditions the precipitation is very simple and satisfactory.

The method is equally advantageous in the determination of nickel which is deposited under practically the same conditions as copper, with the exception that the current is usually on a few minutes longer. There has been little occasion or opportunity to experiment with metals other than the above, but Dr. Stoddard's work would indicate that it can be employed to advantage on other metals.

ALLEN P. FORD.

Laboratory, Crane Valve Company, Bridgeport, Conn.

* * *

Efficiency of Dryers.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—Mr. Ruggles' reply to my criticism—not of the dryer test,—but of a misuse of the word "efficiency" in describing the performance of the machine hardly seems sufficiently to justify the new and unprecedented meaning with which this term has been invested by him.

I believe it will hardly be questioned that the definition I have given of "efficiency," as applied to the performance of any machine, conveys its meaning as universally understood by engineers and scientists, and any technical man on hearing that the efficiency of a machine was 85% would instinctively understand this as being meant in the ordinary and accepted sense.

Mr. Ruggles is perfectly correct in saying that he had fully stated his position as regards "loss of heat in dried material,"

in a preliminary paragraph, but while as he says, "there could be no possible deception," there is, in this misuse of words, abundant opportunity for grave misconception as to the performance of the machine,—the more so as the article was introduced by a comment on the remarkable efficiency attained.

True, the discussion is one of terms. What other form could such discussion assume? It may be open to the stigmatization of being "academic," but not necessarily therefore unprofitable, for, if it be legitimate to use one word in a novel and entirely arbitrary sense, widely differing from its accepted technical usage, where shall the line be drawn? Why may not the next account of a dryer test use B. T. U. with a new and special value to suit the exigencies of the case?

While the discussion may be academic, the correct understanding of the performance of a machine is intensely practical. With some uncertainty as to whether "academic" or practical, I would point out that Mr. Ruggles' defence of his method of computing "efficiency" (in his second paragraph) is not valid, nor is the distinction drawn between "necessary" and "un-necessary" losses a correct one.

The losses (departures from theoretical perfection) in any well designed machine are all necessary. Good design reduces losses to a minimum. Unnecessary losses are, de facto, the marks of poor design.

Loss by radiation in any heat-using machine is, practically speaking, a "necessary" loss. It can, in no practical way, be avoided, but, by good design, may be reduced. Similarly in any fuel-burning machine loss by unconsumed fuel is unavoidable and, therefore, necessary, but, by good design of furnace, it may be reduced to a minimum.

In a dynamo, loss by heating of the armature is "necessary," and so on indefinitely. With precise similarity, loss of heat in dried material is inseparable from the operation of any practical dryer, but may be minimized by modified design or mode of operation.

In every case cited, in speaking of the efficiency of the machine we mean its utilization of energy *in useful work after deducting all losses* and not after crediting the machine with one of the major losses on the ground that it is necessary.

Nor is Mr. Ruggles correct from a purely practical standpoint in assuming that "it is necessary to heat such material (sand) to 212°." Nor again, that "to attempt to dry sand, etc. * * * at a low temperature by absorption would be absurd on account of the small capacity such a process would give." If I correctly understand the last quoted sentence, "drying at a low temperature by absorption" means drying under such conditions that all the moisture is carried off as vapor by the air (or gases) passing through the dryer, such gases being discharged partly or completely saturated at a temperature below 212°. If this understanding be correct, such a process would by no means be absurd; nor, to attain a capacity equal to that described in the test under discussion, would it involve any extraordinary conditions—either as to size of apparatus, initial temperature, volume of air, or in any other way. Furthermore, it is entirely practicable to discharge the dried material at a temperature far below 212° and hence greatly to diminish the "loss of heat in dried material."

This loss, which in Mr. Ruggles' case far exceeds any other, is, as I have already pointed out, one of which the magnitude is largely dependent upon the design of machine—one which can be materially reduced and which differs in no essential from any other source of loss. There is absolutely no legitimate reason for according to it a special treatment.

Mr. Ruggles' position would be correct only if the dryer be considered as a machine for *heating* sand. Such is not its true function, which is purely evaporation of water, and any heat diverted from the true function of the machine is wasted as regards the specific end to be attained and, therefore, detracts from "efficiency."

The 85% efficiency claimed represents no real thing. It stands for no actual tangible accomplishment of the dryer and conveys no correct concept of the merit or demerit of its performance. The 85% efficiency is a fanciful and ideal figure, compounded of the actual useful performance in drying, and the useless waste of heat in dried material. If it means anything at all, it must be regarded as expressing the compound efficiency of the machine as a dryer and a sand heater.

"Efficiency" has hitherto been a useful and respectable member of the community of words, but we may well shudder for its future reputation, if such hybrid and illegitimate concepts are to be fathered upon it.

To sum up:—My letter in your April issue was a protest against using "efficiency" in a sense differing radically from its accepted technical meaning,—a transgression that is hardly palliated by a statement of position as regards a certain loss in another part of the article.

I contend that any one reading the article in question, with its deduction of 85% efficiency, would primarily understand that the dryer had, in the test described, utilized in useful work 85% of the heat supplied to it, instead of only 54.5% as was actually the case.

Only after a somewhat careful study of the figures does the fallacy become apparent, and then only to those having some familiarity with the science of drying.

Whatever the 85% represents, if indeed it represents anything, it certainly does not express the efficiency of the machine as a dryer.

Therefore, without implying any intent of deception, the erroneous use of the word "efficiency" certainly does convey, in the article discussed, a quite erroneous idea of the actual performance of the machine under test.

Bayonne, N. J. W. E. Wadman.

* * *

To the Editor of Metallurgical and Chemical Engineering

Sir:—From Mr. Wadman's point of view, taking one particular kind of material, his position may be correct, but in drying over 40 different kinds of material under the greatest variations of moisture and temperature, there can be but one way of figuring the efficiency of dryers so as to bring them all under a uniform rating, and this is the system which I have adopted and which I maintain is perfectly fair as long as the purchaser and user understands the basis of comparison.

A great many dryers are sold for drying and heating sand in asphalt work where it is necessary to raise the temperature of the sand to 400° to mix with the melted asphalt without chilling it. In this case Mr. Wadman would credit the heat carried out with the sand and make the efficiency of the dryer 88 per cent, while the same dryer drying the same sand for a plaster mill would only have an efficiency of 54 per cent.

In many other cases the material after drying has to be heated either to calcine it, to burn out organic matter, or for some other special reason. In drying granulated blast furnace slag, the water is imprisoned in cells so that a temperature of at least 250° has to be given to the material in order to burst the cells and release the water.

In some cases, as with cement rock, the moisture to be evaporated is not over 1 per cent, in which case Mr. Wadman would only credit the dryer with 22 per cent efficiency, while the same dryer drying a precipitate carrying 60 per cent of water he would give an efficiency of 83 per cent. This is rather an absurdity when the dryer in one case is as efficient as in the other. With my method of calculation any one type of dryer will give practically the same efficiency under any or all conditions and different types may be compared even under totally different conditions. By charging the dryer with only unnecessary losses, such as imperfect combustion, heat lost in exhaust, in radiation and in dried material above 212°, and by crediting it with the necessary heat of material up to 212°, no injustice is done and the question is much simplified.

New York City. W. B. Ruggles.

The New Blast Furnace of the Bethlehem Steel Company—1.

By W. S. Landis, M.S.

On Thursday, April 7, Furnace "D" of the Bethlehem Steel Company was lighted by the veteran ironmaster, Mr. John Fritz. The company had invited several hundred representative citizens of the Bethlehems and adjacent towns, as well as students of the Lehigh University, to the interesting occasion, they meeting at the offices of the company and being led by President Schwab to the new furnace, heartily cheered "Uncle John" and President Schwab as the torch was applied. After the lighting they again returned to the offices, where a reception was held, at which speeches were made by Mr. Fritz and the officials of the company. With all his long and active career in the iron industry Mr. Fritz acknowledged this as the first time he had ever set a furnace in operation himself.

Some 50 years ago this summer the first blast furnace of the old Bethlehem Iron Company was built. Four others followed it at various intervals, the four still being in active operation though the original one has long since been demolished. These four furnaces are 18 ft. x 70 ft. and have capacities of 200 tons of basic iron per day. In April, 1906, work was commenced on a 22 ft. x 90 ft. stack in anticipation of the demands of the new rail and structural mills begun in the fall of this same year. This furnace, designed by the Bethlehem Steel Company and built by the Riter-Conley Company, was the

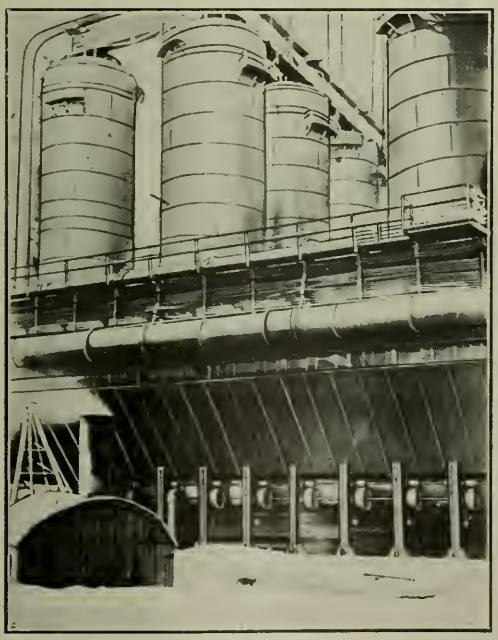


FIG. 1.—ORE BINS.

prototype of the furnace just lighted, though in lines and general arrangement only.

In 1908 the steel company determined to install an addition to their steel mills, the present 35,000 tons capacity per month not nearly sufficing to keep up with their orders. This addition is to take the form of a duplex plant, similar to the one at Ensley, Ala.

To meet the demands of this new plant for pig iron another furnace was determined upon.

Work being somewhat slack in the construction departments of the works it was determined to undertake the construction of this new furnace at home in order to keep up the personnel of the various departments during dull times. Every drawing, and every part of the furnace with the exception of a traveling crane and two motors entering into the whole construction, was built by the Bethlehem Steel Company themselves in their own shops, the construction being almost entirely in the hands of Mr. W. F. Roberts, assistant general superintendent. The entire

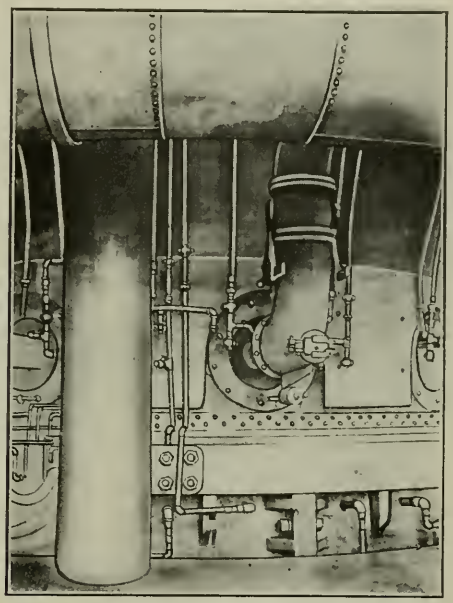


FIG. 2.—PORTION OF BOSH.

work was completed in an actual working time of eight months, and at considerable saving over contract work, both in time and money.

Although it is intended to give a complete description of the many novel features of the furnace in detail in Part II of this paper (to be published in the next issue), it might not be out of place here to call attention to the general equipment of Furnace "D." The ores used come from the Santiago district, Cuba, and are rather refractory, necessitating high temperature of blast for successful treatment. For this reason ample stove capacity in the shape of five 22-ft. x 100-ft. McClure-type stoves has been provided. To ensure these stoves working at their maximum efficiency it was necessary to provide at the same time a most elaborate system of gas-cleaning apparatus, the gases passing through a conical dust-catcher, a Mullen washer and three 12-ft. x 75-ft. Steinbart towers with interior rain-makers, before going to the stoves. Attention will again be called to this feature of the furnace.

Fig. 1 shows the ore bins with elevated railway for carrying stock from the yard to the bins. This railway is over half a mile long and extends the full length of the present furnace equipment, embracing five furnaces. Just above may be seen the stoves and to the extreme left the stack. The steam shovel in the foreground is excavating for the foundations of the new blowing house, gas engines of the company's own manufacture to be installed as rapidly as they can be built. At present the surplus power of the other furnaces is used to drive Furnace "D."

Fig. 2 shows a portion of the bosh, a tuyere connection and at

the lower left the emergency tap hole. The cooling system of the bosh is unique and will be referred to in detail later.

Fig. 3 is a general view of the interior of the cast house, the skimmer being at the extreme left. The ladle cars, carrying the metal to the mixer in a steel mill a mile away or to the casting machine, run in under the floor, the illustration showing the various runners and gates controlling the flow of metal.

The slag cars come in on a track laid in the pit between the

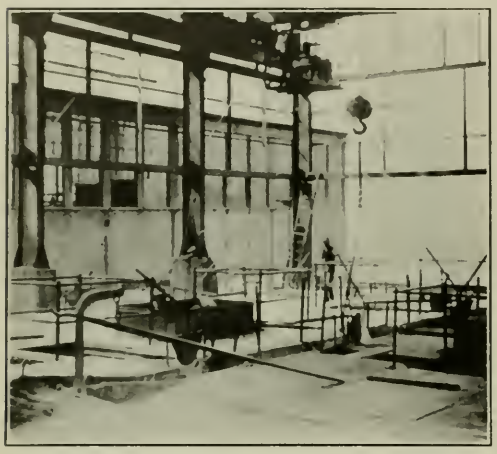


FIG. 3.—CAST HOUSE.

cast house and the boiler house on the left. A 25-ton crane has proven itself a most useful adjunct to this portion of the furnace.

One cannot help but marvel at the cleanliness of the cast house, the visit of the writer having been unannounced and no special dressing up having been done. The same cleanliness is

FIG. 4.—STEINPART TOWERS.

everywhere evident around the furnace, concrete floors with convenient sewer outlets greatly assisting in its maintenance.

Fig. 4 shows the Steinbart towers during erection. The furnace was in operation at the time this photograph was taken, the gas from the Mullen washer on the right being shunted around the towers through the large inverted siphon at the left. Similar siphons connect the washers with the gas mains, no water being used other than water admitted or run out of these siphons.

The rated capacity of the furnace smelting hard ores is 450 tons per day, using 41,000 cu. ft. of blast per minute. The total capacity of the whole blast furnace equipment at present completed is 1500 tons a day. With the completion of the duplex steel plant it is estimated that at least 2500 tons of pig per day will be required.

The company has, therefore, let contracts for two similar furnaces to "D" to the Pennsylvania Engineering Company, of New Castle, Pa., work on which has just been started. An interesting feature of this new work is the large traveler used for handling the material entering into the construction of the 10 stoves and two stacks.

Fig. 5 shows it in operation setting plates on one of the stoves of furnace "F." It is 125 ft. high, weighs 140 tons and cost alone $10,000. The boom is 90 ft. long, sufficient to reach to the furthest stove on the left. The extremely heavy bracing in the lowest story is necessitated by the fact that the traveler spans the track leading to the slag tap of furnace "E."

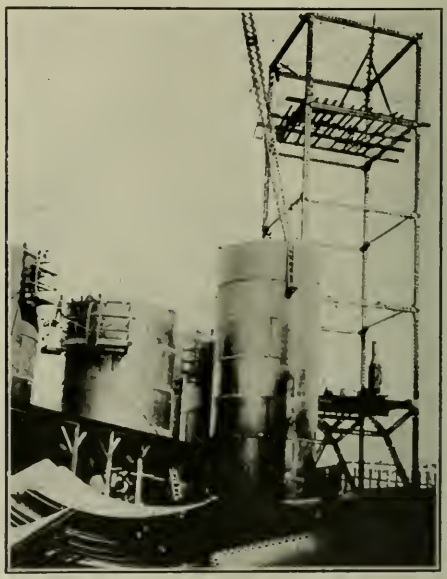


FIG. 5.—TRAVELER.

In concluding this portion of the description the charge sheet of the furnace for its first filling will be given. The furnace was scaffolded as usual, 16 ft. of cord wood being used. Next 21 rounds of coke (10,000 lb.), 21 rounds of blast-furnace slag (2000 lb.), and 21 rounds of stone, half dolomite and half calcite (1600—2100 lb.). Then 7800 burden was added per round of coke, this being increased by 400 lb. each fifth round until the full charge in the furnace, limestone increasing correspondingly 100 lb. every tenth round. At 3.30 p. m. the torch was applied. At 4 p. m. the blowing engines were started at 9000 cu. ft. of air per minute. Slag was tapped 17 hours after lighting and iron 24 hours.

Columbia Night at the New York Chemists' Club.—The smoker held at the Club House on March 26 was the second in the series of college evenings which are being given by the Club. This time it was Columbia night and a representative delegation from the faculty was present, including Dr. Chandler, Dr. Rogers, Dr. Metzger, Prof. Gies, Prof. Sherman, Prof. Coblentz and Prof. Woodhull. Prof. Morgan and Prof. Tucker had sent papers.

Electric Furnaces in Non-Ferrous Metallurgy.

BY JOSEPH W. RICHARDS.

The electric furnace is monopolizing attention to such a great extent in the metallurgy of iron and steel that its use in the metallurgy of the other metals is being relatively neglected. The interests involved in the manufacture of iron and steel are so great that any suspicion that the electric furnace is able to bring about a revolution in the manufacture of pig iron, structural steel or fine steel, focuses the attention of metallurgists on these possibilities.

Signs are not lacking, however, to indicate that non-ferrous metallurgy may likewise be deeply concerned in the development of the electric furnace. The recent great advances in electric steel furnaces have given us furnaces of great capacity, which are equally suitable for use in non-ferrous metallurgy. The electric shaft furnace for pig iron manufacture is simply an improved electric shaft furnace suitable for any kind of shaft furnace work, and may quite possibly find as large application in non-ferrous metallurgy as in the manufacture of pig iron.

The furnace operations of non-ferrous metallurgy are very diverse; the principal ones are:

(1) The melting of metals for refining and casting.

(2) The melting together of metals to form alloys.

(3) The roastng of sulphide and arsenide ores.

(4) The concentrating smelting of raw or partly roasted ores to matte.

(5) The reducing smelting of raw or roasted ores to metal.

(6) The reducing smelting of raw or roasted ores to metal vapor which is to be condensed.

I. The non-ferrous metals are in general so valuable and so easily injured in quality by furnace gases that they are usually melted in pots, crucibles or muffles. The same is true of fine steel, up until recently melted only in crucibles. The great development of electric steel melting furnaces has a direct bearing on these non-ferrous operations. What can be cleaner and neater than, for instance, the melting of gold or silver in an induction electric furnace, using no fuel, producing no ashes, employing no agent which can contaminate the metal, carrying off no metallic vapor by the stream of hot furnace gases, and with the temperature controllable to a nicety by the revolving of a resistance-box handle? And what is true of gold and silver is true, in many cases, to a greater extent, of copper, zinc, nickel, aluminium, magnesium, platinum, etc. The advantages of the electric furnace melting of these non-ferrous metals are either unrecognized or are only beginning to be appreciated.

II. The non-ferrous industries manufacturing alloys will gain particularly by the use of electric melting furnaces. As is generally recognized, it is the atmosphere of the ordinary melting furnace which plays havoc with the quality and proportions of the alloy with which it comes in contact, while the hot gases carry off considerable quantities of metallic vapor. To obviate these losses and changes in the composition, careful melting is done in closed crucibles, with all their attendant disadvantages of high fuel consumption, cost of wear and tear, and small heats.

The electric furnace provides the means for producing at one heat a ton or even several tons of alloy, all uniform in composition, and without any necessary losses by oxidation or changes in composition. In other words, just as in the steel industry, it provides *crucible quality* metal in the large heats furnished by the hearth furnaces, and at almost the price of operation of the latter.

We confidently anticipate a great extension of the use of electric furnaces in the manufacture of such alloys as are usually made to best advantage in crucibles, such as brass, bronzes, German silver, coin silver, alloyed gold, and in general all the more difficultly fusible alloys which cannot be safely melted in the ordinary iron melting pot.

III. Turning now to the metallurgical treatment of non ferrous ores and intermediate products, we consider first the roasting operation, ordinarily undertaken to remove sulphur and arsenic. These depend primarily on oxidation by the air, at a carefully regulated temperature, which will not melt the sulphides or arsenides present. These operations ordinarily require little and sometimes no fuel, because the heat of oxidation of the ore itself furnishes considerable heat.

When the ore is thus self-roasting it is self-evident that no electrical assistance is necessary or required. Where supplementary heat is needed it may in all cases be furnished with advantage electrically if it can be furnished cheaply enough. If fuel is burned to supply the supplementary heat, the fuel itself and its products of combustion interfere with the roasting operation if they are mixed with the ore and burned by the same air which does the roasting; if the fuel is burned separately and its heat of combustion transmitted to the roasting material through the walls of a retort or muffle, then such heat is applied ineffectively. That is, with but poor heating efficiency.

In the case of using electrical energy for this supplementary heating, it has the advantage of exerting no hindering action whatever upon the roasting operation, and it may be applied either by direct heating of the ore by its own resistance or by transmission from an electrically heated resistor through the walls of the retort.

In any case, the substitution will have advantages in the roasting operation, and will be advisable if the electrical energy can be obtained cheaply enough. That will be the principal question to be considered; where fuel is dear and power is cheap, the substitution may easily be of economic advantage, especially in attaining "dead" or complete roasting.

IV. The concentrating smelting of raw ore or partly roasted ore to sulphide matte is an operation carried on on an enormous scale in non-ferrous metallurgy. There is nothing analogous to it in the metallurgy of iron with which to compare it. The gangue of the ore and fluxes unite to form a fusible slag, as in the iron blast-furnace, but the metals remain combined with sulphur and concentrate as melted sulphides, instead of as melted metal.

In some cases the charge contains so much iron sulphide (pyrites) that the heat of its oxidation by the blast is sufficient to replace all other fuel, and we have an auto-smelting operation similar in thermal characteristics to the auto-roasting previously mentioned, and known usually as "pyritic smelting." Such an operation is highly desirable where possible, because it furnishes matte and slag, utilizing only the self-contained fuel of the ore: The heat of oxidation of part of its iron and sulphur. Electric heating has nothing to do with such an operation.

In the majority of cases, however, the raw ore or partially roasted ore cannot be "pyritically" smelted, and supplementary fuel is necessary. The products of combustion of this fuel, mixed with the charge, are not favorable to the concentrating operation, since they hinder its oxidation. If electrical heating is used in such a shaft furnace, it will supply the deficit of heat needed without absorbing any of the oxygen of the blast.

Some of the electric shaft furnaces already operated would be fitted for these operations if provided with tuyeres above the electrodes in order to perform whatever oxidation of the sulphides of the charge was desired. The air supply thus furnished would have to be regulated closely to accomplish the amount of oxidation required and the production of the desired quality of matte, while the electric current would be regulated to provide the heat deficient for the smelting operation.

A furnace thus operating would probably run more smoothly than a furnace running "semi-pyritically," that is, with some fuel, because the amount of oxidation could be regulated exactly by the blast used and the heat supply by the electrical energy. It would be cheaper run electrically only where fuel was high in price and power cheap.

V. The reducing smelting of suitable raw ores or "dead

roasted" ores to metal is a different class of operation. One phase of it is distinctly analogous to the shaft furnace reduction of iron ores. The charges may be oxide of a metal, such as copper, nickel, lead, tin, antimony, even zinc together with the gangue of the ore, flux and a reducing agent.

As ordinarily operated, the charge is either heated by fuel mixed directly with the charge, as in shaft furnaces, or by fuel burned apart, as in reverberatories.

In reverberatories, enough fuel is mixed with the charge to reduce the metallic oxides, and the whole is heated by gases from a fireplace. The efficiency of such a scheme of heating is low; usually less than 25 per cent. of the heating power of the fuel burned on the grate is usefully applied to heating the charge, the rest being lost by radiation and conduction and sensible heat of the furnace gases, while the large volumes of gases superficially heating the charge usually carry off considerable metallic vapors into the chimney flues.

Electrical smelting of such a charge should be highly advantageous in respect to saving of metal, if the cost of the electrical energy compared to that of the relatively large amount of coal used made the change possible.

In shaft furnaces, fuel is mixed with the charge for the double function of chemical reduction and to furnish heat by its combustion at the tuyeres. The operation is a duplicate of iron blast furnace operation, except that in the case of somewhat volatile metals, like tin and lead, great losses may occur, while with zinc all of it would be driven out in the fume, and the shaft furnace is impossible.

For such shaft furnace reduction processes, the electric shaft furnace, as already built and operated, should prove itself very well suited. Its advantages are the absence of the large volume of products of combustion of fuel, and consequent great decrease in volatilization and fume loss; the absence of the relatively high temperature of combustion in the tuyere region, and the regulation of the maximum temperature in the furnace to simply that required for the running out of the products. These changes would increase greatly the regularity of running, and decrease to practically nothing the volatilization loss. Of course, sufficient carbon for reduction of the charge must be used, but none for fuel. The amount of electric power required would be small, even for large furnaces, since the heat requirements are not large.

Quantitative calculations can be made for any given locality, comparing the cost of electric smelting of any given ore to metal in the electric shaft furnace, with its reverberatory or blast furnace smelting. The writer is confident that there are localities where the conditions already exist which would make the electric shaft furnace smelting preferable to the older methods of non-ferrous metallurgy. The whole problem is very recent, and the questions involved have scarcely received any proper amount of intelligent consideration.

VI. Reduction to metallic vapors, with subsequent condensation thereof, is the problem *par excellence* of the zinc industry. It cannot be done in shaft furnaces, because the products of combustion of the fuel mixing with the metallic vapors prohibit their condensation. The only remedy is to enclose the ore mixture in retorts and supply the necessary heat from outside. This is highly inefficient of fuel, because the sides of the retorts are poor heat conductors, and the thermal efficiency of such furnaces is therefore very low.

Electrical heating offers great promise of revolutionizing such metallurgical operations. Electrical energy can be converted into heat *inside* the retorts, thus furnishing most economically the desired heat supply without having any but the usual gases in the retort. The problem is not a simple one. The iron and steel furnaces offer very little or no assistance in working out the problem, because no analogous problem arises in that industry. Many metallurgists are at present working on this problem, and at any time we may be able to chronicle the successful solution of the problem.

When the problem is finally solved electrically, it may result in the removal of our zinc works to cheap power supplies. Italy produces large quantities of zinc ore, for instance, but smelts practically none of it, because of lack of fuel; a successful electrical zinc furnace would enable Italy to utilize her "houille blanche," or "white coal," from the snow-capped Alps, and keep her zinc industry at home, where it should be.

We are, moreover, confident that a good working electrical zinc furnace is not far away, and when it arrives electrometallurgy will make one more change in the commercial map of the world.

All in all, may we not well say that electrometallurgical apparatus and processes are the most interesting, the most attractive, and potentially the most powerful of the chemical and metallurgical agents and reagents at our command?

Lehigh University.

Commercial Oxygen.

By Ernest A. Le Sueur.

The importance, for certain large applications, of oxygen and of atmospheres rich in that element is being more and more recognized. Strictly pure oxygen will always be relatively expensive because it must be produced by chemical or electrochemical means. It may be separated from air, by various physical methods, of a strength so nearly 100 per cent as to amount to absolute purity for all applications. The yield, however, of such an oxygen (in any physical process) compared with the total mass of oxygen in the air under treatment is so small as to render its cost excessive save for purposes consuming it by the thousands as compared with by the millions of cubic feet.

Not only can oxygen be produced in rich atmospheres at a cost a small fraction of that of substantially pure oxygen, but for all large uses the former is the only condition in which actual application of the oxygen can advantageously be made. Air contains by volume approximately one part of oxygen and four of nitrogen (and argon, etc.), and, in burning, for instance, one ton of coal, we raise at least 10 tons of diluting and useless nitrogen to the full temperature of the flame.

Now, a mixture of oxygen and nitrogen containing but 33⅓ per cent oxygen presents but two parts of nitrogen to one of oxygen, and, for a given oxidation job, throws a nitrogen burden into the reaction of only one-half of what a 4-to-1 mixture does. This is, in fact, the proper way to consider the matter, and it may be pointed out that to change a 20 per cent oxygen air to a 33⅓ per cent one by the direct addition of oxygen it is necessary to add another 20 per cent of oxygen.

An atmosphere containing 33⅓ per cent of oxygen is thus much stronger than might superficially appear, and it is, furthermore, the case that there are very few metallurgical operations to which an oxygen mixture stronger than said percentage could be supplied without producing a temperature beyond the ability of tuyeres, furnace linings, etc., to withstand. An ordinary combustion experiment in 33⅓ per cent oxygen air as compared with the same in ordinary air is of extraordinary brilliancy and might well be supposed by a person unversed in such matters to be occurring in pure oxygen.

So much by way of introduction. The processes with which the writer has had somewhat wide experience are adapted to delivering oxygen of any strength from about 40 per cent to substantially 100 per cent, but, as the latter is relatively expensive and is actually needed for but very restricted consumption, it has seemed well briefly to set out the above considerations. It is obvious that if a process delivering 40 or 50 per cent oxygen is used as a feeder to an application requiring a lower concentration, the necessary strength is readily obtained by suitable injection of air.

The work to which reference is made dates back a long time, as developments in this line go, that is to say, it is 11 years since the writer first produced oxygen (substantially pure

as well as in rich atmospheres) via the liquid condition and using pressures round 2000 lb. per square inch as the instrumentality for the necessary refrigeration. It is over nine years since he prepared rich atmospheres using the pressure of an ordinary mining air compressor, or about 110 lb.

The operativeness of the method in which oxygen is obtained in the process in question depends primarily on the fact, denied by the authorities at the time, that a "fractional condensation" from air *under moderate pressures* directly precipitates a liquid rich in oxygen. The accepted belief prior to the writer's demonstration was based on the work of Prof. James Dewar, of the Royal Institution, who had some four years earlier endeavored to investigate the relative proportions in which oxygen and nitrogen liquefy when a mass of air is exposed to partial liquefaction.

Unfortunately he arranged matters so that the only possible result was a complete liquefaction of a fraction of the air (the residue remaining isolated from the liquefaction compartment and the low temperature thereof) instead of a fractional liquefaction from the whole of it. The erroneous inference was drawn that air, in its first stages of liquefaction, could only deposit a liquid of its own composition, and the conclusion was accepted throughout that section of the scientific world that interested itself in this field of investigation. One result of this was that if a liquid, and subsequently a gas, rich in oxygen was desired to be obtained from air it was considered necessary to liquefy air as a whole and then subject it to fractional distillation.

The old process by which Linde obtained liquids rich in oxygen, which consisted in liquefying a portion of a stream of air by the combined influence of low temperature and enormous pressure and then effecting fractional distillation of a gaseous mixture weak in oxygen by a relief of pressure from 2000 lb. per square inch down to atmospheric pressure, failed to show the incorrectness of Dewar's conclusion because of the further fact, discovered by the writer, that "the higher the pressure under which fractional condensation of liquid occurs from air, the weaker in oxygen is the said condensation."

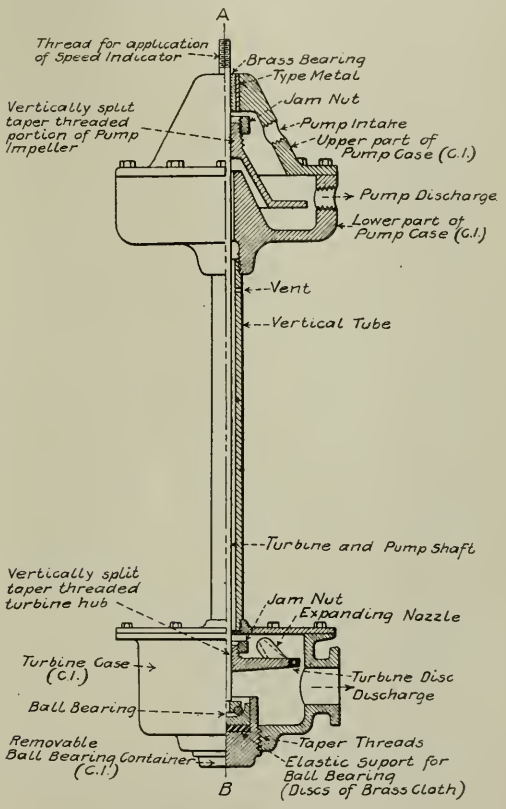
In terms of distillation the inverse holds as follows: "The higher the pressure under which a fractional distillation occurs from a liquid mixture of oxygen and nitrogen, the richer in oxygen is said distillation."

The following experiment made by me in the early days of my work in this field suffices to demonstrate the fact that the condensation, from air which has undergone a partial liquefaction, is much richer than is air. I immersed in a boiling mixture of oxygen and nitrogen much richer in oxygen than is air (and consequently boiling at a decidedly higher temperature than either the boiling or liquefying point of air proper), the lower end of an empty, unstoppered test tube of fairly large diameter (about ⅞ in.). After immersion for some minutes examination showed the tube to contain a small quantity of a decidedly blue liquid, which was found, on analysis, to contain over 40 per cent of oxygen.

The following will indicate the lines along which a cheap and efficient method of preparing fluids rich in oxygen was developed. In the first place, in order to attain anything like efficiency in the use of a gas, such as air (as distinct from a vapor liquefiable by pressure alone, like ammonia), as the working fluid of the refrigerative cycle, it is imperative that the power represented by the drop of pressure of the gas shall be abstracted as far as possible in the act of expansion. This fact has long been recognized, and Sir William Siemens attempted to liquefy air by expanding previously cooled compressed air through an engine. His apparatus was, however, totally inadequate to attain the intensely low temperature required for his purpose.

The apparatus for preparing liquids and gases rich in oxygen depends for its operativeness on the fractional condensation of a liquid rich in oxygen from air and on the development in workable form, first, of a motor adapted to abstract as much as possible of the power of the expanding gaseous residue at the point of lowest temperature, and, secondly, of a form of counter-current thermal interchange apparatus combining moderate cost with high heat interchange efficiency, and having spacious passages for the air on its way down in temperature so that trouble with clogging by aqueous frost at comparatively high temperatures and by solid carbon dioxide at very low ones is avoided.

These last requirements are almost wholly absent from apparatuses intended merely to liquefy air, because, while atmospheric air is the raw material in each case, the volumes treated by an apparatus of a given horse-power for producing atmospheres rich in oxygen are enormously greater (some 40 times) than when liquid air is the product. It was found better on the whole to allow the low temperatures developed in the process to take care of the separation of the final portions of the frost and carbon dioxide than to attempt to remove these materials completely, by any chemical treatment, from the very great volumes of air handled.



APPARATUS FOR ENRICHING AIR WITH OXYGEN.

The first motor developed in this connection (in 1900) was, in general, a de Laval type turbine disk arranged specially for the very peculiar conditions of operation. As might be imagined, the greatest problem in connection with its regular operation was to withdraw its power from it without serious losses of refrigerative effect. This might have been done (and it was originally proposed to do so) by connecting the turbine

to an electrical generator, but for various reasons a direct-connected rotary water pump was employed.

The turbine worked near the lower extremity of a vertical shaft, supported beneath on a ball bearing, in a casing from which the intensely cold exhaust (a mixture of gases rich in nitrogen and of liquid rich in oxygen) was discharged. The compressed air (pre-cooled by its course down in temperature through one set of passages in the thermal counter-current system) was admitted to the turbine guides through one of more expanding nozzles.

The shaft extended some 2 ft. vertically from the turbine case through a steel tube which enclosed it with but little clearance up to the pump casing. The pump impeller was an absolutely smooth disk (if disk it may be called in view of the shape shown in the figure) of but 4⅝ in. diameter, and the high speed of rotation gave a pressure equal to a head over 30 ft. of water.

The water was prevented from running down the vertical tube aforesaid, when the machine was idle and centrifugal force consequently not operating to drive the liquid toward the periphery of the casing, by an extension of the said tube inside the peculiarly shaped impeller. The whole arrangement is illustrated in the illustration. The water was pumped round and round, past a throttling valve, through a coil immersed in a calorimeter tank, and, as all the energy of the pump was thus transformed into heat, it was an easy matter to measure the rate at which heat was being withdrawn from the working fluid.

We thus had hot water circulating in the upper portion of the machine and cold fluids in the lower portion. With a view to preventing water vapor from finding its way down the vertical tube and freezing things up solid, an extremely small vent was provided in the upper part of said tube through which a gradual stream of cold gas from the turbine case continually passed, being driven by the slight residual pressure existing in said case and which arose from the back pressure in the pipes and passages leading the exhaust upwards in temperature. Any sudden drop in the performance of the turbine was instantly evident on the pressure of a gage attached to the pump.

In dropping the temperature of the apparatus preparatory to an oxygen run things would go smoothly down to a temperature well below that at which the apparatus could be run for liquid air in the method hereafter described, and then, all of a sudden, the turbine nozzle would start to plug with solid carbon dioxide. Save that we circumvented this by the operation described below, we could never have got the temperature any lower because we could not continue to withdraw energy (heat) from the working fluid.

The way I got over this obstacle was as follows: As soon as the nozzles plugged the supply of intensely cold compressed air was shut off and a current of relatively warm compressed air (about 75° below zero C.) was shunted in for a few seconds from a point part way down in temperature on the counter-current system. This almost instantly caused the solid carbon dioxide to vanish and we then shut down the compressor and turbine for a few minutes while we opened a valve leading from the exterior to the exhaust pipe of the turbine and poured through said valve some 30 gal. of a liquid mixture of oxygen and nitrogen.

On closing the valve and starting up again the chilling effect of this "priming charge" sufficed to carry the temperature of the turbine feed below the point at which trouble could occur in the nozzles from CO_2. The fractionating of the air into fluids respectively rich and poor in oxygen then commenced and proceeded. The liquid portion (rich in oxygen) was delivered (with or without further concentration) to the low temperature end of one of the three sets of passages of the counter-current system and the gaseous portion, or portions, delivered to a second set of passages. The remaining (intermediate) set of passages was, of course, utilized by the down-flowing current of compressed air.

The final result was a discharge from the high-temperature ends of the counter-current passages of gaseous mixtures respectively rich and poor in oxygen, the temperatures as they finally left the system being about 10° C. lower than when (as a single current of compressed air) they entered it. The oxygen runs, as we called the above, required about eight times as long to get the apparatus properly chilled off as the liquid runs, and we succeeded in making these latter many months before the apparatus had been perfected sufficiently to make the former.

For making liquid we simply ran the exhaust of the turbine through a lagged galvanized-iron drum about 16 in. diameter by 4 ft. long, containing a coil of *lead* pipe to which air, compressed by a small auxiliary compressor to about 400 lb. or 500 lb., was admitted. This air readily liquefied. In these liquid runs the efficiency of the main heat interchange was remarkably high, only about 2° difference being observed between the temperatures of the inflowing and outflowing fluids.

The reason we had the trouble with solid CO_2 at the excessively low temperature we did (just at the point at which the air was preparing to deposit a liquid condensation rich in oxygen) was because, in spite of the fact that carbon dioxide freezes at about—56°, its partial pressure, as it exists in air, is so excessively slight that it does not make its appearance until around 100° below its freezing point.

While there is very much to be said in favor of the turbine type of motor for this purpose, especially should the instrument be built by experts in de Laval construction, I nevertheless have to consider much of the above of historical interest only because I was successful later in operating a specially built "steam engine" on the mixture of liquid and gaseous constituents of air (exhausting at about 300° below zero Fahr., and incidentally leaking a little liquid oxygen at the metallic packing at the piston gland).

This apparatus, when in operation, was a sight worth seeing because the engine was surrounded by a mist and by wreaths apparently of vapor, exactly like a steam engine with a little steam blowing off, but the reason was the condensation, out of the air in contact with such parts of the engine as were unavoidably exposed without lagging, of the contained moisture as frost, with the steamy aspect familiar to all those who have worked with these intensely low temperatures.

Using the engine the trouble with CO_2 completely vanished. The power was taken off a belt-connected Westinghouse direct-current generator, and the instrument readings on same gave a continuous indication of the progress of the energy (heat) abstraction.

The results obtained fully justify the statement that, on a large scale and with cheap power, the total gross costs of the oxygen contents in a 40 per cent to 50 per cent rich atmosphere would be about 14 cents per 1000 cu. ft. While, in order to secure this figure, a production not far from 1,000,000 cu. ft. is required per 24 hours, there are many metallurgical and chemical applications requiring oxygen to which taking care of this amount would be a small matter. The plant is simple in construction and liable to but slight repair cost, which latter are included, together with interest, in the above estimate of product cost.

An experiment of some interest was tried when working on the turbine which consisted in the employment of a frozen mercury bearing for the turbine shaft. Obviously no lubrication was possible at the low-temperature end and the frozen mercury was an ideal anti-friction metal so far as the coefficient of friction was concerned. But it was too utterly soft in wearing qualities, even at very low temperatures, to give satisfaction.

I may mention that I relied almost entirely on cotton batting as a heat insulator. For special purposes, requiring a covering susceptible of being used as a plaster, a mixture of roofing tar and hair gave what were considered surprisingly good insulating properties, considering its lack of cellular character. I

may also put up a good word for lead, both in sheet and pipe, for low-temperature work. In important parts of the plant its use made practicable a form of construction which I regard as almost indispensable.

A good form of storage for liquid mixtures of oxygen and nitrogen consisted in what I called the "wasp's nest" container. It was composed of a cylindrical galvanized-iron holder, with cone-shaped top and bottom, well lagged with about 4 in. of cotton batting contained in an outer casing of tin or light galvanized iron, and then another 4 in. of batting and a final outer container. The lagging covered practically only the sides and bottom of the holder and the whole arrangement was supported by being hung by ropes from the ceiling, there being thus no props from beneath to conduct the heat into the liquid charge.

It is of interest to note that the value of the oxygen in a 50 per cent mixture as compared with pure oxygen for applications requiring a strength not greater than 50 per cent, is 0.74. That is to say, 74 parts of pure oxygen added to 126 parts of air (costing nothing) give 100 parts total oxygen in a 50 per cent mixture.

OTTAWA, ONT., CANADA.

Defects in Refractory Brick and Their Causes.

BY GILBERT RIGG.

Some time ago a friend of mine was trying to buy firebrick for a special purpose. His efforts to obtain suitable material were, to put it mildly, only moderately successful. To quote his own words, "What with the furnaceman's prejudices, the firebrick salesman's optimism, my own ideas, and the actual results, I am having a delightful time."

I am inclined to believe, judging from my own, that my friend's experience is by no means unique. The four indeterminates quoted above need to have fixed and definite values substituted for them. By bringing the first term to zero, substituting for the second set of specifications directly correlated to the third, and for the third a knowledge of the chemical and physical conditions under which bricks required for a given purpose must work, we obtain three new values by which the fourth, or the actual results, may be determined in a manner to give satisfaction to the consumer.

M. Baraduc-Muller, assistant to M. LeChatelier, in an excellent paper read before the Congress of Applied Chemistry, in London, brings a strong indictment against refractory products in general. After making a careful analysis of the factors which go to make first-class goods, which factors he afterwards combines into what he calls a "ceramo-metric coefficient" of value, he sums up the defects of what one frequently finds on the markets as follows:

"The principal defects, which the best of them show when in use, are:

"(1) Insufficient purification of material.

"(2) Methods of manufacture inadequate in mechanical power, superannuated in fact, and inappropriate for the new conditions which have to be met.

"(3) Burning at a temperature much lower than that at which the products will be employed.

"From these errors there results:

"(1) Too low a fusion point.

"(2) A lack of constancy in the dimensions of the pieces during service.

"(3) Want of compactness.

"(4) A porosity too great to resist the destructive action of gases and liquids.

"(5) A mechanical resistance too low to resist the wear of friction and shock.

"(6) Lastly: Much too great a sensitiveness to temperature changes for the prevention of cracks and fissures, which facilitate chemical and mechanical destruction."

This is a somewhat severe indictment, and in many cases, I believe, a well-deserved one. But, to be fair, some trouble arises also from lack of knowledge, on the part of the consumer, of the conditions under which the bricks are to serve. Given that these conditions are known and standarized, then the blame is entirely the manufacturer's, if he fails to meet them.

The term "refractory" is a relative one and covers a wide range of properties. Because a given firebrick does not melt down at the temperature at which it is employed, it is not necessarily refractory. Not infrequently, of two bricks with different fusion points, the one with the lower fusion point may be the more refractory under the conditions of employment.

In a case that has come under my notice, two types of firebrick (A and B) have been employed for the same purpose. A is less resistant to temperature than B, but resistant enough to withstand any temperature that can arise under working conditions. Still, if a cone test were all that was concerned, B would be classed as the better brick.

And yet A lasts five times as long as B.

The reason is not far to seek. On breaking a specimen of A we have a dense stony mass, the fracture passing through the grains of grog, which are welded into a solid mass with a bond. A broken surface of B shows the body of the brick to be a more or less crumbling mass, the grains of grog sticking out like loose teeth, and ready to fall out at a touch. Although the outside looks good, the inside is penetrated by a network of interlacing cracks.

Under working conditions A slowly wears away from the surface inwards. B falls to pieces. And B is typical of a large percentage of the so-called refractory bricks on the market. Lack of care in sizing the grog (the bricks in question contained "chunks" ½ in. in diameter), and insufficient temperature in burning, are the cause of the trouble in this case. The clay is right. The bricks are hopelessly wrong.

The properties possessed by a product which may properly be spoken of as refractory may include any or all of the following according to the conditions of service:

(1) Absolute infusibility at the highest working temperature.

(2) Complete absence of deformation and shrinkage under working conditions.

(3) Mechanical strength.

(4) Complete resistance to penetration of vapors, slags, etc.

(5) A chemical composition fitted to withstand as perfectly as possible the corrosive action of the substances to which the bricks are exposed.

(6) Equality and fixity of form and dimensions.

Let us consider these points in some detail.

A truly refractory product should be infusible at the temperature of service, not only as a whole, but as regards every part of it. A brick cannot be considered as really infusible which retains its general rectangular outline, and yet shows deep pits and slaggy patches due to particles of pyrite and other fluxes. A truly refractory product must be uniform in composition, and that composition must correspond to a melting point substantially higher than the temperature of service. Both the bond clay and the grog must conform to this condition. Very frequently they do not.

Deformation and shrinkage may be due to a wide range of factors which may be classified under two heads, composition, and temperature and duration of burning, the first set controlling the second.

The temperature at which the burning of the goods is carried out must be such as to ensure that the shrinkage has reached a maximum for the temperature of employment; the same thing applies to the duration of the burning. The readjustment in the relative position of the molecules, or particles of clay, which takes place at high temperature, requires time for its completion, different clays requiring different times. With a mixture of two clays, one high in silica, the other high in alumina, the mass burned at 1000° C. for 24 hours, had a breaking strain of

240 lb. per square inch. After seven days at the same temperature, the breaking strain was 534 lb. per square inch. The breaking strain is substantially a measure of the density and compactness of the mass.

Further, the material must be such that deformation (by softening or irregular shrinkage) does not occur. When a body of new masonry is heated, it expands. If the original burning of the bricks has not been carried out long enough, or at a temperature sufficiently high, shrinkage will supervene after the work has reached the maximum temperature, with formation of cracks and other troubles. The consumer should, therefore, know what is the maximum temperature the bricks must stand, and the manufacturers should be in a position to know at what temperature to burn his goods to ensure them against shrinkage, and what materials to use to ensure them against deformation at the temperature in question.

Under the head of deformation I include not only change of form as a whole, but also change of internal form or separation of constituent grains, giving rise to disintegration and crumbling.

A brick of high infusibility, burned at a temperature too low to ensure dense burning, and employed at a temperature too low to develop density, may not shrink seriously as a whole, but, being mechanically weak, will frequently crumble and disintegrate. Such a brick will be inferior under working temperature conditions to one more fusible and burnt dense, if the fusibility is not so great as to cause softening when in service.

The question of density is of tremendous importance in relation to the question of penetrating vapors and slags.

Whenever firebricks are to be exposed to these exceedingly destructive influences they should be burned at a temperature at which porosity is as nearly as possible zero.

The extent to which some slags will penetrate into the pores of a brick is remarkable and the effect is often disastrous. Under the microscope the pores of such bricks are seen to be filled with slag, even when the appearance to the naked eye will show nothing unusual. Comparison of a "penetrated" brick with a fresh one, at high temperatures, may indicate several hundred degrees difference in the fusion point. To quote a case from my own experience, the melting point of a fireclay mass exposed to a penetrating slag fell from over 1700° C. to 1400° C., although, except for a coat of glaze, it appeared unchanged. Under the microscope the filling of the pores by slag was plainly visible.

It is entirely illogical to go to the expense of the highest class of refractory clay to manufacture a porous product whose fusibility is bound to increase enormously under working conditions.

In dealing with this question the maximum temperature of service should be known, and the manufacturer must arrange his mixture to withstand this and burn the brick at a temperature that will reduce its porosity as nearly as possible to zero. Under these conditions the attack of the slag will be from the surface only, and the life of the brick will be greatly prolonged.

This brings us to the fifth item, or the relation of the chemical composition of the brick to the slag, etc., it must withstand. The necessity for this is obvious and is more or less recognized in connection with furnace linings.

Fixity of form and size is essential in reducing the thickness of the joints to a minimum and facilitating the work of the masons. The joints are always the weakest part of the work, as in the nature of things the mortar must be porous and more or less open in texture. Bad mason work is not always due to the mason. Some of it is due to warped and irregular brick. Moreover, the relining and repairing of a furnace is frequently carried out under hot and trying conditions when the men are not in shape to devote close care and attention to making up for the faults of the brick manufacturer.

One sometimes hears it said that "A's" firm used to turn out a first class brick for our purpose, but the quality has gone off altogether latterly." Is it quite certain that the quality of the brick has gone off in all cases? Have not changed conditions of service something to do with it? Increase of output on a given metallurgical unit unusally means increase in the severity of the working conditions. Similarly the manufacturer who loses a customer and who feels, nevertheless, that the quality of his product is as good as ever it was, has perhaps failed to recognize that the quality of his product has not kept pace with the demands of service.

Papers like the one quoted above tend to show that all is not well with much of the refractory ware now on the market. It would be interesting to know from those responsible for successful smelting operations how many would be found to curse and how many to bless? My own experience is that there is often cause for the former.

New Jersey Zinc Company.

Furnace Electrode Losses.

At the Charlotte meeting of the American Institute of Electrical Engineers, held from March 30 to April 1, two papers on furnace electrode losses were presented, the authors being Mr. Carl Hering and Dr. A. E. Kennelly. Both papers are printed in full in the March issue of the *Proceedings* of the Institute.

The paper of Mr. Carl Hering is entitled "The Proportioning of Electrodes for Furnaces." Mr. Hering explains that his investigation applies to "the electrode proper, or what might be termed the essential electrode, or the chief part of an electrode, namely the part which passes through the walls, and in which the co-operation of the two heat flows is the governing feature. Any additional parts within the furnace or projecting beyond the outside of the walls are not theoretically essential parts, even though they may be very necessary in practice: such parts are evidently determined by entirely different conditions and considerations and must be treated apart from that portion which is absolutely necessary to lead the current from the outside to the inside of the furnace. To attempt to combine the various parts of such a longer electrode and to treat them as one in an analysis for finding out the laws governing the proportions would either be impossible or would, at least, lead to endless complications rather than to a simplification." Hence, throughout the paper the proportions of the electrode, and particularly the length, refer to this essential part passing through the wall. The heat insulation of the essential part of the electrode from the surrounding walls is assumed to be perfect.

Mr. Hering sums up the results of his analysis and introduces two quantities as "true and correct measures of excellence" of materials for use as electrodes. These are the "electrode voltage" and the "specific section." They are constants for the particular materials, in so far as they are independent of the dimensions. The former concerns the power loss and the latter the size. The lower their values are the better is the material for electrodes.

Electrode Voltage.—For every material there is a fixed and definite voltage for 1° which voltage is the same no matter what the size of the electrode or the normal current, provided only that the electrode is correctly proportioned so that no heat flows through the hot end in either direction, the hot electrode end acting as a perfect heat value. This electrode voltage e in volts is

$$e = \sqrt{2\,k\,r}$$

if k is the thermal conductivity in watt, inch cube units, and r the electric resistivity in ohm, inch cube units, and k as well as r are assumed to have the same values over the whole length of the electrode. The total voltage E between the hot and cold electrode ends is then

$$E = \sqrt{2\,k\,r}\,T = e\sqrt{T}$$

where T is the temperature drop in centigrade degrees between the hot and cold ends. E multiplied by the current in amperes gives the minimum loss in watts which can possibly be obtained

with that material, for that current and temperature. On this account Mr. Hering calls E the "watts per ampere."

Specific Section.—Under the same assumption that r and k have the same values over the whole length, the relation

$$S = IL\sqrt{r \div 2kT}$$

exists between the electrode cross-section S in square inches, the electrode length L in inches ("essential length"), the current I in amperes and the temperature drop T in degrees C.

If I, L and T are made unity, that is, for 1 amp, 1 in. length and 1° C. of temperature, the resulting cross-section, now represented by s, becomes

$$s = \sqrt{r \div 2k}$$

This is called the "specific section." This is for 1° C., while for the temperature drop T, there is

$$S^1 = s \div \sqrt{T} = \sqrt{r \div 2kT},$$

which, since $S^1 = S \div IL$, is called the "section per ampere per inch length." By multiplying S^1 with the current and the length, the actual cross-section is obtained, for which the electrode is correctly proportioned so that the hot electrode end acts as a perfect heat valve.

Experimental Results.—The experimental determinations were made by the method described in Mr. Hering's Am. Electrochem. Soc'y paper (our Vol. VII, page 514). The main results are given in the following table and diagrams. The full lines in the curve sheets represent the ranges over which the results were actually measured; the dotted lines are exterpolated and "therefore, are only probable values." "There may, of course, be radical changes in these physical constants at the melting points of the metals, and at some points at which carbon may change its condition. These cannot be predicted. The present extensions are all based on a regular continuation of the measured relations."

The curves for the electrode voltage and the specific section are given in Figs. 1 and 2. In Fig. 1 they have been drawn to different scales so that they are nearly superimposed, thus enabling their relative inclinations or percentage values to be compared better with each other, while in Fig. 2 they have all been drawn to the same scale, so that they may be compared quantatively.

In Fig. 1 the curves E, which measure the minimum loss, or the watts per ampere, all have a family resemblance, except in the case of graphite. The marked difference of the graphite curve means that the losses in graphite electrodes do not grow as rapidly with the temperatures as they do for all the other materials. This, of course, is a very good quality, especially for high-temperature furnaces. For all the others the relative increase of this loss is about the same.

A comparison of the same curves E, in Fig. 2 (the heavy line curves) shows the quantitative relations of the actual losses per ampere. The much greater loss for carbon is quite striking; but, of course, there are many cases in which metals are excluded by the very nature of the furnace.

The loss for carbon is seen to be 1.55 times that for graphite.

Temperature Cent. degrees	Temperature Fahr. degrees	E Watts per ampere				S^1 Square inches per-ampere per inch length				S^1 Square centimeters per ampere per cm. length			
		Carbon	Graphite	Iron	Copper	Carbon	Graphite	Iron	Copper	Carbon	Graphite	Iron	Copper
400	752	1.00	0.85	0.145	0.095	0.00165	0.00040	0.000103	0.000016	0.0042	0.00100	0.00026	0.000041
600	1112	1.38	1.04	0.225	0.140	0.00114	0.00031	0.000102	0.000015	0.0029	0.00078	0.00026	0.000037
800	1472	1.68	1.19	0.295	0.184	0.00090	0.00027	0.000101	0.000014	0.0023	0.00068	0.00026	0.000036
1000	1832	1.93	1.32	0.350	0.226	0.00076	0.00024	0.000098	0.000014	0.0020	0.00061	0.00025	0.000036
1200	2192	2.15	1.43	0.410	0.270	0.00069	0.00022	0.000094	0.000014	0.0018	0.00056	0.00024	0.000035
1400	2552	2.36	1.52	0.460	0.310	0.00062	0.00020	0.000092	0.000014	0.0016	0.00051	0.00023	0.000035
1600	2912	2.53	1.62	0.510	0.351	0.00057	0.00019	0.000089	0.000014	0.0014	0.00048	0.00023	0.000035
1800	3272	2.68	1.71	0.565	0.391	0.00053	0.00018	0.000087	0.000014	0.0013	0.00046	0.00022	0.000035
2000	3632	2.83	1.79	0.610	0.430	0.00049	0.00017	0.000085	0.000014	0.0012	0.00044	0.00021	0.000035

If the current and temperature drop are given, and if the essential length is taken as the thickness of the furnace walls, the section in square inches is then determined at once by multiplying the proper temperature value of S^1 from the above table by the current in amperes and by the length in inches. If this is too large or too small to be practicable, the length may be changed and the section may be redetermined. The quotient of the section divided by the length is a constant for any specified conditions (numerically equal to S^1 from the table multiplied by the current), hence they may both be increased or decreased in the same proportion.

The loss in the electrode in watts is entirely independent of the section or length adopted, provided only that their quotient is approximately as above. It is calculated in watts by multiplying the corresponding temperature value of E in the above table by the current.

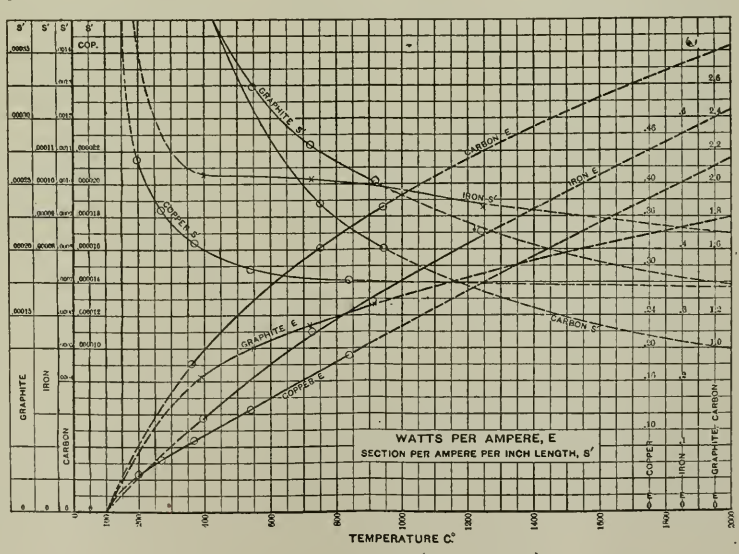


FIG. 1.—LOSSES AND SIZES. (SUPERIMPOSED.)

* * *

The paper by Dr. A. E. Kennelly deals with the "modifications in Hering's laws of furnace electrodes introduced by including variations in electric and thermal resistivity."

The assumption made by Dr. Kennelly concerning these variations is that both resistivities are linear functions of the distance from the hot electrode end. Under this assumption the integration of the fundamental differential equation leads to comparatively simple formulas, which lend themselves readily to numerical applications.

Two numerical examples are given.

The general results arrived at are summed up by Dr. Kennelly as follows:

1. The combined loss through the cold end of an electrode is equivalent to the sum of the loss by heat conduction alone (when there is no current) and approximately half the I^2R loss, the exact fraction depending on the temperature coefficients of the resistivities. With constant resistivities, the fraction will be one-half.

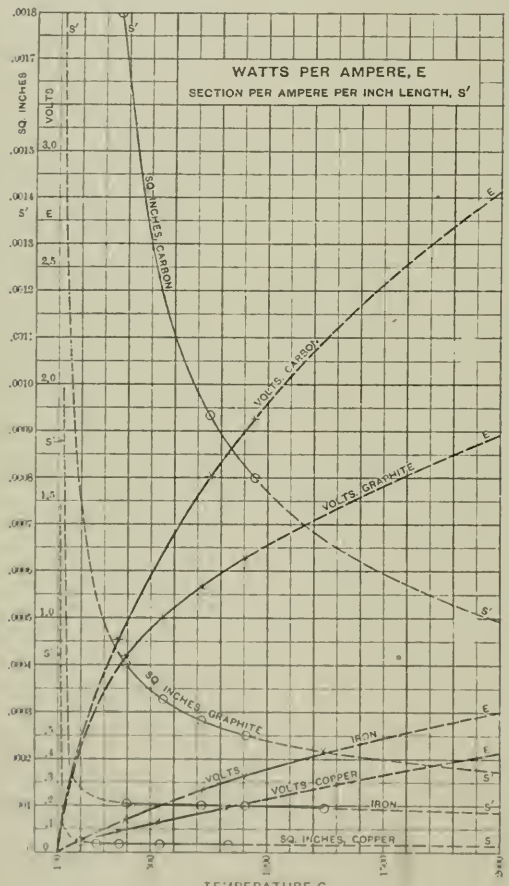


FIG. 2—LOSSES AND SIZES (SAME SCALE).

2. The above combined loss will be least when the loss by heat conduction alone is equal to the Joulean loss through the cold end.

The total loss will then be approximately equal to the total Joulean loss, and very little heat will flow into or out of the furnace.

With constant resistivities, this resultant furnace flow will be nil.

Chemical Reactions at Electrodes.*

By Reston Stevenson, Ph.D.

The following article is merely a review of the broad field of electrochemical reactions in liquid solutions and more particularly in aqueous solutions. It is in no way a presentation of new experimental research or of new theories. The writer wishes to acknowledge his indebtedness to Dr. Bancroft, of Cornell, and to Dr. Turrentine, of Wesleyan University, for suggestions and references, and to Dr. Baskerville, of the College of the City of New York, for assistance in the preparation of this paper.

In electrochemical reactions, the ions are carried to the respective electrodes, where they lose their charge and react with whatever is at hand. Obviously the nature of the ions originally in the electrolyte is of prime importance. This consideration is complicated by the fact that ionization is not always of the simple sort, such as $NaCl = Na + Cl$, but sometimes double ionic natures are displayed.

For example, Br in liquid sulphur dioxide gives upon electrolysis Br at both the anode and the cathode. Similarly, I in sulphuryl chloride gives I at both electrodes. In the electrolysis of an alkali solution with tellurium electrodes, Te'' is formed at the anode and $Te^{\cdot\cdot}$ at the cathode. $Pb(OH)_2$ may ionize in any of these ways:

$$= PbOH^{\cdot} + OH'$$
$$= Pb^{\cdot\cdot} + 2\,OH'$$
$$= H^{\cdot} + PbOOH'$$
$$= 2\,H^{\cdot} + PbO_2''$$

In an acid solution the ionic product of H_2O is exceeded and, therefore, the OH ions practically disappear, so that we have only Pb and OH ions.

In an alkaline solution, the ionic product of H_2O is again exceeded and causes the practical disappearance of the H ions so that we have only PbO_2 and OH ions in the solution.

Glycocoll dissociates thus:

$$HOH_2NCH_2COOH = H_3NCH_2C{\cdot}O'O + H^{\cdot} + OH'.$$

Of course the two dissimilar charges upon the residue neutralize each other, leaving H and OH ions in the solution.

When the electrolyte is not a single substance in solution, then the relative amounts of the different ions which are carried to the electrodes to react, depend upon the migration velocities of these different ions. Electrolytes are ionized quite differently according to the solvents[1] which are used, sometimes showing association instead of dissociation, etc.

Not only may there be a variety of ions in an electrolyte, but these ions may react in various ways at the electrodes and many of these reactions may occur simultaneously. For example, in the electrolysis of NaCl solutions, we may obtain NaOH, Cl, NaCl, NaClO, $NaClO_3$, $NaClO_4$ and O_2.

As a review of the field of electrochemical reactions, we will classify all of the various reactions which have occurred at electrodes, and then consider the variable factors concerned in the equilibrium of these reactions. The following reactions occur at electrodes:

1. The discharged ion reacts with itself
2. The discharged ion dissolves in the electrode.
3. The discharged ion reacts with the electrode
4. The discharged ion dissolves in the solution.
5. The discharged ion reacts with the solvent.
6. The discharged ion reacts with the solute
7. The discharged ion reacts with suspended particles.
8. The discharged ion reacts with dissimilar ions of similar charge.
9. The discharged ions of dissimilar charge react with each other.
10. The reaction products which are at first formed, break down and form secondary products.

* An address delivered before the New York Section of the American Electrochemical Society on March 16, 1910.

[1] Le Blanc, Whitney and Brown, Electrochemistry, p. 142.

11. The products at the electrodes diffuse and react away from the electrodes.

12. Electrolysis with alternating current.

13. Electric endosmose.

14. Electrostenolysis.

15. Reversed electrolysis.

1. The Discharged Ion Reacts with Itself.

Examples at the cathode are:

$$H^{\cdot} + H^{\cdot} = H_2$$
$$Cu^{\cdot\cdot} + nCu^{\cdot\cdot} = Cu_{n+1}$$

At the anode:

$$O'' + O'' = O_2$$

2. The Discharged Ion Dissolves in the Electrode.

For instance Na forms sodium amalgam with a cathode of Hg; H dissolves in a cathode of Pd; O dissolves in an anode of platinized platinum.

3. The Discharged Ion Reacts with the Electrode.

Cathode: Na forms an alloy with a cathode of Pb. Zn plated on Pt may alloy to a slight extent.

$$2H^{\cdot} + PbO_2' + H_2SO_4 = PbSO_4 + 2H_2O$$

Anode:

$$O'' + PbSO_4 + H_2O = PbO_2 + H_2SO_4$$
$$3Cl' + Fe = FeCl_3$$
$$4O'' + CuS = CuSO_4$$

In the electro-analysis of halides, phosphates, etc., a silver anode is used and the corresponding compounds are formed and adhere to the anode and can be weighed.[2]

4. The Discharged Ion Dissolves in the Solution.

At the cathode the solution is saturated with hydrogen and, at the anode, with chlorine, bromine, etc. The substances may be dissolved either in the solute or in the solvent.

5. The Discharged Ion Reacts with the Solvent.

Cathode:

$$Na^{\cdot} + H_2O = NaOH + H$$
$$Ca^{\cdot\cdot} + 2H_2O = Ca(OH)_2 + H_2$$

Anode:

$$SO_4'' + H_2O = H_2SO_4 + O$$
$$2CH_3COO'' + H_2O = 2CH_3COOH + O$$

6. The Discharged Ion Reacts with the Solute.

Cathode:

$$H^{\cdot} + FeCl_3 = FeCl_2 + HCl$$
$$Al^{\cdot\cdot\cdot} + 3HCl = AlCl_3 + 3H$$

Anode:

$$Cl_2(SO_4)_3 + 3H_2O + 3O'' = 3H_2SO_4 + 2H_3ClO_4$$
$$C_2H_5OH + 10I' + H_2O = CHI_3 + CO_2 + 7HI$$

7. The Discharged Ion Reacts with Suspended Particles.

Cathode:

$$C_6H_5NO_2 + 6H^{\cdot} = C_6H_5NH_2 + 2H_2O$$

Anode:

$$C_6H_6 + 2O'' = C_6H_4(OH)_2$$

8. The Discharged Ion Reacts with Dissimilar Ions of Similar Charge.

At the cathode Hg and Ag can be deposited as an amalgam from acid solutions. Zn and Cu are deposited as brass from aqueous KCN solutions. In platinizing electrodes, an alloy of Pb and Pt is deposited and the Pb is dissolved away, leaving the Pt in finely divided form. In the plating of Ni, an alloying with NH_4 is often the cause of the peeling of the plate.

At the anode, Cl and O combine to form ClO′ or ClO′$_3$.

9. The Discharged Ions of Dissimilar Charge React with Each Other.

This necessitates that the anode and cathode be very near each other. This would be of practical disadvantage in any electrochemical process.

10. The Reaction Products Which Are at First Formed Break Down and Form Secondary Products.

In the electrolysis of NaCl aq. with a Pb cathode, the alloy is formed and then the Na reacts with the H_2O and leaves the Pb as a fine suspension in the solution.

[2] Smith: Electro-analysis, p. 296.

With a Cd anode in a $Na_2S_2O_3$ solution, there is formed CdS_2O_3, which reacts thus:

$$CdS_2O_3 + H_2O = CdS + H_2SO_4$$

11. The Products at the Electrodes Diffuse and React Away from the Electrodes.

In the preparation of NaClO, the NaOH at the cathode meets the Cl from the anode and forms the hypochlorite. In the electrolytic preparation of $PbCrO_4$, the electrolyte consists of $NaNO_3$ and K_2CrO_4. The NO_3 attacks the Pb electrode and the $Pb(NO_3)_2$ reacts with the K_2CrO_4 to form $PbCrO_4$ away from the electrodes. In the production of Scheele's green, the electrolysis of Na_2SO_4 yields Na and SO_4 ions. With the Cu anode, $CuSo_4$ is formed. At the cathode, NaOH is formed, which reacts with the As_2O_3 around the cathode, to form Na_3AsO_3. The anode and cathode products diffuse and react to form Cu_2HAsO_3.

12. Electrolysis with Alternating Current.

In the application of the alternating current to a reversible cell such as Cu — $CuSO_4$ — Cu, then the same amount of Cu is dissolved at either electrode with the current in one direction as is deposited by the reversed pulsation. The Cu goes into solution and precipitates with practically equal ease. If, however, the Cu electrodes are in a KCN solution then the Cu which dissolves forms a compound with KCN which has only very few Cu ions. From this solution, the Cu is precipitated with much greater difficulty than it dissolves, so that with the alternating current, the Cu dissolves by a zig-zag movement in which the zig is always longer than the zag. In a 4 normal solution of KCN, 75 per cent Cu is dissolved when there are 10,000 alternations per minute, 55 per cent with 20,000 alternations, and 40 per cent with 300,000 alternations. Zn and Ni also dissolve in KCN solutions with alternating current. Ag is practically insoluble in KCN solutions when the alternations exceed 4000 per minute.

13. Electrical Endosmose.

Consider a liquid in contact with a solid, such as the glass walls of a capillary tube, or solid colloidal particles of Au, Pt, $Cr(OH)_3$ dyes, such as indigo, methyl violet, etc. Upon these bodies, in contact and moving past one another, are dissimilar charges of electricity. Upon that body which has the greater dielectric constant (usually water), there is a positive charge, and there is a negative charge upon the other body. Therefore, upon passing a current there is a separation at the different poles. When the solid is fixed stationary, e. g., glass tube, wood, etc., then the water alone moves to the cathode.

14. Electro-stenolysis.

Consider a salt solution in a beaker and in a test tube which stands in the beaker. The test tube is cracked, so that the two solutions are separated by capillaries. An electrode is in each vessel. Upon passing a current, water goes (c.f. 13) to the cathode and the capillaries are left negatively charged. Therefore positive ions of metal are deposited in these capillaries, but the amount of metal is microscopic because of the large charge on an ion and the small charge on the glass. However, the amount of metal becomes noticeable when it takes part in the conduction of the current and does not dissolve at its anodic end. Such a condition is obtained with the platinum metals and with metals which form insoluble oxides at the anode, such as Co.

15. Reversed Electrolysis.[4]

At the cathode occurs reduction and at the anode oxidation, but the apparent action at these respective electrodes may be reversed. This occurs when the reduction product at the cathode is an active oxidizing agent. When a dilute solution of HNO_3 containing KI is electrolyzed with Pt electrodes, the HNO_3 is reduced to HNO_2 at the cathode. The HNO_2 oxidizes the KI setting iodine free at the cathode. In the electrolysis of a solution of $KMnO_4 + H_2SO_4$, with Pt electrodes, the

[3] Le Blanc, Whitney and Brown, Electrochemistry, p. 157.
[4] Turrentine, Trans. Am. El. Soc., 1908, Vol. 13, p. 281.

$KMnO_4$ is reduced to MnO_2 at the cathode with a simultaneous evolution of oxygen at the cathode. Probably the $KMnO_4$ is reduced to Mn_2O_3, which spontaneously reduces to MnO_2 with the evolution of oxygen.

Upon electrolysis of Fehling's solution between Pt electrodes the tartaric acid at the anode is oxidized to formic acid which reduces the cupric salt with the deposition of cuprous oxide on the anode. Upon the electrolysis of a solution of $AuCl_3$ in methyl alcohol with Pt electrodes, at the anode the alcohol is oxidized to formaldehyde which reduces the $AuCl_3$ and causes the deposition of Au upon the anode. Upon electrolysis of a solution of Na_2SO_4 with a Mg anode, the Mg is oxidized to $MgSO_4$ which reacts with the water to set free hydrogen at the anode.

Of these reactions, the nature, direction, extent and velocity may be affected by the following variable factors of equilibrium: temperature, pressure, concentration and nature of electrolyte, current density, light, diffusion, e.m.f., nature and concentration of electrodes, and catalytic action. In any one reaction many or all of these factors might change so that the effect would be the resultant of many different forces. In the following pages will be considered the effect of the variation of some one of these factors while the others remain constant. These three generalizations apply at the condition of equilibrium: the phase rule, the law of mass action, and the theorem of Le Chatelier.

1. **Temperature.**

In all electrochemical preparations, e. g., iodoform, aniline, etc., there is always a narrow range of temperatures within which the greatest efficiency is obtained. Above 300° Na is deposited from the fused salts, but if the temperature is too high, the Na reacts with the bath and the yield is greatly lowered. Similarly, for Ca, Ba, and Al. In the electrolytic formation of persulphates, percarbonates, perchlorates, etc., the temperature must be kept low because these compounds decompose upon heating. In the electrolysis of an aqueous solution of HCl, when the anode is kept cold, chlorine hydrate is formed in crystals. In the electrolytic preparation of NaClO, increased temperature causes the formation of chlorate.

In general, the increase of temperature causes the increase of the diffusion of the electrode products, the velocity of migration of ions, the dissociation of the electrolyte, and the speed of reaction. That reaction which occurs with an absorption of heat would be promoted by an increase of temperature.

2. **Pressure.**

The effect of pressure is rarely studied in electrochemical reactions. Usually, a large change of pressure accomplishes only a slight variation in the point of equilibrium. In general, it may be said that an increase of pressure would promote that reaction which occurs with a diminution of volume (theorem of Le Chatelier).

3. **Concentration and Nature of Electrolyte.**

Since the reactions at electrodes are usually reactions with the electrolyte, the nature and concentration of the electrolyte is all-important. In reversed electrolysis, it has been seen that the nature of the electrolyte can reverse the apparent action of the electrodes. In the preparation of NaClO, if the solution becomes acid, the free HClO oxidizes the hypochlorite to chlorate. If the NaClO becomes concentrated, it diffuses to the cathode and is reduced. In the electrolysis of $CuSO_4Aq$, in neutral solutions at 100°, Cu_2SO_4 is formed which hydrolyzes and Cu_2O is formed. In concentrated HNO_3 solutions and in very concentrated H_2SO_4 solutions, no Cu is precipated because of the greater rapidity of solution than of precipitation. Similarly, Li is not deposited from water solutions, but is from a pyridine solution which conducts the current and does not dissolve the Li. In the electrolysis of HNO_3, with a Pt anode and a Hg cathode, hydroxylamine is obtained, but if Hg ions are in the solution, NO is obtained (a paste of Hg_2SO_4 on the Hg).

In the electrolysis of a 0.3 normal NaCl solution, without diffusion, 5 per cent of oxygen is obtained. In alkaline solutions, ketones are reduced and in acid solutions are formed pinacones. Al can be deposited from ethyl bromide solutions. In the electrolysis of neutral solutions of $Na_2S_2O_3$ the tetrathionate is formed, but this can be prevented by the presence of $Hg(CN)_2$ or by brucine. Salts of Mn and of Cr are used as carriers in electrolytic oxidation. In electroplating, deposits are finer from dilute than from concentrated solutions, the presence of a colloid makes the crystals finer and the deposits are made pure by the introduction of substances to dissolve any oxide or hydroxide that might be formed: e.g., in plating Zn, H_2SO_4, KOH, NH_4Cl, and KCN are used.

4. **Current Density.**

In electroplating too high a current density makes the deposit go bad, due to an impoverishment of ions and the precipitation of oxide, hydroxide, or basic salt. Deposits are finely crystalline with high current density and coarsely crystalline with low current density. When a current of less than 1 millampere is passed between metal electrodes in distilled water, arborescent crystals are obtained.[5] With Fe as anode in sodium acetate acetic acid solutions, at low current density, ferrous acetate is formed and with high current density Fe is passive. Since current density is proportional to the rate of deposition, Al can be deposited as a metal from $AlCl_3Aq$ with such a current density (0.085 amp per square centimeter) that the rate of solution is less than the rate of precipitation.

A high current density is conducive to low electrode efficiency, because the H or Cl., e.g., which is deposited escapes before it comes into contact with any of the reacting electrolyte. Therefore, in the oxidation of $Cr_2(SO_4)_3$ to CrO_3, and in the preparation of iodoform, the anode is large and the cathode is small. In general, the effect of current density is that of mass action.

5. **Diffusion.**

In the deposition of metals a high current density causes an impoverishment of ions at the cathode and thereby causes the deposit to go bad. This impoverishment is prevented by means of stirring with CO_2, H_2, or with rotating cathode or anode. In the precipitation of Al, Tucker[6] uses a pasty solution of $AlCl_3$, 40°, C.D. = 0.085 amp per square centimeter, and a cathode of brass rotating at 15,000 r.p.m.

In many electrolytic processes the reaction at only one electrode is utilized and the reaction at the other is harmful. In the electrolysis of NaCl to form NaOH the caustic at the cathode is separated from the Cl at the anode by means of a diaphragm of Hg. Diaphragms should be very easily permeable to charged ions and should offer a high resistance to diffusion. In making iodoform, the CHI_3 at the anode is separated from the H at the cathode by means of porous porcelain in order to avoid reduction. In making NaClO, $CaCl_2$ or K_2CrO_4 is added to the electrolyte, giving a precipitate of CaO or $Cr(OH)_3$ at the cathode. This precipitate is a membrane which allows the passage of OH and Na ions, but prevents the diffusion of NaOCl to the cathode.

6. **Light.**[7]

Bancroft has shown that those changes which are produced in compounds by means of light can also be produced electrolytically. "Light is a low-voltage current." Also he has shown that the presence of light may affect the direction of a current of electricity and its incidental reactions. If a ray of light which acts as a reducing agent is allowed to fall upon one pole of an electrolytic cell and there brings about a reduction, we then have that pole functioning as the cathode of the cell, and an electric current flows through the cell from the unilluminated pole to the illuminated pole. AgBr is reduced by light which is rich in violet rays and Ag is set free. If

[5] Comptes Rendus, 130, 325, 1900.
[6] Trans. Am. El. Soc., p. 497, 15, 1908.
[7] Bancroft, *Jour. Phys. Chem.*, 1908, et seq.

AgBr is in close contact with the illuminated pole, Ag is deposited on that pole and it becomes the cathode.

If one AgBr electrode is dipped in erythrosine and then replaced in the KBr solution, the dyed electrode is the anode. These sensitizers are used with actinic cells (1) to intercept those rays of light which tend to work oppositely to that reaction which we wish to induce, and (2) the sensitizer is altered by light and yields a product which tends to accelerate the desired reaction.[8]

7. **E. M. F.**

Unless a certain difference of potential (decomposition voltage) exists between the anode and cathode, there will not be sufficient force to separate the ions. This decomposition voltage varies for different combinations of ions so that by careful regulation of the voltage we can select which ions to deposit. In electroplating, two ions are easily separated if the decomposition voltages of all the ions are exceeded and H has an intermediate decomposition voltage. Frequently it is noticeable when the same products are set free at the electrodes and at the same rate (equal current density), that there is obtained different reduction and oxidation products, varying with the voltage of the current. This voltage does not refer to decomposition voltage on anode or cathode, which varies with the distance apart of the electrodes without a change in the nature of the electrode products. The voltage which seems effective is the absolute voltage measured as the decomposition voltage of the electrode and the standard H electrode.

Alcohol is oxidized to acetaldehyde when the decomposition voltage of the anode and the H electrode is 0.7 volt to 1.0 volt; if a higher voltage is used it is oxidized to acetic acid. The decomposition of HF is higher than of HCl, and, therefore, in the oxidation of $Cr_2(SO_4)_3$, $MnSO_4$ and $CoSO_4$ solutions, by electrolysis, greater oxidation is obtained when HF is present. (Z. A. C., 33, 25, 1903.)

In reducing nitrobenzene with many different metals as cathode, various products and in various proportions are obtained, but the same products, azoxybenzene and aniline, are obtained when the cathode potential is kept constant. (Z. P. C., 47,418, 1904.) At platinum electrodes O_2 is formed at 1.08 volts and ozone is formed at 1.67 volts (voltage between anode and H electrode). At the cathode, reduction depends upon the voltage of the cathode compared to that of a H electrode in the corresponding electrolyte.

Counting the voltage of the cathode in a reversible H cell with platinized Pt electrodes as zero and then the "excess voltages" for the different cathodes are as follows:

Pd	−0.26 volt.
Pt	0.00 "
Fe	+0.03 "
Au	+0.05 "
Ag	+0.07 "
Ni	+0.14 "
Cu	+0.19 "
Al	+0.27 "
Pb	+0.36 "
Hg	+0.44 "

The greater force necessary to set free the H shows that these metals offer less accelerative influence in the change $H + H = H_2$. Therefore, these metals accelerate instead of retarding the actions of the H ion in the order of their excess voltages. The effects of these higher voltages are due in most cases to the specific nature of the electrodes and are probably catalytic effects. (See 8 and 9.)

8. **Nature and Concentration of Electrodes.**

Caffeine is reduced more quickly at a cathode of Hg than of Pb, although both metals possess the same cathode excess potential in pure acid. From nitrobenzene, with Cu cathode and NaOH, aniline is formed. With high current densities $KClO_3$ is only slightly reduced when the cathode is Pt, Pb, Zn, or Ni; moderately reduced with a Co cathode, and strongly reduced when the cathode is wrought iron. PbO_2 anodes oxidize $Cr_2(SO_4)_3$ with 80 per cent efficiency; Pb anodes with very slight efficiency. Aqueous KNO_3 is reduced 76 per cent with a Pt cathode, 15 per cent with a platinized Pt cathode and 97 per cent with Fe or Zn cathodes. A cathode of Fe reduces neutral $NaClO_3$ with 23 per cent efficiency, while Zn or Pt cathodes give practically no yield. (Z. E., 7,398.) $HNO_3 + H_2SO_4$ gives 100 per cent NH_2OH with a cathode of Hg, 40 per cent with a Pb cathode and 15 per cent with a Cu cathode. If the Cu cathode is covered with finely divided Cu then no NH_2OH is obtained, but a 100 per cent yield of NH_3 (Z. A. C., 31,289, 1902). If the Hg cathode is covered with Hg_2SO_4, insuring the presence of Hg ions, NO is formed. In the electrolysis of $HNO_3 + H_2SO_4$, if $CuSO_4$ is added, NH_3 is obtained quantitatively, but NH_3 is not obtained if instead of Cu we add Co, Ag, or Cd. When HNO_3 is electrolyzed with a Cu cathode, NH_3 is obtained, but NO is obtained when $Cu(NO_3)_2$ is present at the cathode. (Turrentine, 10,715, 1906.)

To remove brass from Fe tubing the whole is placed as anode in a bath of $NaNO_3Aq$, and the Fe remains passive and the brass dissolves away. (Burgess, *Trans.* Am. El. Soc., 1903, page 31.) To plate Ni upon a surface of Ni, the cathode Ni must first be made active by making it the cathode in another bath (acid solution) for a few minutes before use. (Snowden, J. P. C., 1905, page 399.) Na can be precipitated from aqueous solutions into Hg amalgam because of the slow reaction of Na amalgam with water. Fe and Ni in the amalgam make the reaction with water so rapid that no Na is precipitated.

When the anode is attackable, then the case is complicated by the presence of other compounds in the electrolyte.

9. **Catalytic Action.**

Above 90° cobaltic oxide acts catalytically in the electrolysis of chlorides, setting oxygen free. In 7 and 8 we have noticed that different metals as electrodes have different specific effects in electric reactions and that these effects are obtained without any apparent chemical action on the part of the electrodes. Such influences are probably catalytic. Bancroft[9] has shown that aromatic nitro compounds in acid solution are reduced to amines by Fe, Sn, Pb, and Cu as cathodes or when used chemically; and in alkaline solutions reduced to hydrazo compounds by Sn, Zn, and Pb chemically or electrolytically.

Ostwald[10] has said in regard to catalytic action in electrolysis that an ion first gives up its charge and then occur secondary reactions, and many different secondary reactions may occur.

Catalytic agents may retard some of these reactions and accelerate others.

Conclusion:

In conclusion we may say that reactions at electrodes differ from ordinary reactions in these ways:

(1) Energy is used instead of matter. The amount of substance used in electrochemical reductions and oxidations is small and its active principle is regenerated by electrical energy and used repeatedly.

(2) Localization of effect.

(3) Two dissimilar reactions occur simultaneously.

(4) The regulation of the reaction velocity is easy.

Bancroft has shown the identity of chemical reactions and electrode reactions, and has shown that the same variables affect the equilibrium in both cases. "The only specific action of the electric current is that it tends to set free anions at the anode and cations at the cathode. Reactions occur after the loss of the charge of electricity, and these reactions are ordinary chemical reactions."

Department of Chemistry,
College of the City of New York.

[8] Schleuderberg, *Jour. Phys. Chem.*, 1908.

[9] *Jour. Phys. Chem.*, 1908, p. 120.

[10] *Trans.* Am. El. Chem. Soc., 1904, p. 187.

Can National Professional Societies Be Exclusive and Representative?

The *Mining Magazine*, of London, edited by Mr. T. A. Rickard, prints in its March issue the following editorial, which is entitled "Mining and Metallurgical Society," but has a broader scope and more general interest:

"In April, 1908, the Mining and Metallurgical Society of America was organized, in part as a protest against the nondescript membership of the American Institute of Mining Engineers, and in part to stimulate the discussion of subjects expressly avoided by the Institute, and finally to promote the solidarity of the profession by frequent local meetings. It was intended on the one hand to avoid the duplication of facilities for publishing technical literature and on the other hand to stiffen the moral backbone of the profession by the crystallization of opinion in regard to the ethics and customs of mining engineers, metallurgists and geologists.

"The society started with a nucleus of representative men and at once reached a membership of 100, which on Jan. 1, 1910, had increased to 163, of whom 35 are geologists, 27 are professors, and 4 are editors. The other 97 are active mining and metallurgical practitioners. Of the several geographical divisions of the society formed with a view to local meetings, only two have evinced any life, namely, New York and San Francisco. At Philadelphia one or two small meetings have been held, but the members living near Chicago, Denver, Salt Lake, and similar centers of industry, have failed to organize for purposes of meeting. At New York and San Francisco the attendance has not exceeded an average of a dozen at each monthly meeting.

"Despite the smallness of attendance and the lack of co-operation between the general body of the members, the society has exerted a useful influence by expressing decided opinions in regard to company reports, the duties of consulting engineers, mine inspection, the Bureau of Mines, share speculation, and other matters having a vital bearing upon the business of mining. It is noteworthy that a deputation of the society, headed by the president, recently waited upon the stock-listing committee of the New York Stock Exchange and submitted the resolutions passed—at an earlier meeting of the society—with regard to the protection of mining investors. The resolution, and the members that submitted it, were well received. In another direction the society has also done good work, for the American Mining Congress has been pleased to ask the society to express its opinion and to formulate suggestions in regard to new laws regulating mine inspection.

"Thus it is apparent that the Mining and Metallurgical Society has won respect and has been able to exercise a beneficent influence. To complete this sketch of its career we may add that Mr. H. S. Munroe was the first president; he is succeeded by Mr. J. Parke Channing. No better men could have been selected as standard-bearers. The new officers include Mr. F. W Bradley as first vice-president and Mr. C. W. Goodale as second vice-president. Here also the excellence of selection is obvious. Mr. J. R. Finlay was the secretary and Mr. W. R. Ingalls was and remains the treasurer. As far as relates to the personality of its officers, the society is peculiarly fortunate.

"And yet the Mining and Metallurgical Society of America is a fiasco. It masquerades under a name far too comprehensive. No group of 160 men has the right to speak for the thousands of mining engineers, metallurgists, and geologists contributing at this time to the industrial development of the American continent. If there be not 1000 competent mining engineers and metallurgists worthy to rank as properly qualified practitioners, then the mines and smelters of the United States, Canada and Mexico are worse served than we had supposed. At the very inception of the organization it was proposed to issue a Blue Book of the elect, it is true that this proposal was killed, but apparently the spirit that animated the proposal, however well intentioned, lived to stimulate an exclusiveness wholly uncongenial to the main body of professional men in America.

"The society has failed to be representative because it has aimed to be exclusive.

"Soon after it was started a number of men applied for membership, but most of them were blackballed. Among these were two or three men as fit as any of those already enrolled. Apparently a few egregious blockheads deemed that they could place themselves on a pedestal by limiting the membership to the select group who became charter members before a severe selective process has been set to work. The scandal created by the refusal to admit several first-rate men into the society led to a revision of the by-laws, whereby the election of new members was transferred from the society as a whole to the executive council. Since then the proportion of applications rejected has been sensibly diminished, but the society has not recovered from the irritation produced by the crude effort to be select by blackballing all applicants. Only 27 new members were added in 1909, and 8 members resigned, including two or three men of the highest repute.

"We note that the retiring president referred to the failure to enlist the many professional men that should be members of a society arrogating a national name. He said: 'We have every reason to be proud of the character and attainments of our present members, but we have not, as yet, enrolled even a majority of the reputable and desirable engineers of the country.' Speaking without prejudice, as one fortunate enough to be within the pale, the present writer deems it proper to lay stress on the wholly un-American sentiment that has led this new society into a great blunder.

"The attempt of a fortuitous aggregation of individuals to sit in judgment on the members of a profession that includes thousands of men of varying education and occupation has failed utterly. In the effort to remedy the nondescript character of the American Institute membership, the new society went to the opposite extreme.

"The result is to indicate the difficulty of applying any known process of selection.

"Thus, in justice to the Institute, and particularly its veteran secretary, it must be confessed that the generous policy is the better of the two, simply because, humanly speaking, no tribunal exists by which character may be weighed, abilities appraised, and accomplishment valued. Men who are fit to-day may be unfit a year hence; the man who blundered a few years ago may be among the most efficient now; the personal equation still remains an insoluble problem.

"A mining engineer is the man who does the work of the mining engineer; whether he does that work well or ill it is for a jury of his peers to decide; but who is to select the jury? No self-constituted tribunal can assume disciplinary powers. Among the charter members of the Mining and Metallurgical Society are several whom we, in the exercise of a most fallible judgment, would exclude; among those refused admission are several with whom we would be proud to be associated.

"The lesson is one that should be learned by all engineering societies; for example, by our (British) Institution of Mining and Metallurgy.

"It is well enough to require certain qualities and to promote homogeneity as nearly as possible, but it is easy to err on the side of hypercritical requirement. Engineering societies arrogating national and even international guidance of world-wide professional activities must be representative before they dare to be exclusive.

"A professional organization is not a social club. An institution with only 1200 fully qualified members or a society of 160 is not warranted in the assumption that the one represents all the mining engineers, metallurgists and economic geologists of the British Empire, any more than the other can be permitted to assume to be spokesman for the thousands of professional men engaged in honest and efficient work on the continent of North America."

The Electrical Testing Laboratories and Electrochemistry.

By Clayton H. Sharp.

The electrical arrangements required for the conduct of any single electrochemical experiment are usually fairly simple and require from the laboratory in which they are conducted no elaborate equipment. There must be a supply of current of the proper strength and of the proper potential, arrangements for controlling and measuring the same, and the furnace or tank, or other vessel, containing the substance to which the electrical

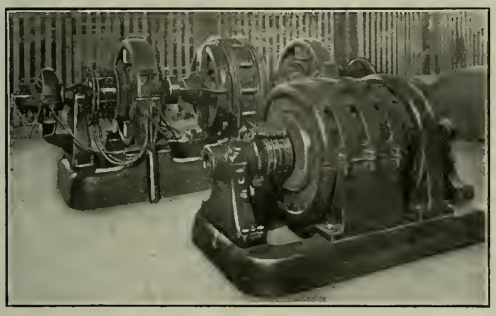


FIG. 1.—HARMONIC GENERATOR SET AND THREE-WIRE MOTOR GENERATOR.

energy is applied. When considering the laboratory equipment required for conducting electrochemical work in general, however, we are struck at once by the great range of the equipment and resources required to handle all classes of work. Taking the case of the supply of current alone, there must be available current of several thousand amperes at low voltage; current of several hundred amperes both direct and alternating at ordinary supply voltages, and current at very high tensions, but of limited intensity. From the one extreme of the heavy current of low voltage to the other of feeble current at high tension, it is necessary to be equipped to supply nearly any intermediate step.

Each step of an electrochemical process needs to be controlled by making accurate electrical measurements. This involves an equipment of electrical standards, and also a complete outfit of commercial electrical measuring instruments covering the entire range over which measurements may have to be made. The above requisites are the fundamental ones. When they are at hand and suitable space is also available, a laboratory may be said to be well equipped for carrying on electrochemical experiments of practically any kind, since in such research the energy-receiving device is usually of a special design, such as must be constructed by the experimenter in any case and cannot be provided for in advance. Laboratories in which facilities along the lines just outlined are available and in which the experimenter in electrochemistry may personally carry on such experiments as he desires, or may have them performed under his direction by the laboratory's testing force, are not so numerous as to make a description, in this place, of the equipment of such a one superfluous.

The main laboratory of the Electrical Testing Laboratories, being located in New York City, is within easy reach of a larger body probably of those interested in electrochemistry than would be the case if it were located at any other point in the United States. The plant is housed in a very substantial building, most of which is of fireproof construction, and which has a large stack, which formerly carried away the products of combustion of a battery of 14 steam boilers. This stack furnishes an excellent ventilating appliance. The points in the equipment of the laboratory which are of the most significance for electrochemical work will be described in much greater detail than those features which have no such immediate bearing, although they may be of great importance for other lines of electrical work. The description may fall under the general heads of the Supply of Current, the Standards and the Apparatus and Methods.

Supply of Current.

The supply of current is obtained from the New York Edison Company through underground cables. The main portion of the supply is furnished in the form of three-phase alternating current, 25 cycles and 6600 volts. The cables are capable of transmitting from the great Waterside Stations of the Edison company a large supply of energy in this form. Besides this, cable connection is had with the direct-current 120-240-volt service of the Edison company. Two cables furnishing this connection are taken from the street mains, while a third is connected directly to the end of one of the feeders of the three-wire system and consequently being a direct transmission from a substation equipped with rotary converters and storage battery furnishes current which is free from erratic fluctuations in voltage due to varied loads on the distributing mains. A total of 500 or 600 amp at 240 volts is available from this source.

In the ordinary operation of the laboratory the high-tension three-phase current is used for driving a 150-kw motor-generator which furnishes the requisite current for the life testing of incandescent lamps. The high-tension supply is also stepped down in a three-phase transformer, thus making available a supply of 25-cycle current at 220 volts. This may be used directly in experimental work and is useful also for driving motor-generators. The motor of the set for the life testing of incandescent lamps is wound for 6600 volts, while the generator is a three-phase, 60-cycle, 120-volt machine, from one phase of which the lift-test operations are carried on. The generator voltage is maintained very constant by a Tirrell regulator, supplemented by hand regulation through a sliding contact rheostat in series with the alternating-current helix in the regulator. Except at times of maximum life-test load a supply of 60-cycle current is available from this machine which is very useful in connection with electrochemical work.

The Edison direct-current service, besides being available directly for experimental work, is used to drive the elevator motor and other motors, to charge the storage batteries and to

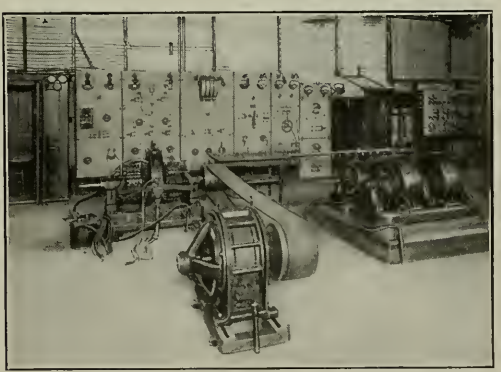


FIG. 2.—HEAVY-CURRENT DIRECT-CURRENT GENERATOR.

light the building. An additional three-wire service is furnished by a motor-generator of special construction, consisting of a 220-volt, 25-cycle induction motor of 50 hp rating, directly coupled to two direct-current generators, each rated at 120 volts and 15 kw. These machines are equipped with slip rings so that alternating current of approximately 37 cycles per second can be taken from them and so that their voltage can be controlled

by the use of Tirrell alternating-current regulators. In order to permit the use of this type of regulator each machine is equiped with a separate directly connected exciter. The very steady direct current at 120-240 volts furnished by this set is utilized chiefly for burning tantalum lamps on life test, it being a well-known fact that tantalum lamps have a somewhat uncertain life when operated on alternating-current circuits.

Two sources of direct current at low voltage are available. One of these is a set of four storage cells with an eight-hour discharge rating of 250 amp; these cells can be connected in series or parallel, as required, and are capable of delivering several thousand amperes for a short time. They are chiefly useful in connection with the calibration or standardization of instruments. Another source of very heavy currents is a double-commutator generator, which is belt-driven from an induction motor and the field of which is excited from a 120-volt circuit through a rheostat. Each side of the generator is rated to deliver 1500 amp at 8 volts. The sides may be put in series or parallel, so that 1500 amp at 16 volts are available, or 3000 amp at 8 volts. This is the normal rating of the machine. As high as 7000-amp currents have been taken from it without difficulty or distress, and the limit has not been reached. This machine is used at times for the calibration of large electrical

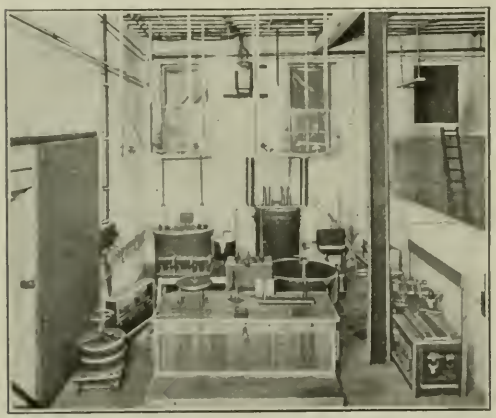


FIG. 3.—HIGH-TENSION LABORATORY. 250,000 VOLTS.

instruments or circuit-breakers, and when so used the terminals of the machine are short-circuited through the instrument and a standard manganin resistance. The current is controlled by varying the field current and consequently the voltage of the machine. Its utility in electrochemical work is manifest.

Heavy alternating currents are obtained by means of two step-down transformers, wound for 100 volts primary and 4 volts, 8 volts or 16 volts secondary. Each transformer is mounted on a truck so that it is readily portable, the truck carrying also a switchboard, by means of which the four sections of the secondary winding can be placed in series or in parallel. The normal rating of these transformers is 12 kw, so that they are capable of delivering continuously 3000 amp at 4 volts, 1500 amp at 8 volts or 750 amp at 16 volts. These two transformers can be operated in parallel or in series.

The equipment of storage batteries includes among others one battery of 70 cells with an eight-hour discharge rating of 60 amp; one battery of 70 cells with eight hour discharge rating of 30 amp; four batteries of 10 cells each of 30 amp; one battery of 70 cells of 7.5 amp, and 400 cells of 0.75 amp. For the purposes requiring a greater range of potential than is available from the batteries, namely, about 1000 volts, a set of five direct current generators is provided. These generators are driven by a common motor and each one is insulated from the earth and carries its own directly connected exciter. In series, these generators will supply direct current at 12,000 volts.

The supply of alternating current at high voltage is obtained from transformers. For voltages up to 6600 ordinary lighting transformers are used. For higher voltages there is available a transformer rated at 10 kw with 100-volt primary and four secondary terminals of 30,000 volts each, giving a total of

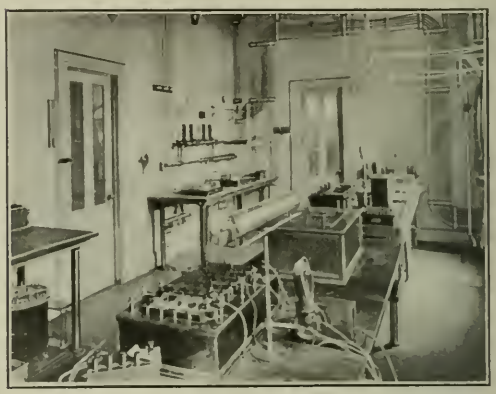


FIG. 4.—STANDARDIZING LABORATORY.

120,000 with the secondaries in series. There is also a transformer rated at 50 kw stepping from 220 volts to 250,000 volts. This last voltage seems high enough to meet all the present practical demands of testing and experimental work. The voltage of the secondary of the high-tension transformers is controlled by a variable-ratio auto-transformer in the primary circuit.

The main part of the work in which high tensions are required is involved in the inspections of high-tension insulators

FIG. 5.—APPARATUS FOR STANDARDIZING CURRENT TRANSFORMERS (UP TO 5000 AMP).

for long-distance transmission lines. These inspections are made in the insulator factories.

There are two other sources of alternating supply which are, however, chiefly interesting in connection with testing and standardizing work. One of these is a pair of alternators exactly alike, placed on the same bedplate. These machines may be coupled together or may be driven separately at any required

speed, three-step pulleys being employed for the wide variations in speed and motor control for the finer adjustments. They are of the revolving-field type with their armatures arranged so that they can be rotated by means of a worm through 180 electrical degrees. The particular value of this arrangement is in connection with tests of meters where power-factors, other than unity, are desired. For this purpose the current for the field coils of the meter is taken from one of the generators, usually with a step-down transformer, while the potential is taken from the other generator and any desired phase relation is obtained between the current and voltage by shifting the armatures.

The other set referred to consists of a direct-current motor having three alternators on the same shaft. The motor is wound for 120-240 volts to give a wide range of speed variation. The alternators have the numbers of their poles in the ratios of 1:3:5. Their armatures are wound to carry the same current. The armatures of the two generators having the largest number of poles can be rotated by means of a worm. This set is designed primarily to furnish alternating current of any required wave-form by the superposition on the fundamental of the third and fifth harmonics. The amplitude of these harmonics is controlled by the field rheostats, while their phase relations can be changed by moving their armatures. Apart from its usefulness for the purpose for which it is directly designed, this set, furnishing as it does the wide range of alternating frequencies up to 300 cycles per second, is a very practical one in connection with the testing work.

Standards.

With the introduction of new and simplified methods of measurement, the discovery of new materials, the adoption of new designs, and the establishment of the great national standardizing laboratories to which the question of the accuracy of standards can be referred, the making of electrical measurements of precision has been in recent years greatly simplified.

The fundamental standards which must be on hand for electrical work are those of electromotive force and resistance. It is desirable also to have suitable standards of capacity and of inductance. The universally accepted standards of electromotive force are the Clark cell and the Weston cadmium cell. Both of these types of cell are at hand. The advantage of

FIG. 6.—CHEMICAL LABORATORY.

having both types is that the accuracy of the standards can be determined with somewhat greater certainty if it is possible to check the two types against each other. The ratio of the electromotive force of the Weston normal cell to that of the Clark cell is known with very great accuracy and hence an intercheck of this kind is of considerable value. For practical work, the Clark cell is very little used. The Weston normal cell, i.e., the cadmium cell, with an excess of cadmium sulphate, forms the most desirable primary standard. It is found in the operation of the laboratory, however, that it is usually more convenient to use secondary standard cells having a zero temperature coefficient. Consequently, the cells manufactured by the Weston Electrical Instrument Company (in which the solution of cadmium sulphate is saturated at 4° C.) are in con-

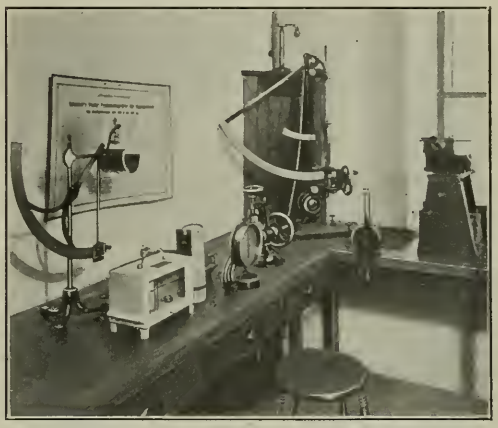


FIG. 7.—SCHOPPER TESTING MACHINE AND IVES' CALORIMETER IN PAPER LABORATORY.

siderable use. Cells of Hulett's reversible type, containing 1 per cent of sulphuric acid, are also in use and are very convenient working standards. These cells have the advantage that they are not damaged as a result of allowing them to deliver very considerable currents. Considerable success has been had also in the use of large reversible cadmium cells as working batteries for high resistance potentiometers. Two such cells of the dimensions given by Professor Hulett will deliver the current required by a 15,000-ohm potentiometer with remarkable constancy. The current may be interrupted when the potentiometer is not in use, and on closing the circuit little or no adjustment of the auxiliary resistance is necessary unless measurements of a considerable degree of refinement are to be made.

Standards of resistance of the Reichsanstalt type, having manganin as a resistance material, ranging in resistance from 100,000 ohms to 0.0001 ohm by steps of 1/10, are at hand in duplicate and triplicate, while oil-immersed standards of the Bureau of Standards type, ranging from 1 ohm to 100,000 ohms, are also used. While variations in some of these standards have at times been detected, yet they are in general so constant that they make the measurement of resistance exceedingly easy. In the measurement of current strength, the modern practice of using the method of the fall of potential through a manganin resistance is adopted, although a silver voltameter is also at hand. For purposes of measurement of this kind manganin resistances designed to have a maximum capability of dissipating heat are used. The largest of these is a resistance of 0.0001 ohm through which a current of 5000 amp can be passed continuously, and a still larger current for a short interval of time.

A standardized mica condenser serves as a standard of capacity while fixed and variable standards of inductance are also at hand.

For the measurement of temperature, there is available a set of four standard thermometers by Richter, in Berlin, covering the interval between zero and 400°, with 0.5° graduations and with the zero point and 100° point on each thermometer. For temperatures above this, a Callender platinum thermometer

is available. Special thermometers are used for calorimetric work.

In the department of photometry, the equipment of standards includes two Hefner lamps and two 10-cp pentane lamps, one of which is of American and the other of English manufacture. These lamps, however, are not referred to as standards in the ordinary course of photometric work. The real standards are constituted by a series of incandescent lamps carefully sea-

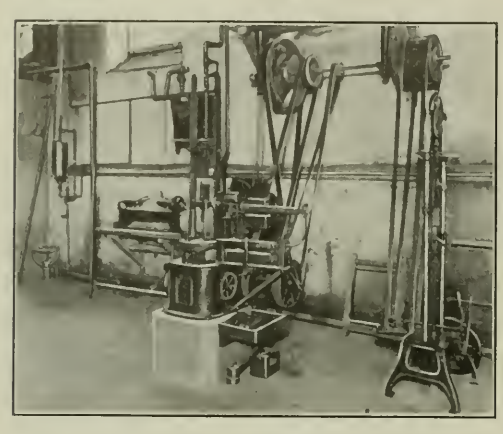


FIG. 8.—TESTING MACHINES.

soned and compared. The value of the unit of luminous intensity represented by these standards is the candle as recognized by the Bureau of Standards in Washington, by the National Physical Laboratory of England and by the Laboratorie Central d'Électricité in France. The greatest care is taken to avoid the possibility of any variation from the photometric standards once set, copies being continually made and compared with the older representatives of the standard and a frequent reference being made to the Bureau of Standards.

Incidentally to the above statement of the fundamental electrical and other standards used, the fact may be mentioned that the laboratories are entrusted with the periodic checking and certification of the accuracy of the standards of measurement of a number of the largest electricity supply companies in the country.

Apparatus and Methods.

In the measurements of electromotive force, the calibration of voltmeters, etc., the potentiometer and the standard cell are employed. Potentiometers both of the high-resistance and of the low-resistance type are used. For operations in which it is desirable to move the potentiometer from place to place the low-resistance type has the advantage that a sensitive pivoted galvanometer which is quite portable and which requires no leveling can be used with it. For precise measurements, the high-resistance potentiometer with a reflecting galvanometer is usually employed. Voltages higher than 1.5 are measured by the use of the volt box multiplier. For the high resistance potentiometer, the volt-boxes are of 100,000 to 200,000 ohms total resistance, for the low-resistance potentiometer, from 5,000 to 10,000 ohms, thereby insuring sufficient sensitiveness in working with the portable galvanometer.

For the measurement of alternating voltages, instruments of the soft-iron type, of the electrodynamometer type and of the electrostatic type are at hand. Of the latter type, mention may be made of a Ayrton Mather reflecting instrument giving full-scale deflection with 7 volts to 8 volts, a Kelvin multicellular electrostatic instrument specially equipped with a telescope and scale and reading to 150 volts, and a Westinghouse instrument reading directly to 120,000 volts, or operating in two lower ranges when one or both of two oil-insulated condensers are short-circuited. The last-named instrument has its moving parts entirely immersed in oil, and indicates high tensions in a very satisfactory way.

Evidently not all of these instruments can be standardized on direct current. The soft-iron instruments and also the electrodynamometers when used on currents of high frequency are subject to error. These are checked by the use of electrostatic instruments as transfer instruments from direct to alternating voltage. The Kelvin multicellular voltmeter with telescope and scale referred to above is the most important instrument for this purpose and when carefully used yields results of relatively high precision.

Ordinary resistance measurements are made on a decade type Wheatstone bridge. For precise work the bridge is used whenever possible by the substitution method. For instance, if another decade box is under investigation, and the 10 coils of 100 ohms each are being measured, the 100-ohm reference standard resistance is first measured by the bridge and then the 100-ohm coils in the box. By this method errors of the bridge are eliminated.

The measurement of low resistances is usually carried out by means of the Kelvin double bridge, though the fall of potential method with a potentiometer and standard resistance is sometimes employed. The Kelvin double bridge used is of the dial pattern with four decades. Among its uses is the measurement of the conductivity of copper and other materials. To assist in the measurement of the conductivity of copper, a set of copper standards of resistance have been prepared, one for each two sizes of the Brown & Sharp wire gage. These consist simply of pieces of copper wire, to the ends of which lugs for convenient attachment are securely sweated, and to which at a distance of about 1 m from each other potential terminals are permanently attached. The resistance of each of these standards is carefully measured at 20° C. by comparison with a manganin standard, the copper being immersed in an oil bath. Knowing the resistance of the standard, a table is prepared to show what length of copper wire of a given weight per unit length would have the same

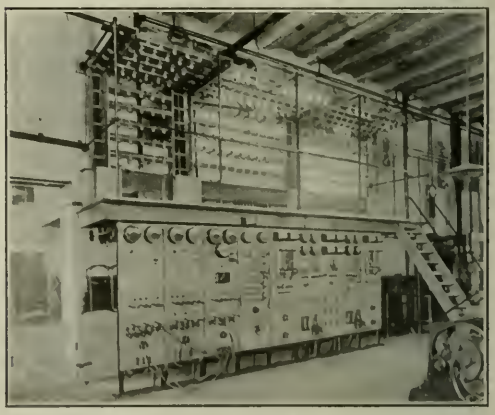


FIG. 9.—SWITCHBOARD AND TANTALUM LAMP LIFE TEST.

resistance as the standard, if its conductivity were 100 per cent in terms of Matthiessen's standard. The wire, the conductivity of which is to be determined, is cut to a length of exactly 120 cm, and is weighed. It is placed in series with the copper standard and knife edges are pressed upon it, giving potential connections at the distance apart which is indicated by the table. The bridge then reads percentage conductivity directly.

In the measurement of high resistances, such as that of insulating materials, in wire and cable tests, etc., a sensitive high-resistance d'Arsonval galvanometer is employed. The source of electromotive force is a special battery of 150 small accumulators installed on large rubber insulators in such a way as to insure immunity from grounds. Where the sensitive galvanometer method of measuring resistance is inadequate, the loss of charge method, using a Kelvin electrostatic voltmeter equipped with mirror, telescope and scale, to measure the potential, is employed.

An abundant supply of ammeters is on hand which can be calibrated accurately, using the standard resistances of low denomination and high-current carrying capacity which are constructed for the purpose.

A set of Kelvin balances, the largest of which will carry 1000 amp, is of particular use in connection wit hthe standardization of alternating-current ammeters and of series transformers for the measurement of current.

Use is made also of a bank of incandescent lamps as a means of standardizing small alternating-current ammeters. The voltage on the lamp-bank is measured by an alternating voltmeter of the electrodynamometer type which gives the same indications on both direct and alternating voltages. The lamp-bank is standardized on direct current and is brought to the same voltage on alternating current, whereby the value of the current is determined.

A supply of wattmeters is also on hand, including precision wattmeters of the Duddell-Mather and of the Westinghouse types. These can be standardized on direct current. Convenient apparatus is set up for the standardization of voltmeters, ammeters and wattmeters, including integrating wattmeters.

The important question of the precise determination of the ratio of transformation and of the phase displacement of current and potential transformers used in the measurement of large currents or high voltages has been gone into very thoroughly and a method developed and apparatus constructed for accomplishing the same. By this method the ratio of transformation is given as the ratio of two resistances and the phase angle is computed from the indication of an adjustable mutual inductance. The design and construction of nearly non-inductive manganin shunts of large current-carrying capacity has been involved and this has been successfully carried out in the laboratory work shop. The largest of these carries 5000 amp with water cooling and has a resistance of 0.0002 ohm. The phase angle between the current and the potential difference in this shunt has been measured and found to be 18 minutes.

By the use of transformers standardized as above the range of precision measurements in the laboratory is extended almost indefinitely.

A supply of apparatus, including among other things a Koepsel permeameter and a Picou permeameter, is on hand for the determination of the magnetic properties of iron and steel. The well-established ring method of testing iron with a ballistic d'Arsonval galvanometer, standardized by means of a mutual induction coil, is preferred for work of this character.

An oscillograph and a "point-by-point" apparatus for getting alternating wave forms are also set up and available for use.

A bomb calorimeter is used for determining the calorific values of coal, and a Junker's calorimeter for determining that of gas.

A small but well-equipped chemical laboratory is provided in which all of the ordinary analyses and other chemical operations required can be carried out. This is furnished with a hood and with gas, hot and cold water and compressed air. Opening from the chemical laboratory is a small balance room in which the balances are supported on a firm shelf.

A spectrophotometer of the best construction, and also an Ives' colorimeter, are available for use in such chemical and other tests as require a careful analysis of color.

The apparatus is also on hand for the accurate comparison of working thermometers with standards. This apparatus has been developed chiefly for the purpose of checking clinical thermometers along the lines pursued by the Bureau of Standards.

A series of machines by Schopper, Riehle and Olsen are on hand for testing the tensile strength of materials up to a maximum of 15,000 lb. This series will soon be supplemented by a machine for tensile and compression tests up to 200,000 lb., and also by one for torsional tests up to 240,000 inch-pounds.

A well-equipped photographic darkroom is also provided.

The laboratory for the measurement and testing of various sources of light is most complete. Its equipment includes a Matthews' arc photometer installed in a photometer room 70 ft. long and used for determining the mean spherical candle-power, the mean hemispherical candle-power and the distribution of light in a vertical plane, of arc lamps and of large gas lamps, such as are used in street lighting, etc. For the determination of mean spherical and mean hemispherical candle-power of large sources of light, there is also an integrating sphere 2 m in diameter constructed of sheet iron and conveniently arranged to give access to the interior and to fit it for practical use.

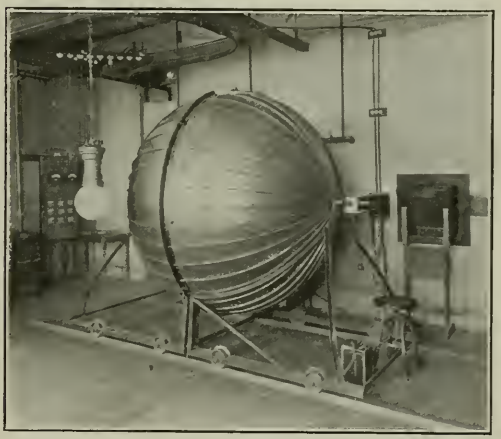


FIG. 10.—INTEGRATING PHOTOMETRIC SPHERE.

A photometer room is set aside chiefly for the photometry of gas lamps. This is equipped with the necessary gasometers, meters, pressure regulators, etc. The photometer track in this room is 5 m long. There is an arrangement of three very large mirrors placed on a frame which can be turned to any angle about a horizontal axis, by means of which the intensity of the light emitted at any vertical angle from a lamp is thrown on to the photometer disk. By the use of this apparatus the curve of vertical distribution of the intensity can be obtained without disturbing the lamp.

The standard photometer room is equipped with a long photometer bar of the Reichsanstalt pattern carrying a Lummer-Brodhun contrast photometer. The normal distance between lights in this photometric outfit is 5 m. The arrangements, however, are such that this can be varied at will. The electrical measurements are made by means of a potentiometer and standard cell. On this photometer, as on all of the others, the greatest attention is given to the question of screening the photometer disk from the influence of stray light, an important point which is too often overlooked in photometric work. The work of this room consists largely in the standardization of incandescent lamps as standards of luminous intensity. The lamps to be used for this purpose are carefully seasoned until

they reach the point at which the rate of change in candle-power is very slow. In the process of standardization their candle-power is read carefully by different observers on different days. In some cases the process of standardizing a lamp extends over some months. In this work, as in all the photometric work, the substitution method is followed, a set of standard lamps first being controlled against a comparison lamp, and afterwards the lamps to be standardized, which are then placed in the same receptacle and are measured by the

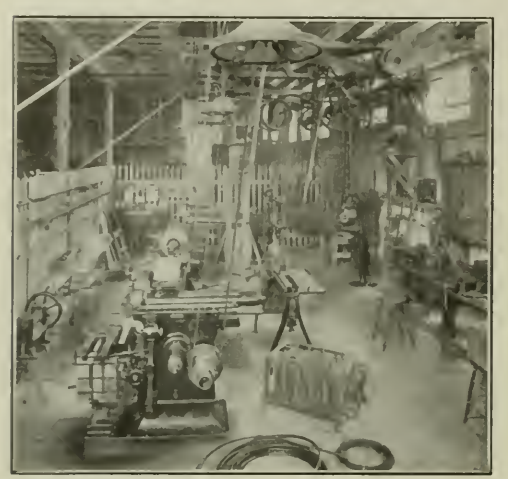


FIG. 11.—MACHINE SHOP.

same instruments as have been used in the case of the standard lamps. Several hundred such lamps are prepared and sold each year to lamp makers, users and photometrists generally.

A large part of the photometric work is incidental to very elaborate inspections and tests of incandescent lamps, which the laboratories are carrying on, and which cover many million lamps per year. The chief part of the inspectional work is carried on in the factories in which the lamps are made, but samples of the lamps so inspected are brought to the laboratories for purposes of a life test. These sample lamps are carefully read on industrial photometers and are put on racks to burn at their proper voltage to the nearest tenth of a volt. At approximately 100-hour intervals, they are removed and re-photometered, whereby their life history is determined. Some 3000 lamps can be life-tested simultaneously.

The industrial photometers, by the aid of which this work is performed, are a modified form of the photometers which are in use in lamp factories. They have been very greatly refined in details, so that they are capable of yielding results of relatively great precision. The Leeson star disk form of photometer is used in this class of work on account of its simplicity and on account of the convenience with which the operators can observe it.

In connection with the incandescent lamp work, there is a Matthews' integrating photometer for determining the mean spherical candle-power of incandescent lamps, and also an integrating sphere 30 in. in diameter which has been found to be a superior instrument for this purpose.

There are also various portable photometers which are found to be of the greatest value in the measurement of distribution of light from sources of large dimensions, in the measurement of mean spherical candle-power in connection with the spheres, and particularly in the measurement of the illumination in rooms, and of the candle-power of lamps on the street. A most convenient form of photometer for this purpose has been developed as the result of the experience so gained.

This laboratory is not without that most important feature of any large working laboratory, namely, a machine shop. Various machine tools of the best types are therein installed and are used for the repair of apparatus and for the construction of new apparatus as required.

A feature of the laboratory consists in a number of private rooms which can be rented by those wishing to conduct their own experiments. These rooms are supplied with water, gas, compressed air, etc., and with electrical connections to the main laboratory switchboard, whereby all electrical currents, both direct and alternating, can be obtained.

The uses to which these private rooms have been put are most varied, but more of the work which has been done has been of an electrochemical character than of any other. Among the work which has been done, bearing upon electrochemical research, has been the experimental development of new types of electric furnace, and by two different parties, on the fixation of atmospheric nitrogen. In the case of one of the latter the plant installed was of such a size as to give data on the industrial value of the process, involving the question of cost, etc. Experiments have also been made in the fusion of quartz for the production of silica vessels, and in the use of the electric furnace in annealing transformer plates. Many tests of primary and secondary batteries have been made. Experiments have been carried on looking to the development of new insulating materials, etc.

The equipment, of which the above is but a meagre description, is being added to continually as requirements indicate, the purpose being to maintain the laboratory in a high state of efficiency and to keep it abreast of all of the latest developments along its line of work.

Plans have been prepared for apparatus which will make a considerably larger supply of alternating current available for general laboratory work, and particularly for electrochemical work. This apparatus will consist of a step-down transformer in connection with an autotransformer whereby any required voltage can be obtained between 5 volts and 600 volts by steps of 5 volts. Its normal rating will be of 150 kva. The considerable supply of power furnished through this apparatus

FIG. 12.—INTERIOR OF PRIVATE LABORATORY.

will enable electrochemical experiments to be carried out on a fairly good scale.

It is hoped and believed that the Electrical Testing Laboratories will be an important factor in the future development of electrical science along the chemical lines as well as along the electrical engineering lines to which its attention has in the past been chiefly devoted, and in which it has received extended recognition and substantial encouragement from manufacturers, users, engineers, experimentalists and many others.

Causes of Non-Uniformity of Blast Furnace Operation.

BY EDWARD B. COOK.

The writer has felt for some time that there is not enough co-operation between scientific men engaged along mechanical and metallurgical lines and the operating men who are running the blast furnaces. The cause of this is misunderstanding. The practical man feels that the scientific man does not understand practical conditions, and that although many of the suggestions made are of value they are impracticable.

It is the object of this article to take up the items, one by one, given in Prof. J. W. Richards' article on blast-furnace operation, printed in this journal, January, 1910 (page 18), and also to discuss the chief difficulty in blast-furnace practice which is causing a great deal of trouble and loss of money. No new information is contained. It is simply a statement of the practical man's troubles and his point of view.

Composition of Ore, Flux and Fuel.

There can be no doubt that this is a matter of great importance and to-day it is usually well looked after. Even where numerous ores, frequently irregular, are used in a furnace, the manager insists on knowing what the ores are and is able to mix them in such a way that there is seldom much disturbance of the flux. At many places cokes varying greatly in ash and sulphur are used, and a fairly uniform iron made from them. There is more or less leeway in flux in a good-working furnace, i.e., the slag can vary 3 per cent or 4 per cent in acids without throwing off the iron, provided the condition does not continue too long.

It has been found helpful at some plants to have the blast-furnace chemist make a monthly report of his analysis of all the materials entering into the furnace throughout the month and make up a theoretical analysis of pig iron and slag, and compare these with the actual analyses from the furnace. This makes the laboratory an integral part of the furnace operation, and if the chemist's reports do not check closely with those of the furnace, he is under the necessity of making some explanation, and he cannot blame it on the sampler, as the sampler is under his direct charge.

It is found that the theoretical slag is usually 2 per cent lower in acids, i.e., silica and alumina, than the actual slag.

Size of Lump.

It is impracticable to keep all the material entering the furnace, using 2000 tons of raw material a day, free from rain and snow—that is, irregularities on account of weather conditions. It probably will always be impracticable, particularly with such a bulky material as coke.

This, however, is not as troublesome a condition as might be expected. Within the last year it has only proved of annoyance once at our plant, and then was a disturbing factor for three days. Water, of course, is of benefit, and is being added largely to the charges before entering the furnace, usually after the material is weighed.

The Western furnaces during the past year have used large quantities of water—in fact, equal to from 5 per cent to 10 per cent in weight of all the ore put into the furnace. The purpose of this is to keep the top of the furnace cool, and also to prevent the loss of fine ore. In some places this water is measured by meter, but oftener it is put in at random and varies greatly in amount. There is no objection, usually, to the water going into the furnace. The only objection is to the varying quantities of water in the raw material before it is weighed, and the consequent changing of the burden and flux of the furnace.

As a general proposition, it is advisable to have all the material going into the furnace broken to a uniform small size. It is, however, impracticable to break all of this material down by sledges, and there are many objections to breaking it with crushers. For instance, at a German plant I inquired of a director whether they did not break any of the material. He replied that 12 years ago they did, but it had been discontinued at that time in order to save flue dirt, and that occasionally they break down large ore with sledges, but most of the material went into the furnace in good-sized lump. Even the dense Swedish ores are charged in big lumps. In this country there have been several instances where furnaces would not work on small limestone, and it was found necessary to use large stone in order to prevent top slips.

On a furnace that is working well, material irregular in size can be used without seeming detriment. As a general proposition it is well to have stone broken down to 4-in. size, provided it can be screened afterward and the dust taken out. It is well to have hard magnetic ores, and hard ores of all kinds, crushed down small, provided the percentage of dust so formed is not excessive, for in many ores loss through dust on account of crushing will very quickly use up the saving to be gained in improved furnace work by better preparation of the ore.

In regard to size of coke, in the same day I have had one successful blast-furnace manager tell me he attributed his success to the use of small coke, and another man equally successful say that he attributed his success to the use of large-size coke.

One cause of great trouble in a blast furnace is the practical man's lack of knowledge of the fusible slag. It is remarkable how little is known about the presence of spinel in blast-furnace slags, and many furnace managers are in difficulty to-day on this account. Others are using a slag entirely too infusible, unable to contain much sulphur, and are making a good deal of bad iron in consequence. The justification for this lack of knowledge is that few of the blast-furnace men have had the advantages of scientific training, and even those that had, do not know where to go for detailed information in regard to slags.

Intervals of Charge.

There can be no doubt about the necessity of regular charging. We are, however, very much troubled with poor distribution, and although great advances have been made in the last six years, so that now there can be no doubt but that mechanically filled furnaces with revolving tops give as good distribution as the old hand-filled furnaces, still we feel that the distribution is not all that it should be.

The modern furnace is charged as continuously as possible to be consistent with good distribution, i.e., in order to get the best distribution, a certain number of pounds of coke or a certain number of pounds of ore, or both, must be lowered in the furnace at one time in order to give a proper thickness or layer on top of the furnace. Most furnaces are equipped with a recording instrument showing the gas pressure on the furnace top. It shows exactly when the furnace was filled, day and night, and is a great aid to the superintendent in obtaining regular charging. We will probably never attain absolutely continuously filled furnaces until we have some sort of revolving or stirring arrangement to give distribution, such as has been adopted in modern gas producers.

Running Out of Iron and Slag.

The writer has frequently heard of the continuous-running cinder notch, but has never seen one, either in this country or abroad, and certainly would not venture to put any in at a blast furnace which he was running. The reason for this attitude is that furnaces appear to accumulate in the hearth a certain amount of so-called "dirt," which is mostly fine coke—not consumed in the furnace—together with some graphite. The only way to get this material out of the furnace is to wash it out with the slag.

If a small cinder notch is used it is impossible to keep the furnace clean. If too large a cinder notch is used the furnace is kept clean by washing out this material, but there is a much larger loss of iron carried out of the furnace mechanically with the cinder.

The manager tries to compromise between these conditions and often changes the size of a monkey or cinder notch, according to the condition of the furnace. It is hardly likely that this mechanical difficulty of the ordinary blast furnace will ever be done away with.

In regard to tapping of iron, most large furnaces to-day are tapped every four hours, some of them every three hours. They are tapped at a regular time in this country.

There is one furnace in England, however, which is used as a mixer for the steel works, in that it is tapped for a ladle of iron whenever the steel works asks for it. This is a small furnace, and the condition could not be met with our present skill on big furnaces. It is essential that a body of iron of certain size be accumulated in the blast furnace or else the bottom builds up so that the furnace cannot be tapped. There is no slacking of the wind to flush, nor is there any slacking at cast, ordinarily, except to shut in the iron notch. This time of "shutting in" has been reduced to an average of 30 to 40 seconds. We have run one furnace for two weeks without even slacking the wind to shut in the furnace. We find, however, that there is danger of burning men in this practice and that there is very little detriment to the furnace by throwing off the wind for 30 seconds once in four hours.

Temperature of Blast.

Although equalizing stoves have been invented for a number of years, to the writer's knowledge they are only in use at two furnaces in the United States. The ordinary blast-furnace manager is not prepared to recommend the expenditure of the money required for this installation, and practically no owners are willing to make the expenditure, even when the blast-furnace mahager does recommend it.

Most managers attempt to keep the temperature of the blast fairly regular by regulating a cold-air mixing valve. This, however, has many objections, as frequently the air does not mix properly, and one side of the furnace is receiving much colder air than the other. Another idea is to work two stoves continuously on the furnace—and have only two stoves on gas—regulating the cold-blast valves of the two stoves on the furnace so as to maintain the temperature reasonably uniform.

The difficulty with all these devices is that the temperature cannot be maintained uniformly high, but is usually maintained uniformly low, and there is a great loss of heat units.

There is not nearly as much irregularity caused in a blast furnace by a variation of heats of 200° in one hour as would be supposed.

Furnaces run along for years and have their heats vary from 100° to 200° every hour without showing unusual irregularity as compared with those receiving regular heats.

In this country we have been unable to use high heats. The stove equipment is far below that of German practice, but most managers do not use what they have—whether on account of our coke and ores or lack of knowledge of blast-furnace construction, we have been unable to get our furnaces to do good work on heats as high as those used in Germany. Up to two years ago the successful manager was usually using not over 900° Fahr. To-day 1000° to 1100° is being used by many managers, but there are still a large percentage sticking to the low heats.

When in Germany I asked a blast-furnace director why he did not run his heats over 1475° Fahr. He replied that the furnace would not stand any higher heats, but stuck fast, and, indeed, it was necessary for him to use all cold air after each cast for half an hour to get the furnace to "move off," i.e., start to settle again.

In the East with magnetic ores higher heats are used, but even there there seems to be a limit of about 1200° except at certain times when the furnace turns cold. It will be a very great advantage to the blast-furnace business if someone can devise a furnace on which high heats can be used continuously.

Moisture of the Blast.

There can be no doubt but that the low moisture of the dry blast is the great value and that perfect uniformity is a second consideration. One blast-furnace manager regularly used steam in his air in irregular weather in order to keep the moisture entering the furnace uniform and to keep the furnace from sticking fast by getting too hot high up on low-moisture days. This, however, was unsatisfactory.

It is evident to the manager that irregular moisture, around 1 gr., is of very little importance, while moisture around 2 gr. makes a great deal of difference, so that although uniformity is of value, the main thing is *low* moisture.

In regard to keeping exactly the same weight of oxygen entering the furnace, this also is a refinement which at present we have not found any great necessity for. I know of several managers who varied the revolutions of their blowing engines in proportion to the temperature of the air, so as to deliver as nearly as possible the exact same weight of oxygen to the tuyeres, but this has always been discontinued after a certain length of time, and I do not know of anyone who is attempting to do it to-day. Blowing engines are carefully watched, however, and revolution counters used to make sure that the same number of revolutions are run hourly, also clock-speed recorders to show that the revolutions have been regular during the hour.

At our dry-blast furnace we have attempted to keep the temperature of the air entering the blowing engines regular as well as to keep the moisture regular. We have not, so far, been able to demonstrate that this regulation of the temperature is essential.

The writer does not wish to take the position that regularity as regards composition of material, size of lump, interval of charging, running out of iron and slag, temperature, moisture, and weight of blast are unnecessary and not worth the trouble of attempting, but the point to be made is that practically none of these things, as ordinarily cared for in an average plant, are interfering with fairly regular running and reasonable profits. There is, however, one particular feature which does materially and seriously interfere and which will now be considered.

Furnace Lines and Accumulations on the Bosh.

There is a saying among blast-furnace men that if a furnace is built right anybody can run her; if she is not built right nobody can—and the truth of the matter is that none of us know how to build one right.

There are certain things that have been developed—for instance, that it is not economical to attempt to burn over 4 lb. of coke per square foot of hearth area per minute, and that the furnace must be built of certain size of hearth in order to make the requisite tonnage.

From experience it has been found that it is not well to have too great velocity through the tuyeres—in fact, a velocity measured by a difference in pressure of 1 lb. is about right. It has also been determined that the economical height is somewhere between 85 ft. and 95 ft., but it is not known what is the proper height of the bosh or the proper angle, nor consequently the diameter of the bosh.

It is not necessary here to go into the lines in detail, nor devices for maintaining the lines at stock line, hearth, etc., except to say that there has been great improvement in recent years.

Accumulations on the Bosh.

In *Stahl und Eisen* Professor Osann speaks in his article on the dry blast of furnaces that do not run regularly, the usual cause being that the sectional area of the furnace has become lessened. This is exactly the trouble, and is the trouble which keeps the blast-furnace manager awake at nights and is responsible for his general uneasiness. This lessening of area is no longer due to the old high-melted scaffold, which troubled our fathers. On account of improved working conditions and

lower fuel consumption the high melted scaffold has practically disappeared and we seldom hear the word "scaffold" to-day.

The present lessening of area is about at the top of the bosh, sometimes just below the top of the bosh, and sometimes just above. A furnace in this condition is usually spoken of as having accumulations on the bosh or being a dirty furnace. Sometimes this condition happens in blowing-in, and the furnace is never able to do good work. At other times it will hardly be felt for the first year. Ordinarily, however, it occurs in from four to six months. It has been noticed again and again that a furnace will do very good work for four to six months and then will start to drop back in tonnage, and the fuel consumption will increase largely on account of the irregular settling, i.e., hanging and jumping.

The cause of this accumulation is not known. To be sure, bad lines or any mistake in running or charging seems to accentuate it. There is also a difference according to the material used in the furnace. For instance, there is a great deal more trouble of this kind with magnetic ores than there is with hematites. The material seems to be largely unreduced ore with more or less lime. Some managers blame it on their limestone, others on the quality of the ore used, and still others blame faulty distribution. These are the main causes advanced for furnaces where everything else is closely looked after. Of course, excessive velocity of wind through the tuyeres or some other mistake of this kind accentuates the trouble.

The trouble is world-wide, but is very much worse in some districts than in others. If this condition cannot be remedied, the furnace has to go out of blast or run on very irregular iron produced at excessive cost. If hematite ores are being used, it is not so difficult to clean the furnace, but with magnetic ores the trouble is frequently fatal, as the material, if by chance it comes off at once, is sure to bung up the hearth of the furnace. The cleaning process in any instance takes time and is a very expensive matter.

In the way of prevention, changing the lines of the bosh has been tried without much resulting benefit. Bosh jackets have been substituted for bosh plates, and *vice versa*—in fact, the blast-furnace managers keep wandering around in a circle as far as these things are concerned, always wanting what they have not got. There seems to be no way whatever to absolutely prevent the trouble. In England it is met at one plant by tuyeres on the bosh, which are used either part of the time or all the time. This, however, must greatly interfere with the zone of fusion in the furnace and cause high fuel consumption.

The men who are making the least "off" iron and having the least trouble to-day are those who are blowing slowly, keeping stove heats in reserve, and giving the furnace cleaning blanks every three or four days to prevent the formation of accumulations on the bosh.

Professor Osann suggests that the dry blast be used, together with controlling amounts of steam. Steam or water put into the tuyeres undoubtedly help to remove the accumulation, but dry blast will not do it.

As a preventative, dry blast is a great help, as it lowers the zone of fusion in the furnace and brings whatever accumulations there may be lower down. As a proof of dry blast's help in preventing this trouble, it is noticed that the dry-blast furnaces last year surpassed all other furnaces in average work. It is the writer's opinion that the same furnaces using the same material will take twice as long before being inconvenienced by accumulations on the bosh if dry blast is used, but dry blast alone will not remove the accumulations once formed.

While the furnace is "building"—that is, accumulating on the bosh—the work is excellent as the radiation from the bosh is reduced, the heat concentrated in the center of the furnace, and conditions are ideal. When the "building" has continued too long, however, irregular settling interferes with any kind of proper work in the furnace. If the bosh is kept clean by the use of short large tuyeres good work is impossible. The practice in Germany, and in some places here, is to try to adjust these conditions with the tuyeres, but results are not altogether satisfactory, and there is a certain period of high cost while the furnace is being cleaned.

The salvation of blast-furnace practice is going to be a furnace which can be blown out and put in again in a short time and without excessive expense. Mr. James Scott has done more than any other man in the United States along this line, as he has had the courage to put up a blast furnace containing only 12-in. of brick on the inwall. It can probably be put out and in again within two weeks and without great expense.

So far this furnace has worked very well, and is proving of particular interest, as it gives new information in regard to the loss by radiation and also in regard to the temperature of various zones in the stack. If this type of furnace is generally adopted there will be no longer any excuse for fighting poor working furnaces which contain dirt accumulations.

Blast-furnace managers to-day are not writing many articles —they are all too busy with their troubles to write about them, and for this reason there is not as rapid an improvement in blast-furnace practice as there should be. Many men are troubled with problems that have been solved by their neighbors.

In Germany the blast-furnace managers meet frequently and have papers on various subjects, by both theoretical and practical men. They also visit works in a body. It is to be hoped that something of the kind can eventually be attained in this country. In the meantime it is extremely desirable that someone should start to make scientific experiments in regard to blast-furnace lines. No private individual can afford to do this work. It would seem, however, a very good investment for the largest blast-furnace interests.

Warwick Iron & Steel Company,
Pottstown, Pa.

Metallic Sodium from Fused Sodium Chloride.

BY C. F. CARRIER, JR.

During 1904, the author commenced a study of the art of separating metallic sodium. This investigation was carried on for two years and published in the form of a critical, bibliographic summary in ELECTROCHEMICAL AND METALLURGICAL INDUSTRY, v. 4: p. 442 and 475 (1906). In this same period the Castner sodium process was experimentally tested, and a proposed process of H. Becker thoroughly investigated on a laboratory scale. (*Electrochem. Ind.*, v. 2; p. 357 (1904) and *Zeit. f. Electrochemie*, v. 10, p. 568 (1904). The most important result of this latter investigation was that the temperature permissible in the electrolysis of fused caustic soda is considerably raised by the addition of sodium carbonate to the electrolyte.

As a result of these investigations and researches, it was concluded that the Castner process had attained about all that could be expected from the electrolysis of fused caustic soda. The history of the efforts to produce sodium by the electrolysis of fused sodium chloride did not offer any encouragement in that direction, but still the cheapness of the raw material and the directness of the process made it the most inviting line of attack.

The Acker process had demonstrated that sodium-lead alloy could be produced on a large scale with good efficiency by electrolyzing fused sodium chloride in the presence of a melted lead cathode. This suggested the possibility of using the preparation of the sodium-lead alloy as an intermediate step in the preparation of sodium. In 1903, Prof. J. W. Richards suggested to his students the possibility of refining metals by using the impure metal as a fused soluble anode in a fused electrolyte and plating the desired metal over to the cathode in the fused condition. The same idea has been advanced by

Borchers for the refining lead (*Zeit. f. Electrochemie*, v. 1; p. 13 (1894), and also by Hoopes for refining aluminum (U. S. P., No. 673,364 of 4-30-01).

From the above observations were evolved the general lines of a possible solution of the difficult problem of obtaining sodium from fused sodium chloride. If the Acker furnace were to be so modified that the lead-sodium alloy could be passed into a second compartment and there made a soluble anode in a fused electrolyte that would not consume the sodium, it would seem possible to recover the sodium at the cathode. This would make a furnace analagous to the Castner caustic cell, fused sodium chloride replacing the brine, melted lead performing the function of the mercury and the sodium being liberated in a fused bath instead of being allowed to react with water to form caustic.

This idea also forms the basis of the apparatus devised by E. A. Ashcroft (U. S. P. No. 788,506 and No. 801,199, *Trans.* Am. Electrochem. Soc. v. 9; p. 123, 1906, and *Electrochem. & Met. Ind.* v. 4; p. 145, 1906). The investigation to be described below was abandoned for financial reasons before entirely completed, but such a favorable state of development was reached that the conclusions are of value as well as of general interest, and are made public at this time through the courtesy of the Elmira Electrochemical Co.

Preliminary Trial.

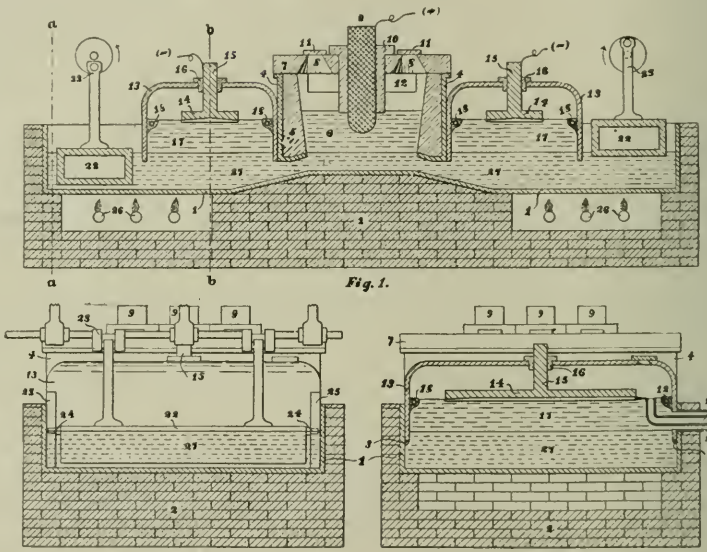
A small laboratory apparatus was constructed in the laboratory of the Technische Hochschule zu Karlsruhe in 1904. This consisted of a sheet iron box about 12 in. x 4 in. x 3 in. deep, divided into three compartments by sheet partitions reaching nearly to the bottom. Each compartment was provided with a sheet iron shell whose lower edge was about twice as far from the bottom of the box as the lower edges of the partitions, and which were of such size that about ¼ in. space was left between the shell and the sides of the compartment. This space was filled with Portland cement, leaving an opening beneath the partitions.

The box was mounted on a fulcrum and eccentric so that it could be rocked in the same manner as the Castner caustic cells. The bottom of the box was filled with sufficient melted lead to form seals below the partitions and a gas burner was provided to keep the contents melted. All three compartments were filled with melted caustic soda, the two end ones being the anode compartments and the middle one the cathode compartment. The capacity of the apparatus was about 50 amperes.

Only a few trials were made with this apparatus, for the results could not be conclusive either for or against. The cement fouled the electrolyte, causing objectionable foaming and unsatisfactory operation generally. Sodium was liberated at the cathode but from the conditions of the experiment, it would be difficult to prove whether the caustic in the anode compartments was the real source or not. Very little gas was evolved at the cathode, which is not the case in the simple electrolysis of fused caustic, therefore it was assumed that a portion at least of the sodium was being produced according to the theory of the process. It was, therefore, determined to test the process on a commercial scale and the above mentioned company was formed for the purpose of erecting and operating an experimental plant.

Electrical Equipment.

It is impracticable to treat fused sodium chloride on a small scale, therefore a dynamo of 1000 amp. capacity was secured, although in the light of experience 2000 amp. is to be preferred. The dynamo was seperately excited by means of a small 110-volt dynamo. Regulation was partly obtained by a speed controller on the motor driving the dynamo, but chiefly by regulation of the field of the exciter, in series with which was a rheostat and a bank of incandescent bulbs. In this manner the voltage could be varied from 2 to 16, the capacity being 10 kw. The main circuit was provided with a 1000-amp. ammeter and a voltmeter. There was also a shunt circuit with a capacity of 300 amp. and a corresponding ammeter. By means of a water-pipe rheostat, the amount of current passing through the cathode compartment could be regulated accord-



FIGS. 1, 2 AND 3.—PISTON TYPE FURNACE

ing to the efficiency of the operation of the anode compartment as will be explained below.

Piston Type Furnace.

This furnace was designed for 800 amp. capacity, allowing a margin for possible increase in the current density without overloading the generator. The general features of this type will be more clearly understood from the illustrations, Figs. 1-3, from U. S. P. No. 830,051, but some modifications were made in the experimental installation, which will be specially noted later. The apparatus will first be described as planned. The changes and reasons therefor will be described in detail, thus marking the progress of the research.

Fig. 1 is a longitudinal, vertical section. Figs. 2 and 3 transverse vertical sections on the lines *aa* and *bb* respectively in Fig. 1, looking to the right. A rectangular, cast-iron pan 1 is set in masonry 2. The other portions of the apparatus up to 18 inclusive are supported within the pan by shoulders 3. Over the raised portion of the pan is located the anode compartment, enclosed by a cast-iron frame 4 which is lined with fire brick 5, which will withstand the fused electrolyte 6.

The cover 7 consists of a cast-iron frame filled with a fireclay concrete composed of broken brick, fire clay, and a little cement. Feed holes 8 and openings for the graphite anodes 9

are also provided. To protect the anodes from oxidation, they are protected by means of earthenware sleeves 10. The feed holes are covered with simple slabs 11. A port 12 located at the back of the anode compartment carries off the gases formed. The two cathode compartments are in the form of an inverted, rectangular, cast iron box, opening downward, forming a sort of bell 13. The cathode proper is an iron plate 14 supported by heavy lugs 15, which are insulated from the bell by an insulating sleeve 16.

To prevent the current from flowing from the cathode to the bell, a portion of the electrolyte 17, is solidified by means of the cooling pipe 18, through which water is circulated. The pipe 19 for removing the metal produced is insulated from the other parts of the cathode compartment. The melted lead cathode 27 is circulated by means of the pistons or floats 22 which are raised and lowered alternately by means of the cranks 23. (To reduce the cost of construction, the pistons in the experimental furnace were operated by a walking beam instead of by cranks.)

As already noted, the operation of the furnace is analogous to the Castner caustic cell. The fused sodium chloride is decomposed by the current yielding chlorine at the anode and lead-sodium alloy at the cathode. The circulation of the alloy carries the sodium to the cathode compartment where it is dissolved out of the alloy by the ions liberated at the surface of the alloy which here acts as anode, and the sodium is liberated at the cathode. The electrolyte in the cathode compartment is not decomposed at all ultimately, simply serving as a medium for the current to transport the sodium from the anode to the cathode. The circulation of the lead is facilitated by the raised bottom of the pan. The constriction of the path of flow increases the velocity of flow beneath the anode compartment and thus gives a positive transfer of the alloy from one compartment to the other.

In the furnace as constructed the pan 2 was 5 ft. long by 3 ft. wide. The anode compartment 28 in. x 13 in. x 9 in. high (internal). The cathode compartment 28 in. x 8 in. x 4 in. high (internal). Pistons 26 in. x 7 in. x 4 in. (external). Stroke 2½ in. Displacement, 450 cu. in. per stroke. The anodes consisted of 5¼ in. diameter by 4 in. long heads screwed onto 3 in. diameter rods, which in turn were screwed into a heavy cast iron head to which the cables were attached. There being three such heads, the total anode surface parallel to the cathode was about 420 sq. cm. (65 sq. in.). The cathode surface was about 2350 sq. cm (364 sq. in.). (This ratio is too large. In practice the cathode surface should not be any larger than the anode surface than can be helped.)

Mr. Acker states 295 amp. per sq. dm. as the current density at the anode in his furnace. The density in this furnace will be 190 amp per sq. dm. using 800 amp and and 235 amp per sq. dm., using 1000 amp. There is a maximum limit to the current density on graphite anodes in a fused bath of sodium chloride. In one experiment using 800 amp. on two of the above heads, a current density of about 300 amp. per sq. dm., it was found that the graphite was very rapidly corroded.

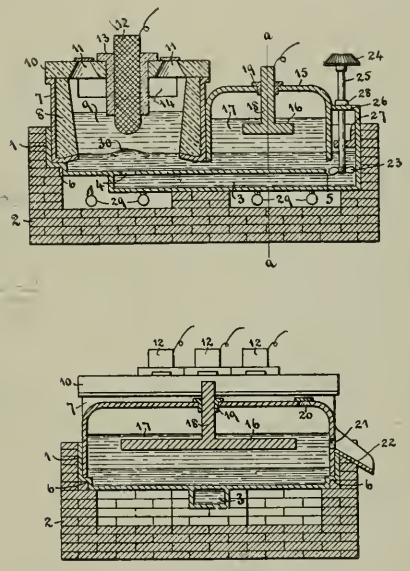
The gas burners were made on the principle of the Bunsen burner, on a greatly magnified scale. The gas supply tube was ¾ in. diameter and the outer tube 3 in. diameter by 24 in. long. The air was supplied by a fan. With natural gas, the whole furnace could readily be heated to a full red heat.

The piston type of furnace was only operated for a short time and not being successful was soon abandoned for the following reasons: (a) Excessive radiation loss: It will readily be seen from the illustration and description that the superficial area covered was comparatively large. The gas burners were only intended to heat up the furnace on starting, the heat being afterward maintained by the electric current, but it will be noted that while the area of the anode compartment, where the greater part of the heat is generated, is only 364 sq. in., the total area of the furnace is about 15 sq. ft. This loss alone was sufficient to kill the commercial possibilities of the furnace, but even greater difficulties developed.

(b): The piston circulation itself was a total failure. It being impossible to make a tight joint between the piston and the sides of the pan, a narrow rim was left which exposed a strip of the surface of the red hot lead. The melting point of litharge is above the temperature of the lead in the furnace and therefore it solidified in the space between the pistons and the side. The solid crust was lifted at each elevation of the piston, exposing a fresh surface of lead, until finally so much litharge had accumulated that the apparatus was brought to a standstill. Reciprocal circulation is also not recommended, as the reversal of the flow makes it impossible to obtain the desired velocity of flow under the anode compartment. It requires a strong, positive flow to maintain a clean metal surface at the cathode and this is more readily obtained by means of the improved furnace described below.

Continuous-Flow Furnace.

To overcome the difficulties mentioned above, an entirely new type of furnace was constructed. The piston circula-



FIGS. 4 AND 5.—CONTINUOUS-FLOW FURNACE.

tion was abandoned, thus rendering it possible to make the furnace more compact, with a corresponding reduction in the radiation loss. Continuous circulation was substituted for the reciprocating movement in the piston type. The general features will be readily understood from Fig. No. 4 and No. 5, from U. S. P. No. 859,431 of 7-9-07.

Fig. No. 4 is a longitudinal, vertical section and Fig. No. 5 is a transverse, vertical section on the line *aa* in Fig. No. 4. The cast-iron pan 1 is set in masonry 2. On the under side of the pan, and cast with it, is a longitudinal compartment 3. communicating with the pan through the circular openings 4 and 5. This forms a return passageway for the fused lead. The opening 4 is arranged axially with one of the anode heads and each anode head has a corresponding opening from the lead conduit. The remaining portions of the apparatus are supported within the pan by means of the lugs 6. The anode and cathode compartments are constructed in the same manner as described under the piston type.

At the end of the apparatus, beyond the cathode compart-

ment, is located the circulating device. This consists of the screw propeller 23, which is rotated in the opening 5 leading into the compartment or passage 3, by means of any suitable power-transmitting device, such as the gear 24. The same gas burners were used as in the piston furnace. The general action is the same as already described for the piston type.

Some small changes were made in the construction which do not exactly correspond with the figures given in this article. Two anode heads were used instead of three, the dimensions being the same as already given. The charging opening was in the front instead of in the top. The anode surface parallel to the cathode was 280 sq. cm. (43 sq. in.) and the cathode area about 1560 sq. cm. (242 sq. in.). On the basis of 800 amp. for the piston furnace, the new furnace was rated at 500 to 550 amp., but was found afterward to work better at a higher density, 600 to 700 amp. being usually used.

About 1400 lb. of lead were required to fill the furnace up to the seals under the edges of the compartments. The circulation was efficiently accomplished by means of a cast-iron propeller 4 in. in diameter, revolving 600 per minute. While the desired vigorous, positive circulation was readily obtained, this arrangement is not recommended for practical operation owing to the frail nature of the propeller. A tangential pump would work equally well and could be made sufficiently massive to withstand the rough usage involved in pumping red hot lead.

The great advantage of this type over the one abandoned rests in the circulation of the melted lead cathode. The strong current of fluid metal rises through the openings beneath the anode, forming a sort of mound on the surface, from which the flow radiates in all directions.

In this manner the surface of the cathode is kept perfectly clean, which is quite essential to the success of the operation. If this is not done, a difficultly fusible layer will coat over the cathode and abnormally raise the resistance of the electrolyte. This layer contains graphite dust torn away from the anode and the difficultly fusible impurities in the salt, as sulfate of lime. In time this layer, which is in a pasty condition, will stop the action of the furnace.

If, however, the circulation is accomplished as described above, the strong radial currents will carry the impurities to the sides of the compartment, where they will accumulate in a thick layer, which may be removed from time to time. In experiments with the piston furnace, where the circulation was weak, the operation was several times brought to a halt by this accumulation spreading over the entire cathode surface.

In starting, the furnace was first heated to a full red heat by means of the natural gas burners. This heat is not sufficient to melt the salt which is accomplished as described by Acker, *Trans.* Am. Electrochem. Soc., v. 1, p. 169 (1902). This consists in passing a strong current through a resistor until sufficient melted salt has been formed so that the electrolyte can carry the current. The resistors were ¾ in. or 1 in. diameter carbons, about 3 in. long, driven into the face of the anode blocks.

The resistors were heated to a bright red heat and the salt piled around them. Two resistors were usually used, one on each anode head. The 1 in. x 3 in. rods would carry 400 to 500 amp. at 4 to 5 volts and the ¾ in. x 3 in. rods, 300 to 400 amp. at 5 to 6 volts. After a time a pool of melted salt will form.

The resistors may then be broken off and the current started through the electrolyte. It is essential that the graphite anode be red hot before the resistors are broken off, for the melted salt will not "wet" the cool graphite surfaces; or in other words a thin layer of solid salt forms on the cold graphite and being a non conductor no contact can be obtained.

The preparation of the original bath, especially in experimental work, is facilitated by using a mixed electrolyte instead of the pure salt. A mixture of 3 molecules sodium chloride, 3 molecules potassium chloride and 2 molecules of calcium chloride melts readily at a red heat and will yield almost pure sodium on electrolysis, a small percentage of potassium being liberated with the sodium, but no calcium. After the current has once been started through the electrolyte, only salt need be added for this purpose only the very purest commercial salt obtainable is to be used, as the impurities form the infusible material mentioned above, which has to be removed. Mined rock salt is usually the best.

The furnace as described above may be used in the preparation of free metallic sodium or in the preparation of sodium compounds which ordinarily involve the use of metallic sodium at high temperatures. If sodium is desired, the electrolyte in the cathode compartment must be of such nature that it will not be decomposed by sodium at a full red heat.

As the present commercial process for obtaining sodium is based on the liberation of the metal in a fused bath of caustic soda, this compound was naturally the first one selected as the electrolyte for the cathode compartment. It was then a matter of considerable surprise when it was found impossible to obtain any trace whatsoever of sodium when the attempt was made. It was first thought that the fused caustic still held some last traces of moisture even at a red heat, but in this case there should have been at least a liberation of hydrogen. The electrolyte simply laid dead in the furnace and not a bubble or a globule of metal was formed to show that 400 to 600 amperes were being passed through.

The writer is not prepared to state positively that it is impossible to liberate sodium from fused caustic soda at 700 to 900° when the anode is soluble and contains sodium, but the indications from these experiments are that such is the case. The electrolyte was not wholly protected from the action of the air and it is possible that sodium reacted with caustic at this temperature to form a suboxidized compound, which took up oxygen from the air. However, it was thought best to avoid the presence of oxygen in the electrolyte.

As pure sodium chloride is so difficult to melt, the cathode compartment was filled with the same mixed chloride bath described above for use in starting the fusion in the anode compartment. As soon as the lead cathode had had an opportunity to take up sufficient sodium to supply the sodium required for the soluble anode, the current was started through the cathode compartment and regulated by the water rheostat so that only 75 per cent of the total would be acting in that comparment. As soon as the current is started the globules of metallic sodium begin to gather on the surface of the electrolyte.

The conditions under which the experiments were carried out were such that it was impossible to continue a single continuous experiment for a sufficiently long period to gain any reliable data on the ampere efficiency and for the same reason no fully satisfactory method of removing the red hot sodium was devised. It might be safe to assume that the ampere hour efficiency would equal the 93 per cent claimed for the Acker furnace (*Trans.* Am. Electrochem. Soc., v. 1, p. 171, 1902), but as some sodium is lost in removal from the furnace, 80 per cent was assumed as the basis of commercial calculations. The voltage was 6 to 8 volts using 500 to 700 amp. The experiments were finally stopped for financial reasons.

A few experiments were also made on the preparation of sodium compounds instead of the metal. Two important compounds requiring the free metal in their preparation are sodium peroxide and sodium cyanide. It was hoped that by liberating the sodium in a fused bath of caustic in the presence of air at a bright red heat that both the caustic and the sodium produced would be converted into a mixture of sodium oxide and sodium peroxide which by further heating in air could be entirely converted into the peroxide. Sodium nitrate was also used as an oxidizing agent in some of the experiments.

No conclusive results were obtained but it was found that the percentage of sodium oxide in the bath increased as the experiment progressed and the product would liberate oxygen

when placed in water. This product was so impure, however, that it was feared that it would not be able to compete with the high-grade material made from the previously prepared sodium.

It was impossible for reasons stated to carry the experiments further but it had been demonstrated that it was possible to produce sodium compounds by electrolysis in a fused bath with a soluble anode containing sodium, the bath being capable of uniting with sodium to form the desired compound. It is to be noted that these compounds cannot be prepared from lead-sodium alloy and reacting upon this with similar reagents, the method described being equivalent to the use of free sodium while the lead-sodium alloy is not.

This is due to the fact that the formation of the alloy is strongly exothermic and therefore a corresponding amount of energy is required to liberate the sodium from the lead before it can react. In brief, it is possible by this method to produce compounds of sodium ordinarily requiring the use of previously prepared sodium, without the expensive operation of obtaining the metal in ingot form.

In conclusion, it may be said that while these experiments left the problem in a partly developed condition, they fully demonstrated the possibility of preparing both metallic sodium and sodium derivatives along the lines described. With the experience gained, it would be possible to modify the apparatus as suggested above and complete the demonstration of the commercial feasibility of the processes, but no further funds being available the work had to be abandoned.

Temperature Coefficients of Resistance of Resistors.

Since nickel and nickel-chrome (the latter, for instance, under the trade name nichrome) are often used as resistors for electric heating or electric furnace work, the measurements of their temperature coefficients of resistance, as given by A. A. Somerville in a recent American Physical Society paper, should be of interest. (See *Physical Review*, February, page 268.)

Four materials were studied—nickel and nichrome, two good furnace wires, inasmuch as they do not oxidize readily; and tungsten and molybdenum, which must be protected from oxygen when heated. The nickel and nichrome curves are very similar in shape, but the coefficient of the former is 20 times that of the latter. The coefficients in either case are relatively small between 400° C. and 1000° C. as compared with the coefficients outside these limits. (Unfortunately, the exact numerical figures are not given.) The tungsten and molybdenum curves are slightly convex toward the temperature axis. The coefficients are large—1 ohm of molybdenum at 0° C. measuring 5.7 ohms at 1000° C. and 1 ohm of tungsten at 0° C. measuring 6.9 ohms at 1000° C.

Silica Standards of Length.

Silica standards of length are quite a novelty. As mentioned in the recent annual report of the (Brit.) National Physical Laboratory, careful experiments have demonstrated that silica is superior to glass as to thermal hysteresis, the coefficients per degree Centigrade being: Silica, from 0.5 to 1; Jena glass, 59^{III}, 2.7; Jena glass, 16^{III}, 3.5, all expressed in 10^{-8} units. The 1-m silica standard of the National Physical Laboratory is a tube supplied by the Thermal Syndicate, 2 cm in diameter, having half disks fused into its ends.

The difficulty of getting good defining lines on the silica has been overcome. Ordinary scratches chipped off in a day or two, and could not be seen when the tube was immersed in water. The ends have, therefore, been platinized, but on the lower surface; the graduation is cut through the platinum film into the silica, and the protecting glass cover rests against the lower platinized surface. The reading from above is, therefore, not affected by any tilt in the cover glass.

High-Temperature Gas Thermometry and Its Present Limitations.

By Arthur L. Day.

The measurement of temperature is different from most ordinary measurements. There is no temperature unit corresponding to a foot rule or meter stick which can be applied successively to measure an unknown temperature as we would measure the height of a room. Two temperatures of 1° cannot be combined in any way to give a temperature of 2°. Temperature measurement is, therefore, wholly a matter of arbitrary definition, of selecting some convenient phenomenon like the expansion of gas which varies continuously and uniformly (as nearly as possible) with temperature changes, of providing convenient arbitrary units of subdivision and then of observing its behavior under the same conditions which surround the unknown body whose temperature is desired. The expansion of hydrogen has been established by international agreement as our fundamental measure of temperature. The gas thermometer is, therefore, now the standard thermometer in terms of which all temperatures are expressed.

The theoretical interest in gas thermometry centers about the quantitative relation existing between the increase in the temperature of the gas at constant pressure or volume and the quantity of heat required to produce it. It is a somewhat inaccessible question by reason of the difficulty of approaching it experimentally with the required accuracy. Although the amount of the expansion of gases is more closely proportional to the quantity of heat applied than that of other substances, no strictly "perfect gas," in this sense, has been found.

The amount of the divergence in the case of nitrogen, which has a greater range of practical utility than other gases thus far studied, is of the order of magnitude of a single degree of 1100° C., and the uncertainty in this correction factor is probably smaller than the observation error of a practical gas thermometer in the present stage of its development. For this reason these theoretical considerations, which have been admirably treated by Buckingham in a paper, "On the Establishment of the Thermodynamic Scale of Temperature by Means of the Constant Pressure Thermometer" (*Bulletin* Bureau of Standards, 3, 237, 1907), do not seriously affect the definition of an accurate and practical high-temperature scale by means of the gas law.

On the other hand, there is no denying the fact that a great deal still remains to be done upon the experimental side before the steadily advancing requirements of both science and industry in the matter of a trustworthy temperature scale of sufficient accuracy, and even more particularly, of sufficient range, can be satisfied. It is no disparagement of the present system of temperature definition to say that the gas thermometer itself is a complicated and cumbersome instrument to use in any of the forms which have hitherto been devised, and possesses limitations both of range and of accuracy which are difficult to overcome.

One consequence of this, particularly in the region of high-temperature measurements, is that temperatures easily come to be regarded with unwarranted confidence in the hands of those who have never acquired a first-hand knowledge of these limiting conditions. This has been further facilitated by the comparative ease with which relative measurements of temperature can be made, even in the more inaccessible parts of the scale, with the thermoelement and the resistance thermometer. These devices are sensitive to temperature differences of the order of magnitude of 0.01° throughout their entire range, but they are dependent absolutely upon fundamental measurements with the gas thermometer for their evaluation in terms of the generally accepted degree Centigrade.

It is sufficiently obvious, though often carelessly overlooked, that no method of temperature measurement, however sensitive or adaptable it may be, can yield temperatures of greater abso-

lute accuracy than the system in terms of which those temperatures are defined. With the gas thermometer as our basis of definition, therefore, we shall know our temperatures with just the certainty which it is able to furnish and no more. There is, to be sure, some justification for expressing the results of thermoelectric or resistance measurements in units smaller than the errors of the fundamental scale, where only comparative measurement is involved, but such measurements must not be allowed to escape from this limitation.

It is not the purpose of the present article to discuss the general problem of thermometry or the particular advantages of one system of thermometric measurement over another in laboratory or industrial practice, but merely to speak very briefly of some of the work which has been done in recent years to increase the range and accuracy of the temperature scale upon which the various devices for measuring high temperatures depend for their calibration. Neither is it necessary in so brief a review to include any discussion of the relative theoretical merit or experimental practicability of the constant-volume and constant-pressure methods of gas thermometry. The reader will find it in Buckingham's paper already mentioned, as well as in earlier discussions to which he refers. The choice of the one or the other method is likely to be governed by the taste or predilection rather than by the necessities of the individual experimenter. Neither system possesses decided advantages over the other. All the recent measurements (including all those referred to in this paper) have been made with the constant-volume system.

By international agreement and nearly universal practice, two fixed temperatures are accepted as the basis of the modern temperature scale, the melting point of pure ice and the boiling point of pure water, both at normal atmospheric pressure (760 mm of mercury). The interval included between these two temperatures is subdivided into 100 parts by measuring the constant volume expansion of hydrogen from an initial pressure of 1 m of mercury throughout this interval.

The actual determination of these intervals or degrees Centigrade was made by Chappuis at the Bureau International du Poids et Mesures at Paris in 1888, and is a work of such extraordinary painstaking character in most particulars that no investigation has found it necessary to repeat it since that time. The probable accuracy of the individual degrees thus determined, i.e., of the international scale of temperatures between 0 and 100°, is stated by Chappuis to be 0.002°.[1]

To extend the scale beyond these limits in either direction it is (theoretically) only necessary to continue the measurement of the expansion of hydrogen, under the same conditions, down to or up to any desired temperature. In practice, this procedure encounters various experimental difficulties which increase rather rapidly as we get further away from the temperatures of everyday life.

These experimental difficulties serve to place definite limits upon the accuracy of temperature measurements at present attainable in different parts of the field, of which it is important to obtain a clear idea in order to be able to form a sound judgment of the probable significance of measurements at widely different temperatures. If we are told, for example, that a solution boils at 90.27°, we recognize that this accuracy is entirely practicable. If, on the other hand, platinum is said to melt at 1755.46°, it is of advantage to know that the last two figures are wholly without significance and the fourth is uncertain.

There is another reason for being especially explicit upon this point, for very few men in this country have ever made the laboratory acquaintance of the gas thermometer.[2] In its modern form the constant-volume instrument includes five principal features which will be briefly considered with particular reference to the limitations which they severally impose upon the accuracy of high temperature measurements: The gas (1) is contained in an impervious vessel or bulb (2) which is heated in an electric furnace (3) arranged to provide the utmost attainable uniformity of temperature about the bulb at all temperatures. A mercury manometer (4) measures the gas pressure within the bulb, and thermoelements (5) are used to transfer the gas thermometer temperature to standard melting points for general application. These will be briefly treated in order:

(1) The Gas.

The first limitation to the upward extension of the gas scale is a direct result of the choice of a gas. Hydrogen was chosen because it approximated more closely to a perfect gas in its behavior than any other, but the experimental development of the subject has since demonstrated that no vessel can be depended upon to hold it without loss above 300°, so that nitrogen has come to be very generally substituted for hydrogen at the higher temperatures.

With reference to the situation created by this substitution, a long series of comparative measurements by Jaquerod and Perrot,[3] using air, nitrogen, oxygen, carbon monoxide, and carbon dioxide, shows no measurable differences in the behavior of these gases for thermometric purposes up to the melting temperature of pure gold (1062°), but some of these certainly could not be carried far beyond this temperature without danger of serious error through dissociation.

Nitrogen has been used, both in the Reichsanstalt and in the Geophysical Laboratory, as high as 1600° without any appearance of irregularity in its expansion or evidence of dissociation. The extension of the hydrogen scale by the use of nitrogen above 300°therefore encounters no serious difficulty up to the present limit of measurement (1600°) except in so far as it may prove necessary to make a small correction (of the order of magnitude of 1° at 1000°) for reference to the true thermodynamic scale if it is desirable to express the results of measurement in terms of it.

(2) The Bulb.

The bulb is perhaps the most important single element in a constant-volume gas thermometer. The progress of gas thermometry during nearly 100 years has been closely associated with and dependent upon improvements in the bulb. The first bulbs were made of metal (Prinsep used a gold bulb in 1827), and there is little question to-day but that their abandonment for porcelain in 1857 (Deville and Troost) constituted a step backward which was not retrieved for 40 years. The bulbs in use at the Reichsanstalt up to 1897 were still of porcelain, as were those used by Barus throughout his epoch-making work.[4]

Porcelain, glazed or unglazed, violates both essential conditions for a good gas thermometer bulb at high temperatures: (1) It cannot hold the expanding gas without absorbing or losing any of it; and (2) it is not absolutely uniform in its own expansion and contraction. Bulbs of platinum containing 10 to 20 per cent of iridium for additional stiffness were used at the Reichsanstalt in 1900 for temperatures as high as 1200°, and one of pure iridium was successfully carried to 1600° in 1907. These bulbs are still in use.

The Geophysical Laboratory began its work with a 10 per cent platin-iridium alloy, but abandoned it for platinum con-

[1] In a recent paper, "Some New Measurements with the Gas Thermometer," *Amer. Jour. Sci.* (4) 26, 405-463, 1908, the authors gave it as their opinion that the determination of the expansion coefficient of the thermometer bulb used by Chappuis, which is, of course, an important factor in the experimental problem, would not warrant an assumption of accuracy greater than 0.005°. To this Chappuis replied in a personal letter (1909) that, considering all the factors of the problem and subsequent experience, he still thought 0.002° a fairer estimate of the probable error of the scale in the region between 0 and 100°.

[2] An incident may serve to illustrate the fact that the gas thermometer is but a name to a great many otherwise well-informed scientific men. Soon after the appearance of one of the writer's earlier papers upon the subject, a well-known engineer wrote to inquire the cost of a dozen gas thermometers, to which I was under the necessity of replying that I knew of but one in this country, and this one had cost us upwards of four thousand dollars to date and might cost even more before we had finished with it.

[3] Jaquerod and Perrot, *Arch. d. sci. phys. et nat.* Genève (4), 20, pp. 28, 128, 454, 506, 1905.

[4] Carl Barus, "On the Thermoelectric Measurement of High Temperatures," *Bull.* 54 (1889), U. S. Geol. Survey.

taining 20 per cent of rhodium, for reasons which will appear in connection with the use of thermoelements below. All these metal bulbs can be used with nitrogen up to 1600° without appreciable error in the return to 0° after heating, provided only that proper precautions are taken against mechanical strain and consequent volume change at the higher temperatures where the metal is soft.

Porcelain bulbs when unglazed are not gas tight; glazed within, they can be used up to 1100° without serious error, but the glaze becomes soft above this temperature and reacts to some extent with the gas in the bulb which then never returns to its initial pressure at 0°. Porcelain bulbs with outside glaze only were employed by Barus up to 1400° with fair success, but it proved impossible to permanently saturate the bulb wall with the gas, so that it would return to the same zero after heating.

Pure silica bulbs which have recently been introduced by Jaquerod and Perrot (loc. cit., 1905) have an advantage over the metals through their low expansion coefficient, but cannot be used above 1100° without permanent volume changes.

In view of the other improvements which have been made in recent years (notably, the introduction of electric furnaces), it is difficult to estimate the proportion of error in the earlier measurements for which the porcelain bulbs were responsible. Barus' value for the melting point of gold (1888—porcelain bulbs) was 1092°, Violle's (1889—porcelain bulbs), 1035°; Holborn and Wien's (Reichsanstalt, 1895—also porcelain bulbs), 1027°; Holborn and Day's (Reichsanstalt, 1900—platin-iridium bulb), 1063.5°; Day and Sosman's (Geophysical Laboratory, 1910—platin-rhodium bulb), 1062.4°.

(3) The Furnace.

Improvements in the furnace have twice contributed materially to the advancement of gas thermometry: (1) Through the substitution of electricity for gas as a source of heat (Reichsanstalt, 1899), which greatly reduced the uncertainty in temperature distribution over the bulb surface and eliminated the possibility of furnace gases passing into the thermometer itself through the bulb wall; and (2) when the bulb and heating coils were enclosed in an air-tight bomb (Geophysical Laboratory, 1908) under conditions such that the air pressure outside of the bulb could be kept under control and always maintained equal to the pressure of the expanding gas within.

This second improvement made it possible to relieve the bulb from mechanical strain at high temperatures where the metal is soft and susceptible to volume changes, and considerably increased the effective range of the instrument in the high temperature region. It also became possible to use different initial pressures within the thermometer bulb in order to evaluate the pressure coefficient of the expanding gas and to vary the sensitiveness of the instrument.

Before this improvement was made it was necessary to select an initial gas pressure such that the final pressure at the highest temperature to which it was expected to attain would be approximately equal to the atmospheric pressure without, in order that the difference in pressure within and without the bulb should be a minimum when its power to withstand such a difference was smallest. The practical effect of the earlier arrangement was to crowd the entire range of the instrument into ½ m of the manometer scale, giving about ¼ mm per degree. With the air-tight furnace bomb we were able to increase this sensitiveness to a full millimeter for each degree.

To the introduction of electric heating we owe the extension of the range of the gas thermometer from 1150° to 1500°; to the control of the gas pressure about the bulb, a considerable (perhaps five-fold) increase in its accuracy in this region.

With electric heating, the furnace temperature is always under control and readily measurable, but in pursuing high standards of accuracy we must distinguish between a constant furnace temperature of irregular distribution and a variable temperature. The substitution of electricity for gas has given us a constant temperature, but it is not always easy to provide that it shall also be perfectly uniform.

Various baths are available up to perhaps 700° for the purpose of equalizing the temperature about the bulb by interposing between it and the heating coil a liquid layer which can be stirred, but higher temperatures have had to depend upon direct exposure to the radiation from the coil. In the Reichsanstalt instrument, temperature differences of from 10° to 30° were observed between the middle and ends of the bulb in the region 1500°-1600°; in that of the Geophysical Laboratory, the maximum differences were diminished to 2° in the same region.

For greater uniformity than this some form of liquid bath will require to be devised for use at these temperatures also. Such a search might prove worth while, for this uncertainty (in the distribution of temperature about the bulb) appears to be the largest remaining in gas thermometry up to 1550°.

(4) The Manometer.

The manometer contains few factors which affect the ultimate temperature readings materially. The arrangement commonly adopted, both in Europe and in this country, is the open U-tube type in which the difference in the mercury level between the two arms of the U-tube represents the difference between the gas pressure within the bulb and the prevailing atmospheric pressure outside, the latter being read on a separate barometer. The minor corrections for temperature of the mercury, gravity, etc., then require to be applied as usual in such cases.

The only real difficulty encountered here has been much reduced in recent years and is no longer dangerous, even at the highest temperatures. The temperature of the capillary connection between the bulb and manometer varies all the way from the extreme temperature of the bulb to that of the room and with it the temperature of so much of the gas as it contains. The error from this cause is reduced to an insignificant magnitude by making this connecting volume so small when compared with that of the bulb that its average temperature need only be known within about 10 per cent.

Some laboratory experience is necessary to make this reduction of volume judiciously, for the viscosity of gases increases rapidly with the temperature, so that it is not practicable to restrict below 0.8 mm the diameter of those portions of the capillary which are heated. A smaller diameter here would have the effect of producing a very slow readjustment of pressure in the various parts of the apparatus with every change in the temperature, and the observer would be without adequate assurance that equilibrium had been reached at the time when his reading was made.

The uncertainty which attaches to this connecting volume, together with the changes in the zero point due to the irregular behavior of porcelain bulbs, were among the most formidable of the classic errors of gas thermometry. Both are now practically eliminated.

(5) The Auxiliary Thermoelement.

The fact that the gas thermometer bulb cannot be safely immersed in different substances for the determination of their thermal constant carries with it a considerable amount of incidental but practically unavoidable inconvenience. The usual plan has been (and still is) to place thermoelements in the furnace with the bulb and to determine their electromotive force for a measured temperature. A few observations of this kind suffice to establish an empirical curve for the electromotive force of the thermoelement in terms of the gas thermometer temperatures, after which the thermoelement is available for determinations of temperature under a great variety of conditions.

This procedure is considerably facilitated by the fact that the empirical curve corresponds closely to a simple parabola and three or four comparisons with the gas thermometer usually suffice to establish a parabolic curve which may be interpolated with little danger of an error greater than the errors of the

gas scale itself over long ranges of temperature. Its extrapolation has not turned out so fortunately.

Before the publication of the recent measurements in the Geophysical Laboratory, the region of accurate gas thermometry ended with the temperature 1150°. The common practice for some years has therefore been to measure temperatures beyond that point by extrapolating the parabolic curve out to the desired temperature—often as high as the melting point of platinum (1755°). Extended extrapolation is always a matter of grave uncertainty, but no other convenient method was available, and after all, observations of this kind with the thermoelement could be translated in terms of an observed scale whenever one should be provided. It happens in this particular case that the extrapolation (from 1150° to 1755°) comes out just 50° low at the platinum melting point.

Now that the gas thermometer is able to make direct measurements of temperature up to 1550°, it is merely necessary to include several thermoelements in the furnace with it, to evaluate their temperature curves by direct comparison, and afterward to determine with them a series of standard fixed temperatures like the melting points of metals, minerals, or salts, which are available for general use.

This procedure has been followed in the recent investigation with the results which are tabulated below. The pure metals are readily obtainable and the minerals easily made up, so that anyone may now calibrate his thermoelement by noting its electromotive force at the (known) melting temperature of three or four of these substances, preferably choosing such as will include the region in which he expects to use the thermoelement.

Such a calibration requires to be checked from time to time by repeating one or more of the melting point observations, particularly if the element is exposed to contamination for vapors of other metals. This contamination is particularly disastrous if a thermoelement is exposed to the vapor of iridium, which explains the remark made above, that in our experience a bulb made from an alloy of platinum and rhodium is to be preferred to the iridium alloy in use at the Reinsanstalt.

The standard melting points in the table below are taken from observations by Day and Sosman[5] and include, for comparison, the standard melting points published by the Reichsanstalt in 1900 and now in general use:

Substance	Point	Atmosphere	Crucible	Temperature	The Reichsanstalt Scale
Zinc	Melting and freezing	Air	Graphite	418.2° ±0.3	419.0°
Antimony	Do.	Carbon monoxide	Do.	629.2 ±0.5	630.6
Silver	Do.	Do.	Do.	960.0 ±0.7	961.5
Gold	Do.	Do.	Do.	1062.4 ±0.8	1064.0
Copper	Do.	Do.	Do.	1082.6 ±0.8	1084.1
Diopside (pure)	Melting	Air	Platinum	1391.2 ±1.5	
Nickel	Melting and freezing	Hydrogen and Nitrogen	Magnesia and Magnesium Aluminate	1452.3 ±2.0	
Cobalt	Do.	Do.	Magnesia	1489.8 ±2.0	
Palladium	Do.	Air	Pure Magnesia	1549.2 ±2.0	1575.2
Anorthite (pure)	Melting	Do.	Platinum	1549.5 ±2.0	

In addition, the following temperatures were incidentally obtained:

Cadmium	Melting and freezing	Air	Graphite	320.0 ±0.3	321.7
Aluminum	Freezing	Carbon monoxide	Do.	658.0 ±0.6	657.
Platinum	Melting	Air		1755.	

A new estimate of the melting point of platinum, which has not yet been directly determined with the gas thermometer, is included in the above list. It was obtained in this way: There is a remarkably close agreement between independent determinations of the *temperature interval* between the melting points of palladium and platinum:

Nernst and von Wartenberg[6] ..... 204°
Holborn and Valentiner (at the Reichsanstalt[7]) ..... 207°
Waidner and Burgess (at the Bureau of Standards)[8] .. 207°

If we, therefore, simply add 206° to the palladium melting temperature (1549°), we obtain 1755° as the melting point of pure platinum, with an absolute error of perhaps no more than ± 5°.

By way of conclusion, the following estimate is offered of the accuracy of existing standards of temperature within certain temperature intervals:

0° — 100° ± 0.002°
100° — 300° ± 0.05°
300° — 1100° ± 0.8°
1100° — 1550° ± 2°
1550° — 1750° ± 5°

Geophysical Laboratory,
Carnegie Institution of Washington,
Washington, D. C.

Sintering of Fine Ore and a Sintering Machine.*

By Alfales B. Young.

Among the metallurgical developments of the last decade one of the most important has been the sintering treatment of fine ore, which beginning as a step in the reduction of lead, with its chief object the securing of a dead roast, has come to be a prominent factor in copper smelting, where its object other than that of obtaining the roast, is also the preparation of the product in a sintered or agglomerated form suitable for blast-furnace work. The first to bring forth the idea of a blast roast were Huntington and Heberlein, whose names have been so linked with sintering in general that a great deal of it is known under the heading of the "Huntington-Heberlein process," although it does not, strictly speaking, belong to their invention.

In a general way, the process of sintering may be classified in two divisions, viz., those requiring a preliminary roast and those in which the raw ore is sintered directly. The former belong to the Huntington-Heberlein process, and in the latter are the Carmichael-Bradford, the Savelsberg, and the Bormette processes.

The above is only a partial list, but I think it contains the more important methods.

The characteristic feature of the Huntington-Heberlein process is the addition of limestone to a partially roasted galena ore. The fine galena concentrates are mixed with limestone and slag-making material, and this is given a preliminary roast. The first roast eliminates sulphur to the extent of 6 per cent to 8 per cent. The Godfrey rotating-hearth roaster was formerly preferred for this part of the operation, but any of the modern types of roasting furnaces will serve the purpose.

The Huntington-Heberlein sintering pot or converter is shaped like an inverted dome having a diameter of 8 ft. or 9 ft. About 1 ft. from the bottom a grate, convex upward, is fitted in the portion of the pot under the grate, having an opening or tuyeres for attaching an air pipe and thus constituting a windbox. Over the pot is a hood, attached to a flue or stack, which may be raised out of the way. Usually there is a row of pots over which runs a traveling crane for charging and dumping them.

Prior to charging, a thin layer of coal ashes is spread over the grate. On this is dumped a layer of hot calcines from the roaster, and on this, in turn, the regular charges of calcines, moistened with water. The time of sintering varies from 8 to 14 hours, and a product with about 2 per cent sulphur is ob-

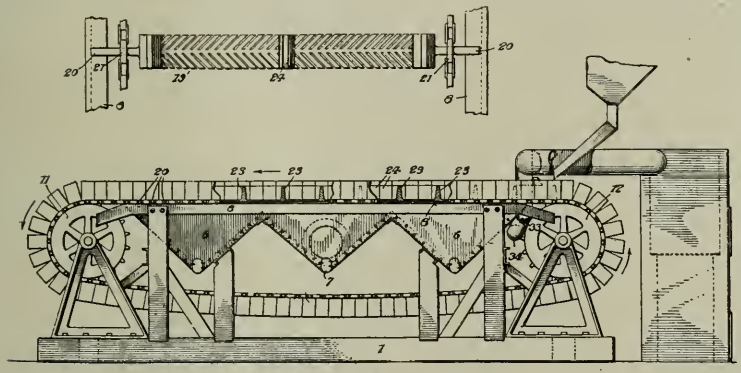
[6] W. Nernst and H. von Wartenberg, Ber. d. Deutsch. phys. Ges., iv, pp. 48, 146, 1906.
[7] L. Holborn and S. Valentiner, Ann. d. Phys. (4), xxii, 1, 1907.
[8] C. W. Waidner and G. K. Burgess, Bull. Bur. Standards, iii, p. 163, 1907.

* A paper read at the Fifth General Meeting of the Western Association of Technical Chemists and Metallurgists and published in the *Western Chemist and Metallurgist*, March, 1910.

tained. The addition of limestone seems to aid the roasting and at the same time prevents a premature fritting before the roast takes place. This is probably accomplished both by a mechanical action of the particles and by absorption of heat in the breaking down of the calcium carbonate molecule into lime and carbon dioxide. The sintering material is dumped from the pot by the crane, where it is sprayed with water and broken up by sledging.

The Carmichael-Bradford process employs dehydrated gypsum instead of limestone. The process is carried on in the same form of pot as the Huntington-Heberlein. The charge at Broken Hill, where this method has met with success, consists, roughly, of one part dehydrated gypsum, three parts of slimes and one part of sand concentrates. The sulphur content of the charge is about 16 per cent and of the product about three. The material for the charge, mixed and moistened in a pug mill, is broken into nut-sized lumps while still wet and spread out and allowed to dry in the sun. The charging of these small lumps enables an extremely free passage of air through the mass, giving a good sinter and lessening the time of the operation.

The Savelsberg process is like the Huntington-Heberlein in that the flux used is calcium carbonate, but differs in the fact that a preliminary roasting is unnecessary. By the addition of sufficient limestone the heat from the oxidation of the contents



APPARATUS FOR SINTERING FINE ORES.

of the charge is absorbed by the breaking up of the carbonate. The sintering pot is of the same general shape as that previously described. However, instead of being manipulated by a traveling crane it is mounted upon a track, much after the fashion of the ordinary slag pot.

The Bormette method of lead and copper smelting consists of the incorporation of the fines in a predetermined amount of liquid slag, accompanied by rapid stirring in the pot. A porous, spongy mass is evolved with about 50 per cent desulphurization. These sintering processes, briefly described, have been gradually taken over into copper work, where mine fines, matte and flue dust have been agglomerated with varying degrees of success. The addition of lime and other kindred substances has practically been eliminated, as the prevention of premature sintering is not required. It has been my experience that the addition of quartz or lime, as the case may be, in such quantities as to give a monosilicate in the slag part of the sintered mixture, gives the best results both in time of sintering and in percentage of charge sintered. With a copper ore running SiO_2 14 per cent, S 27 per cent, Fe 45 per cent, Al_2O_3 4 per cent and lime and magnesia 4 per cent, using a size ½ in. or smaller, I have had success in mixing flue dust to the extent of almost one-third by volume, securing in a Huntington-Heberlein pot a sinter of 85 per cent to 90 per cent of the total charge to 10 per cent to 12 per cent sulphur in seven or eight hours.

The unsintered portion is almost invariably a layer of from 4 in. to 8 in. at the top of the pot. As might be expected, the presence of fairly coarse pieces in the charge is an aid to the action.

The future of sintering as an economic method of roasting copper fines depends chiefly upon the cost of operation plus the cost of smelting in the blast furnace as compared with the cost of roasting in the ordinary way plus the cost of smelting in a reverberatory; and, secondly, the cost of ordinary smelting plus briquetting, as compared with the cost of sintering.

In lead work the Huntington-Heberlein process has proven itself far cheaper than the old lead reverberatory. The best we may say, however, in regard to sintering intermittently in pots is that it is crude. In the first place it is a theoretical waste of capacity for the sintered layers to be dormant in the bottom of a pot waiting for those above to be treated in the operation; secondly, it is a waste of power, having to drive air at a necessarily higher pressure through the layers below in order to reach those above to act upon them; and, thirdly, the handling of the large lumps of sinter is, at best, a clumsy and expensive operation.

The solution of the problem lies in the development of a continuous sintering machine, taking and discharging the sintered material in thin layers. To the solution of the latter part of the problem Messrs. A. S. Dwight and R. L. Lloyd have contributed the down-draft system, in which the air is sucked or blown toward the grate, thus maintaining a lack of movement among the particles and preventing the layers away from the ignition surface from remaining unsintered. This, of course, has its disadvantage in the corrosive, moist, sulphurous gases having to pass through the pipe connection and the suction fan.

To the solution of the former part of the problem I have applied merely the endless-belt idea and erected partitions on the bars of the traveling grate, dividing it into compartments, thus dividing the sinter in blocks of suitable size requiring no further breaking or crushing.

This sintering machine consists essentially of a grate in the form of an endless traveling belt which carries a bed of fines and passes for nearly its entire length over a suction or wind box, by which air is drawn through the mass. Means are provided at one end for distributing the layer of fine material over the surface of the grate and for its ignition, and at the other end the sintered product is discharged into a convenient receptacle.

Referring to the drawings, the grate consists of numerous cast-iron segments (19) carried upon the links of the chains (21) which move upon two pairs of sprocket wheels (11 and 12). The segments of the grate are provided with projecting arms (20) which rest upon rails (8) in order to prevent sagging.

Directly beneath the grate is a wind or suction box (6), to which a suction pipe is attached leading to the suction fan furnishing the draught. The sprockets are mounted upon bearings and the rails and the suction box, etc., upon supports, all of which rest upon the bed frame (1).

On the right hand of the drawing are shown the ignition apparatus and the distributor. The former consists of a small furnace or stove of the reverberatory type, from which the flames are led by a pair of flues around the distributor and into the igniter, which is merely a long sheet-metal box extending

in length the width of the grate and probably 3 in. to 6 in. wide.

This box is, of course, open at the bottom and the flame passes from it directly through the ore. Upon this igniter is mounted the plow (40). Fines from the distributor falling upon the grate are spread out evenly as they travel under this plow. Thus the igniter fits closely to the surface of the ore.

Right under the igniter, and beneath the grate, is a special suction box (33). This is connected to a special suction pipe (34), thereby giving at the point of ignition a very much stronger draught than that of the large section box (6). This insures the first-class priming—the chief essential to good sintering.

Chain.—This would be one of the standard roller chains made by transmission machinery manufacturers. The rollers would carry the chain upon the rail (8) and upon a rail on the under side (not shown) where the grate travels back.

Grate.—This grate is divided into compartments by the centrally located ribs (24) and medial ribs (23) upon every third or fourth segment. These ribs are as high as the top of the layer of ore; thus it is sintered in blocks and requires no subsequent breaking of any kind. Any form of perforation may be used for grate holes, although I have shown a fishbone arrangement, the elongated holes extending across the halves of two segments, thereby breaking in the middle when going around the sprocket and tending to dislodge any accretion which may have formed. These grate segments are so designed that replacing them is a very easy matter.

Suction or Wind Box.—(6) The peculiar shape of the bottom is for the purpose of facilitating the removal, through the handholes of any material which may have fallen through the grate.

Frame.—Although this is shown in the drawing as being of wood, it may be made of structural steel.

Operation.—Fine ore, flue dust, etc., are fed from the bin and feed hopper (diagrammatically indicated) to the machine through the distributor. The layer is spread evenly by means of the plow and passes under the igniter, where a strong flame is sucked through it. It then goes over the suction box, where air is drawn through the mass and roasting and sintering takes place. This continues until the end of the machine is reached, where the grate, turning around the sprocket, breaks away from the sinter, which discharges at this point. Here a spray of water will be constantly playing to cool the blocks.

The speed of the machine will be from 6 in. to 1 ft. per minute; the draught a vacuum of about 5 in. or more of water (3 oz. per square inch). The over-all length of a machine, of 30 tons to 40 tons daily capacity, is 18 ft. and the width of the grate 3 ft.

Features.—(1) The machine is very compact for the amount of effective grate area, occupying small floor space and requiring little head room.

(2) The separate suction for the igniter insures a much stronger draught at the point of ignition, and consequently the top layer is thoroughly sintered.

(3) By use of the compartment grate all hand breaking of the sinter is eliminated.

(4) The grate going around the sprocket and thus changing curvature at the point of discharge, breaks away from the sinter and obviates the barring and wedging necessary when a grate of constant curvature is used.

The machine can be used with the regular up-draught by making the suction box (6) a wind box; the suction fan a blower, reversing the position of the igniter, placing it under the grate end and priming through it. In this case a hood should cover the grate.

The cost of sintering with this machine would be far lower than with pots. The mechanical arrangement of a plant operating with these can be so adjusted as to make labor a minimum. The ore and flue dust, or whatever mixture is used, can be thoroughly mixed in a concrete mixer, or a similar appliance, and taken directly to the feed hopper for the machine, where the firing of the igniters and the regulating of the charges for a battery of several machines can be attended to by one man. The dumping is automatic and the sinter could drop directly into small bins just above the charging floor level of the blast furnaces—that is, should the topography of a smelter site allow such an arrangement. At any rate, the labor required would be a minimum. With the exception of the Bormette process, the use of which is not extensive, any of the ordinary sintering methods can employ this machine, and with its reduced cost of operation, I think it will go quite a way in making sintering an economic success.

The Attainment of Highest Purity Iron on a Commercial Basis.

By James A. Aupperle.

The United States Patent Office recently granted to R. B. Carnahan, Jr., two patents,* No. 940,874 and 940,875, covering the process and product for the manufacture of iron of exceptional purity. This iron is known in the trade as American ingot iron and is manufactured by the American Rolling Mill Company of Middletown, Ohio. It possesses properties so radically different from steel, that a description of the process and product will be of interest.

The process of manufacturing American ingot iron consists of charging a high-powered basic open-hearth furnace with iron and high-grade scrap, employing therewith agents for the elimination of the carbon, manganese, silicon, sulphur, and phosphorus, refining the charge in a molten condition at such a high and increasing temperature, and for such a prolonged period, that analysis of bath test shows that the sulphur, phosphorus, carbon, manganese, and silicon, are not over 0.14 when taken in the aggregate, pouring the hot purified iron into moulds, and while in a molten condition, deoxidizing and degasifying to such an extent that the product will not contain over 0.05 oxygen as determined by the method of Ledebur, but will contain at least 99.80% iron.

In practice it is possible to eliminate the silicon, sulphur, phosphorus, carbon, and manganese to 0.03 when taken in the aggregate. The manufacture of American ingot iron is a process of elimination from start to finish, one of the eliminating agents being oxygen.

FIG. 1.—COMPARATIVE TRANSVERSE SECTIONS OF INGOTS.

The process is based on the assumption that iron will not dissolve more than 0.30 oxygen. It is thus a very flexible process, as a large excess of iron ore or mill scale can be added to the bath for the purpose of eliminating the impuri-

*See pages 62 and 98 of the February issue of this journal.

ties, and at the same time the bath at the end of melt will not contain more than 0.30 oxygen, which is then partly deoxidized in the furnace with pig iron, or ferro-silicon.

The deoxidation and degasification are completed in the ladle with aluminium, so that the finished iron will contain less than 0.05 per cent. oxygen and more often the oxygen content will be nearer 0.01 per cent.

Fig. 1 shows a transverse section sawed from the middle of ingots on which no rolling or hammering was done. It shows how completely American ingot iron (at the right) is degasified and also shows the large number of blow holes in the steel (at the left). These ingots were of the following analysis:

	American Ingot Iron.	Steel.
Carbon	.010	.10
Manganese	.003	.35
Silicon	.002	.012
Sulphur	.015	.031
Phosphorus	.001	.065
Oxygen	.012	.034
Specific Gravity	7.95	7.72

It will be seen that the specific gravity is considerably higher in American ingot iron than steel, and often exceeds 8.00 in the rolled sections.

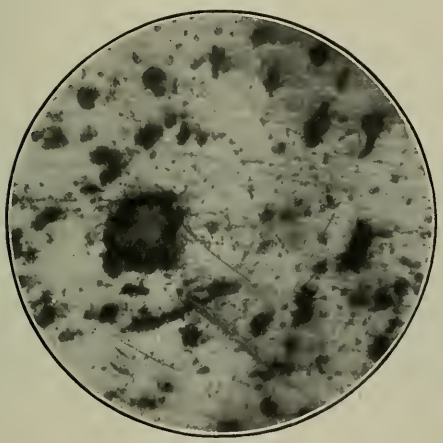


FIG. 2.—TRANSVERSE SECTION. WROUGHT IRON. 56 DIA. SLAG SPOTS. NOT ETCHED.

The importance of low oxygen content of American ingot iron was early recognized, as it was found that besides making the iron red short, it would not galvanize economically if the oxygen exceeded 0.05. The excess of oxygen also caused rapid corrosion. A sample containing above 0.05 oxygen will dissolve more rapidly in acid than material containing below 0.03 oxygen, which is undoubtedly due to the electro-negative character of oxygen, it being more electro-negative than any other element, with the possible exception of fluorine.

The remarkable resistance of American ingot iron to corrosion is shown by the fact that in a 25 per cent. solution of sulphuric acid at 70° C. some grades of 16-gauge Bessemer sheet steel will dissolve in 30 minutes, while under the same conditions, American ingot iron will lose 1 3 of 1 per cent, or in the ratio of 1 to 300.

However, in open-hearth steel of low oxygen contents, the loss is much less than in Bessemer steel, but the American ingot iron is at least twenty times as resistant to sulphuric acid attack as the best open-hearth steel.

Another instance is shown by a neutral copper-sulphate test, where the American ingot iron is several times better than open-hearth steel. This fact is interesting, as it makes the metal suitable for use where there is water containing copper in solution.

In fused caustic soda it outlasts steel about two to one. American ingot iron enamels far better and can be more easily drawn than steel, and also alloys better with cast iron, so that molten cast iron can be poured around it and a perfect union takes place.

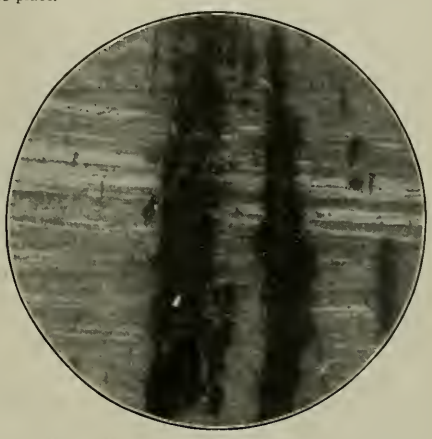


FIG. 3.—LONGITUDINAL SECTION. WROUGHT IRON. 56 DIA. SLAG LINES. NOT ETCHED.

The following table shows the physical properties of American ingot iron to be superior in every respect to wrought iron.

Material	Elastic Limit	Ultimate Stress	Per Cent. Elongation 8 in.	Red. of Area
American Ingot Iron (unannealed)	32290	55550	22.4	54.0
Wrought Iron	30400	49050	16.0	22.4

From the fact that American ingot iron is made in an open-hearth furnace, there is a tendency to call the material steel, but

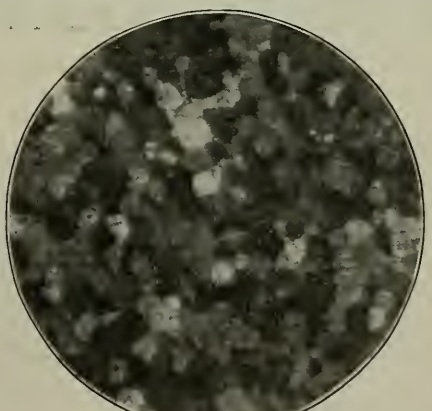


FIG. 4.—AMERICAN INGOT IRON. 56 DIA. DEEPLY ETCHED. GRAINS OF FERRITE.

steel being a compound and American ingot iron practically an element, there is no good reason for calling it steel, and when examined under the microscope, the structure seen is ferrite, which according to the present nomenclature is nearly pure iron. About the same amount of pearlite is present as in the

best Swedish iron (traces only). The micro-structure of American ingot iron and Swedish iron is identical as to the ferrite constituent, the only difference being a few scattered slag lines in the longitudinal and slag dots in the transverse sections of Swedish iron.

In a microscopical examination of a longitudinal section of the common grade of wrought iron (Fig. 3) the large slag lines show a marked difference between it and American ingot iron, which contains no slag.

Fig. 5 shows the effect of positive and negative elements in contact with American ingot iron.* Dish No. 1 shows the effect of carbon in contact with American ingot iron, the carbon being electro-negative, the hydroxyl ions cluster about it and are indicated by the phenolphthaline producing a red color. The positive iron dissolves and is indicated by the blue color produced with potassium ferri-cyanide. American ingot iron contains but traces of carbon, consequently there is less electrolytic action than in steel, which often contains a considerable amount of carbon.

Dish No. 2 shows the effect of oxide of iron in which is a piece of wire, one-half containing oxide of iron, the other half free from oxide, and thus shows the electro negative character of the oxide of iron, the narrow half of the wire being the oxidized side, the wider half showing where the iron has gone into solution, forming Turnbull's blue, while the side containing the oxide of iron became red from the presence of hydroxyl ions.

American ingot iron being exceptionally low in oxide of iron, there will be less electrolytic action than in steel containing considerable oxide of iron.

In Dish No. 3 the electro-positive nature of zinc is plainly indicated, in which a piece of zinc is placed in contact with a strip of American ingot iron, in this case the hydroxyle ions cluster around the iron producing a red color, the zinc in this instance dissolving as zinc hydroxyle and thus protects the electro-negative iron.

This would indicate that should the spelter become broken on American ingot iron there would be very little iron corroded, until the zinc has all disappeared and thus indicates the excellent protection that zinc imparts to iron.

Fig. 6 shows how much better American ingot iron alloys with zinc than steel and also the better protection afforded in resisting the corrosive action of sal ammoniac flux. Two collars, one ingot iron, the other steel, each $4\frac{1}{4}$ in. diameter were put on the necks of the exit rolls of a galvanizing machine, one-half of the roll was immersed in molten spelter, the other half surrounded by sal ammoniac flux. The steel collar corroded five times as fast as the collar made from American ingot iron.

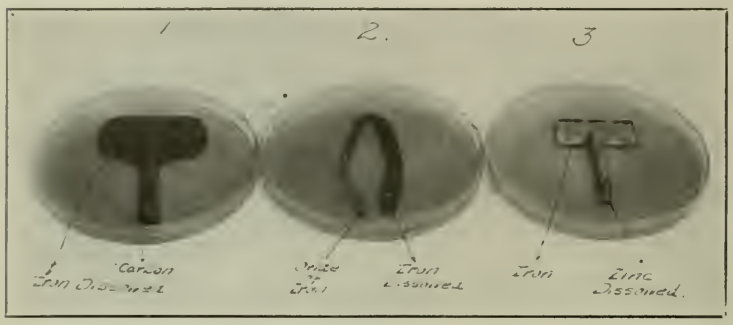


FIG. 5.—ELECTROLYTIC CORROSION TESTS.

FIG. 6.—STEEL COLLAR AND INGOT IRON COLLAR.

Professor Charles F. Burgess of the University of Wisconsin, has the following to say about American ingot iron:

"From time to time we have made microscopic examinations of American ingot iron and have found its structure to indicate high purity, homogeneity, and absence of slag. And from the microscopic appearance we have every reason to believe this iron to be equal to the best Swedish iron in its physical properties and to have the advantage over Swedish iron due to the absence of imperfections caused by slag.

"We have used a large amount of this material for the

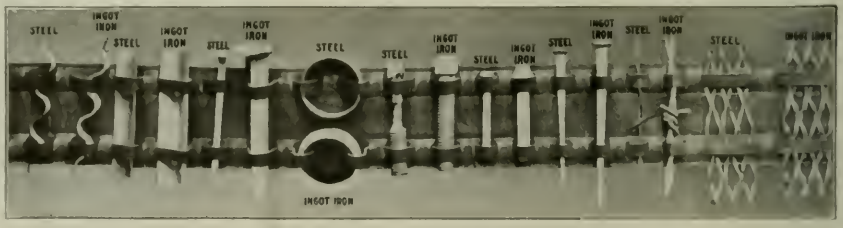


FIG. 7.—INGOT IRON AND STEEL CORROSION TESTS.

*For the electrolytic theory of corrosion and methods of making the anode and cathode spots visible by different colors, see especially the papers of Cushman and Walker, for instance, Vol. V of this journal, pages 264 and 305 (1905).

anode in our electrolytic refining tanks and the manner in which it corrodes shows that it is more dense and uniform than most of the other soft irons we have tried. It is comparatively free from fibers which are so pronounced in some grades of soft iron, which we have employed. In our magnetic tests also, the ingot iron seems to be strictly comparable to high-grade wrought iron.

"As regards corrosion, our experience has been that on ac, count of the absence of impurities, and the uniformity of its structure, there is less local electrolytic action than in most other grades of iron. The result of this is that it dissolves less rapidly than will most other forms of iron."

Prof. J. W. Langley, of Ann Arbor, Mich., refers to American ingot iron as follows:

"American ingot iron is extraordinarily tough and ductile, ranging above wrought iron, and closely approximating the best grade of mild steel of .10 carbon.

"Its combination of great ductility with a fair degree of tenacity is remarkable, and when considered in connection with the resistance to corrosion, makes a distinctly new metal of it. In resistance to acids such as diluted sulphuric and hydrochloric, it resists on an average about twenty times the corrosion of open-hearth steel."

MIDDLETOWN, OHIO.

The Chemist in the Brass Foundry.*

BY W. M. CORSE.

The statement has recently been made that the twentieth century is going to be the era of the chemical engineer, as the nineteenth was of the mechanical engineer. One has but to look about him to see evidences of this development to realize the truth of this statement. More and more of our manufacturers are coming to see that the underlying principles of their processes are chemical and that a knowledge of these principles is essential to the proper conduct of their business if it is to be run at a profit. The brass manufacturer has begun to see this condition, and the number of testing laboratories that have been installed in connection with this industry are proof of this fact.

I have had the good fortune to install four chemical laboratories, three of which have been in connection with brass foundries. A short history of their development may be of interest. The first came into being because of the trouble the manufacturer had with metal melted in oil furnaces. The furnaces did not use a crucible, and the flame came into direct contact with the metal. Oxidation of the metal was greater than in the old-style crucible coke furnace, and the composition of the mixture was varied to such an extent that very bad results were obtained from the castings when they were subjected to hydraulic pressure.

My first work was to ascertain the amount of oxidation and volatilization in order to maintain the composition of the resulting alloy uniform. Numerous tests were made and it was found that 35 per cent more zinc was lost than was allowed for. It was also found that all the remelt was put into one general pile and as this amounted to 50 per cent of the melt, the indiscriminate use of this remelt changed the formula very materially. One thing led to another, until finally all formulas for metal were made up in the laboratory and the resulting melts were analyzed regularly to see if these formulas gave the desired result.

This all may seem very simple, and undoubtedly is, but the fact is that the majority of brass founders are in ignorance of just these principles of their daily operation. The last seven years have witnessed remarkable strides along such lines as described, so that some foundries now know considerably more than they did before the chemist was employed.

* Presented at the March meeting of the Chicago section of The American Chemical Society.

Many times I have been asked the question, "Of what use is a chemist in a brass foundry?" This question was asked by men actually operating foundries. To-day the number of analyses made is probably 500 times as many as five years ago. Although the manufacturer may not have come to the point of using the chemist, he is generally willing to concede his value.

From testing the furnace products for formula, I worked to examine the raw materials for purity. These include copper, tin, lead, zinc, ingot brass, scrap brass, and sometimes antimony. Aluminium also comes in prominently for some kinds of work and has to be tested. The majority of aluminium casting makers buy their alloys ready made as it were, so that less work is done on their component parts than on finished alloys.

The cost of an alloy is, of course, determined by the cost of its component parts, and when these vary quite widely in price, a variation in composition sometimes means the difference between profit and loss. For instance, copper costs now about 14 cents per pound and zinc and lead average 5 cents per pound. An alloy figured for 70 per cent copper, but actually carrying 72 per cent will cost 18 cents a hundred pounds more than it should cost. On a weekly melt of 100,000 lb. this means $180 a week, or $9,360 a year.

This is by no means an unusual instance, and goes to show what a very small saving per pound may mean. It also shows how the exact control of a formula may save money for the manufacturer. It is safe to assert that over a thousand pairs of metals have never been alloyed, and many times this number of combinations of three or more metals have never even been thought of. The surface of this enormous field has scarcely been scratched and offers as much promise for investigation as any that I know.

If any industry needs the attention of the technical man it is the foundry business. Rule-of-thumb methods predominate and opportunities for investigation lie at every hand.

In addition to the metallurgy, we have the various refractory materials and their connection with furnace design and upkeep. They comprise such things as fire brick, fire clay, fire sand and carborundum. The various molding sands come in for attention too. These should be tested for fineness, binding qualities, resistance to hot metal, porosity, etc. Core sands are tested for fineness and refractory qualities. Crucibles are an important item of expense and should be tested regularly to insure maintaining uniform quality.

Coke can be tested for ash and sulphur although a practical test for burning qualities is generally the most important. If oil is used for fuel, it also comes into the laboratory for test.

All brass foundries have copper-bearing refuse of various kinds, which cannot be worked up satisfactorily with the equipment generally available. In order to sell these for anything like their proper value, it is necessary to know either their metallic content or their copper content, which again makes work for the chemist.

As a general thing experimental work cannot be properly taken care of by the foreman. The nature of his duties prevents him from giving concentrated attention to any problem for any great length of time, inasmuch as continued interruptions render impossible the proper recording of his observations. The chemist is, therefore, the logical man to conduct this kind of work. It usually happens that the chemist is the only technical man around the foundry, so that it properly falls to his lot to record the results of all tests made, whether they are chemical, physical, or practical tests of material. He often can be of help in devising production records, loss reports and summaries of such things as pounds produced per man per day, fuel consumed per 100 lb. of metal melted and others too numerous to mention. He can do the planning for engineering work such as is commonly required in a foundry and can aid in the solving of such engineering problems as may arise.

I think I have made it sufficiently clear how a chemist can be used to advantage in any brass foundry large enough to warrant

the outlay for a laboratory. For smaller foundries the commercial testing laboratory is available and does very satisfactory work along some lines. Experience has shown me that many foundries would employ their own metallurgist, as the brass foundry chemist may properly be called, did they but realize the many uses to which his services could be put.

In my position as Secretary of the American Brass Founders' Association, I have had exceptional opportunities to watch the development of the brass foundry laboratory and I have observed that the manufacturers who are making the highest quality goods in a business-like manner are foremost in utilizing this department. Could the practical foundryman be made to realize that a technical man makes his own position more valuable and co-operation vastly increases his own usefulness, a great step forward would be taken in this ancient and intensely interesting industry.

Concentration of Sulphuric Acid.

In our February issue, page 97, we gave an account of the first part of an article by Dr. Gustave Lüttgen in the *Chemiker Zeitung* of January 11, on concentration of sulphuric acid.

The reason for this article is the fact that the hopes placed in the contact process for making sulphuric acid have not been fulfilled in the experience of the last few years, that the numerous difficulties have not been overcome, and that the process is very sensitive. Moreover, the chamber process has had marked improvements in design and operation, and as a result the necessity has arisen for presenting the latest improvements in apparatus for concentration of chamber acid.

The author discusses first the "dish system" (cascade), with its breakage, loss of acid and high fuel consumption; second, the tower system, with its difficulty of operation, expense of installation, high fuel cost and large production of weak acid;* and, third, the "pan system," perhaps better described as "entrainment system" Kessler).

Dr. Lüttgen's article goes on to say that the old Kessler apparatus is so well known that its detailed description is unnecessary. As there is, however, little general knowledge regarding the recent improvements in the Kessler apparatus, the most commanding points will be described.

In the old Kessler system the "saturex" consists of a rectangular chamber built of "volvic" lava having longitudinal channels of pumice stone. The hot gases passed into the "saturex" through channels open at the bottom and reaching downward to the surface of the acid contained in the saturex.

The principle itself was excellent, but the pumice stone was subject to disintegration and failed after a certain time. As a result the gases of combustion did not pass sufficiently through the acid before going to the recuperator, so that the output was decreased. Also these channels did not permit easy cleaning of the saturex because the pumice stone would not stand rough usage.

The Kessler works, owned by Mr. J. Teisset, have now remedied this by the following arrangements, patented by Teisset-Prat in 1909. Instead of the longitudinal pumice stone channels, transverse dividing curtains of volvic lava are now used. The saturex is thus divided into cells above, against which the acid waves caused by the counter flow of hot gases strike.

These changes have increased the output, and also made cleaning very easy, since dividing walls may be taken out for inspection and put back in place without any difficulty. There is practically no breakage and no accumulation of acid mud. The acid produced is always concentrated and perfectly clear.

In the old Kessler apparatus the hot gases were drawn through by one or two steam ejectors of special construction. The porcelain nozzle had to be adjusted very exactly to the steam pressure, and was a rather delicate piece of apparatus. Furthermore, steam pressure was subject to change, whereby the output was diminished. To overcome these disadvantages the steam jet has been replaced by a special apparatus at the end of the system. (Teisset-Prat patent, 1909.)

Some manufacturers tried to insert an exhauster of hard lead or soft lead between the apparatus and the filter. But in these cases it was found that the output decreased, and it was necessary several times in a month to take out the impellers for the sake of repairs necessitated by the attack of the hot acid vapors. There is no danger of this kind with the new apparatus, by which the air is forced from the outside into a pressure transformer inserted in the gas pipe coming from the coke boxes. In the pressure transformer the pressure is reduced, and this reduced pressure is communicated throughout the apparatus.

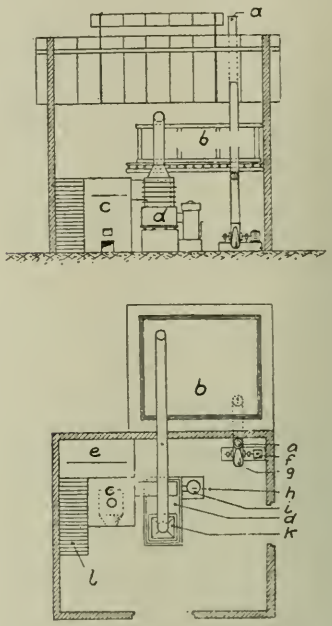


FIG. 1.—CONCENTRATION OF SULPHURIC ACID.

a, steam escape; *b*, condenser; *c*, gas producer; *d*, saturex; *e*, platform; *f*, electric motor; *g*, ventilator; *h*, acid exit; *i*, cooler; *k*, recuperator; *l*, stairway.

With this arrangement the acid vapors do not get into the pressure reducer, so that its life is practically unlimited. Further, while the steam escaping into the air was slightly acid with the old Kessler apparatus, it contains no acid whatever in the new apparatus. On account of these important improvements, the cost of concentration has been considerably reduced.

In the new Kessler apparatus from 80 to 100 kilograms of coke are required per ton of sulphuric acid of 92-93 per cent. The power required is 0.6 horsepower hour per ton. The quantity of weak acid is less than with the old Kessler apparatus, about 80 kilograms of 36° Beaume per ton.

Further, first chamber acid may be used, as the apparatus is very easily cleaned. With the old apparatus of 10-ton capacity two coke boxes were necessary, while one of them may be omitted with the new apparatus. The temperature of the gases when leaving the last plate is only 90°C.

The cost of installation in Germany of the improved Kessler apparatus with a capacity of 10 tons of sulphuric acid of 92-93 per cent is about $6,000, while the cost of repairs has been greatly reduced by the improvements.

* Concerning these two processes see our February issue, p. 97.

Steam is no longer required for the operation of the apparatus. Since electric power is now available everywhere, the new concentration may be placed wherever it is most convenient.

Finally, the new apparatus has also solved the important problem of concentrating chamber acid to acid 98 per cent at small cost. The ease of concentration is largely due to the improvements noted above, as well as the high thermal efficiency.

Radiation losses are reduced to a minimum by the compact construction of the apparatus. This results in a high thermal efficiency.

The coke consumption for concentration to 98 or 99 per cent is from 180 to 190 kilograms per ton; the energy required is 1.5 hp-hours per ton.

An apparatus which yields 10,000 kilograms of 93 per cent

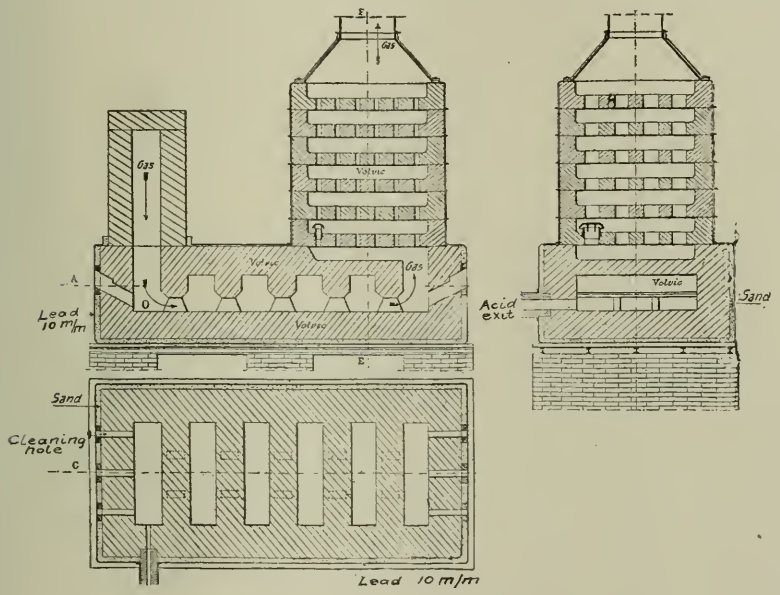


FIG. 2.—CONCENTRATION OF SULPHURIC ACID.

H_2SO_4 yields about 5000 kilograms of 98-99 per cent H_2SO_4. With this concentration the quantity of weak acid is less than 25 per cent.

The following table is a comparative summary of the results obtained with the various concentration apparatus described for a production of 10,000 kilograms of 92-93 per cent H_2SO_4 in 24 hours:

DAILY PRODUCTION, 22,000 LBS., 92-93 PER CENT H_2SO_4, 98-99½ H_2SO_4

	Cascade.	Tower.	Kessler.	Improved.	Improved Kessler.
Coke consumed, lbs.	3,300	2,420 to 2,640	2,200 to 2,420	1,760 to 2,200	1,980 to 2,090
Power used, hp....	2½ to 3	7 to 8	20*	6 to 7	7½
Weak acid, lbs.....	1,320	4,620 1,100	2,640	1,760	2,200 ....
Weak acid, deg....	15	40 14	25 ..	36 ..	38 ..
Installation, cost in dollars	5,000	10,000 to 12,000	6,000 to 7,000	5,600 to 6,000	....
Height without coke filter, in ft.....	12	21 to 45	10½	10½	..

* Coal equivalent to 20 hp.

The Chemical Industries of America*

By Prof. Charles E. Monroe.

(*Concluded from page* 196.)

Having confessed myself, and having found my profession guilty, as charged, I now assert that a chief duty of our profession is to determine methods by which the income may be increased or the costs of living in the land decreased, or preferably both, and I urge as a first measure the advocacy of the policy of preventing any material from leaving the country until it has passed through all processes of manufactures of which it is capable. The meaning of this is evident on inspecspection of the exports of domestic merchandise prepared by the United States Bureau of Statistics, where we find that in 1908 over $885,000,000 worth, or 48.19 per cent of the total exports, consisted of cotton, breadstuffs, meat and dairy products, and coal, much of which had not undergone any degree of manufacture whatever. All this food should have been elaborated in this country into brain and brawn, and the coal made to yield its energy, and then should have been expended here in manufacture. We should further have put into manufactured form the raw materials of other lands.

Inspecting, on the other hand, the table of imports of merchandise prepared by the United States Bureau of Statistics we find that in 1908 but a little over $210,000,000 worth, or 17.87 per cent, of our imports consisted specifically of unmanufactured materials, such as silk, hides and skins, india rubber and gutta percha, wool, cotton, copper, lead and iron ores, and bristles, which would properly go into manufactures here.

We sit back glorying in our country. Its wide extent. Its rich resources. Its teeming millions of independent and self-respecting people. And yet after our fleet has circumnavigated the globe, we continue to sacrifice the fertility of our soils to the support of older civilizations and remain content, while ranging ourselves with those nations that live solely on their primary resources, since the "balance of trade" is in our favor. But we, as chemists, know that this condition cannot last. We know that the average fertility of our soil has been growing steadily less and that only by following sound scientific practice can the fertility of the impoverished soil be restored.

The utilization of the soil as a chemical factory is but one of the problems with which the chemist has to deal. That which appeals most nearly to us as chemical engineers is the item that appears as second in magnitude in the Table of Imports of Merchandise and which has held this second place for years, namely, "Chemicals, drugs and dyes," for this category embraces those substances commonly knowns as chemicals, or the products of the "black art." In 1908 we imported $73,237,033 worth of this class of materials, or 6.13 per cent of our total im-

*Address delivered, by invitation, on December 9, 1909, at the annual meeting of the American Institute of Chemical Engineers in Philadelphia.

ports, while we exported but $20,873,155 worth, or 1.14 per cent of our total exports. There was, therefore, a balance of $52,363,878 against us in this item, in which the chemical engineers of this country are most nearly concerned.

It is true that among these imports are upward of $15,000,000 worth of crude drugs and dyewoods, and quantities of other crude material, but there are many million dollars' worth of substances included here that should have been manufactured in this country. Attention need only to be called to the acids imported to a value of over $1,300,000 to emphasize this fact, for while we are seeking an outlet for our sawdust, we find in this list nearly 9,000,000 lb. of oxalic acid. Or attention might be called to the more than $7,000,000 worth of coal-tar products and preparations, not medicinal. Had this been accomplished there is little doubt that our exports of such substances would also have been large. And what is true of the industries commonly called chemical would equally apply to those larger chemical industries not included in the common category.

Another policy we should follow is the promotion of chemical manufactures throughout a larger portion of our great territory. For this purpose I have prepared Table VI, showing by States the locations of each of the 1786 establishments reported for chemicals and allied products at the census of 1905.

And I find that seven States or Territories, viz., Arkansas, Idaho, Montana, New Mexico, North Dakota, South Dakota and Utah, did not at that time possess a single establishment devoted to any of the large number of industries embraced in chemicals and allied products. Oklahoma, New Hampshire and Wyoming each possessed but one, and the District of Columbia, Nebraska, Nevada, Oregon, Texas and Vermont each less than five.

In order to bring this matter of the distribution of the industries manufacturing chemicals and allied products more clearly to your attention I have, through the courtesy of the director

TABLE VI—NUMBER OF ACTIVE ESTABLISHMENTS FOR CHEMICALS AND ALLIED PRODUCTS, BY STATES, 1905

	1905		1905
Alabama	27	Mississippi	7
Alaska	1	Missouri	47
Arizona	4	Nebraska	4
California	63	Nevada	3
Colorado	6	New Hampshire	1
Connecticut	40	New Jersey	144
Delaware	13	New York	204
District of Columbia	3	North Carolina	42
Florida	15	Ohio	125
Georgia	75	Oregon	4
Illinois	89	Pennsylvania	315
Indiana	52	Rhode Island	17
Indian Territory	1	South Carolina	26
Iowa	6	Tennessee	22
Kansas	10	Texas	3
Kentucky	21	Vermont	3
Louisiana	12	Virginia	62
Maine	9	Washington	9
Maryland	59	West Virginia	25
Massachusetts	77	Wisconsin	19
Michigan	52	Wyoming	1
Minnesota	10		

TABLE IV—VALUES OF EXPORTS OF DOMESTIC MERCHANDISE, BY PRINCIPAL ARTICLES AND CLASSES, IN ORDER OF MAGNITUDE IN 1908, DURING THE YEARS ENDING JUNE 30, 1902–1908

Order of Magnitude, 1908	Articles	1902	1903	1904	1905	1906	1907	1908	
								Value	Per Ct of Total
		Dollars	*Dollars*	*Dollars*	*Dollars*	*Dollars*	*Dollars*	*Dollars*	
1	Cotton, unmanufactured	290,651,819	316,180,429	370,811,246	379,965,014	401,005,921	481,277,797	437,788,202	21 86
2	Breadstuffs	213,134,344	221,242,385	149,050,378	107,732,910	186,468,901	184,120,702	215,260,588	11 71
3	Meat and dairy products	199,861,378	179,839,714	176,027,586	169,998,873	210,990,065	202,392,508	192,802,708	10 51
4	Iron and steel, and manufactures of	98,552,562	96,642,467	111,948,586	134,728,363	160,984,985	181,530,871	183,082,182	10 04
5	Copper, and manufactures of	43,820,070	40,594,613	58,113,006	87,364,000	83,178,635	96,600,698	105,879,711	5 77
6	Mineral oils	72,302,822	67,253,533	79,060,469	79,793,222	84,041,327	84,855,715	104,116,440	5 67
7	Wood, and manufactures of	47,779,848	57,743,535	65,428,417	58,002,977	69,080,394	81,349,575	81,521,305	4 44
8	Leather, and manufactures of	29,798,323	31,617,389	33,980,615	37,936,745	40,642,858	45,476,969	40,688,619	2 22
9	Tobacco, and manufactures of	32,772,849	40,444,689	34,683,531	35,491,019	34,218,847	39,113,011	39,401,679	2 15
10	Coal	20,765,461	21,206,498	27,820,323	29,158,322	28,216,376	34,727,762	39,355,789	2 15
11	Animals	44,871,684	34,781,193	47,977,875	46,728,281	49,139,568	41,201,080	34,101,289	1 86
12	Cotton, manufactures of	32,108,302	32,316,304	32,489,713	49,666,080	52,944,033	32,305,412	25,177,758	1 37
13	Agricultural implements	16,286,740	21,006,622	22,749,615	20,721,741	24,554,437	26,936,456	24,344,198	1 33
14	Cars, carriages, etc	9,872,516	10,499,195	10,936,618	10,610,437	17,788,435	20,513,407	22,072,902	1 20
15	Naval stores	11,733,562	12,918,708	16,145,222	16,106,641	20,075,585	21,686,752	21,641,599	1 18
16	Oil cake and oil-cake meal*	19,779,142	19,743,711	16,899,257	21,498,085	23,386,218	25,738,471	21,004,974	1 15
17	Chemicals, drugs, dyes and medicines	13,626,817	14,150,493	15,065,682	16,595,523	19,155,089	20,173,036	20,873,155	1 14
18	Vegetable oil	15,308,633	16,234,362	12,618,381	16,632,313	15,906,031	19,550,514	19,611,967	1 07
19	Fruits and nuts	8,719,344	18,057,677	20,678,665	15,606,886	15,774,158	17,588,412	14,318,864	78
20	Instruments for scientific purposes	5,389,476	7,130,508	8,297,723	8,172,980	10,887,774	11,661,455	11,578,010	63
21	Fertilizers	6,256,035	6,724,301	7,112,512	7,620,886	8,686,965	8,596,711	10,970,931	59
22	Paraffin and paraffin wax	8,858,844	9,411,294	8,859,964	7,789,160	8,808,345	9,030,992	8,740,929	48
23	Seeds	8,027,824	9,455,283	2,583,325	2,557,747	8,912,662	10,004,609	8,681,688	47
24	Paper, and manufactures of	7,312,030	7,180,014	7,543,728	8,238,088	9,536,065	9,850,733	8,004,706	44
25	Furs and fur skins	5,030,204	6,181,115	5,422,945	6,599,222	8,002,282	7,139,221	7,712,890	42
26	India rubber, manufactures of	4,032,100	4,675,157	5,148,959	5,508,604	6,541,715	7,428,714	7,573,570	41
27	Fibers, vegetable, etc., manufactures of	4,575,219	5,290,948	6,414,616	6,766,809	8,157,211	8,108,112	7,225,798	39
28	Books, maps, engravings, etc	3,997,977	4,442,653	4,347,304	4,844,160	5,839,452	5,841,107	6,107,063	33
29	Grease, grease scraps, etc	2,610,925	2,926,565	3,311,777	3,710,907	4,138,333	5,471,023	5,762,709	31
30	Fish	6,563,199	6,717,274	7,857,041	6,527,803	7,559,178	5,836,856	5,685,916	31
31	Household and personal effects	2,570,369	2,652,787	2,615,076	3,146,969	3,595,128	3,928,946	4,782,416	26
32	Coffee, green	3,209,946	3,295,968	3,656,941	3,966,107	3,484,238	4,692,137	4,314,020	24
33	Sugar, molasses, and confectionery	3,237,139	3,104,053	3,522,805	4,111,707	4,526,314	3,971,800	4,068,167	22
34	Paints, pigments, and colors	2,096,379	2,350,037	2,756,581	3,126,117	3,771,064	3,911,899	4,001,824	22
35	Vegetables	2,546,287	2,543,488	2,603,374	3,200,860	3,667,127	4,007,833	3,895,294	21
36	Explosives	2,062,381	2,454,510	2,441,596	2,559,817	3,568,018	4,082,492	3,705,817	20
37	Brass, and manufactures of	1,910,810	2,000,412	2,657,484	3,025,764	3,474,981	4,580,455	3,701,871	20
38	Soap	1,610,948	2,452,777	2,499,033	2,670,231	2,781,179	3,886,095	3,407,220	19
39	Musical instruments	3,694,141	3,881,509	3,330,982	3,144,787	3,168,062	3,250,061	3,171,521	18
40	Hops	1,550,647	1,909,951	2,116,180	4,486,666	3,136,841	3,543,773	2,961,167	16
41	Nickel, nickel oxide, and matte	1,190,066	1,894,231	942,558	1,196,622	3,240,544	3,218,862	2,948,058	16
42	Clocks and watches, and parts of	2,144,490	2,133,529	2,281,195	2,316,414	2,598,441	3,169,273	2,848,735	16
43	Coke	1,720,457	1,912,450	2,221,211	2,228,442	2,435,604	3,013,088	2,718,185	15
44	Glucose and grape sugar	2,119,286	2,460,022	2,049,545	3,206,794	3,489,193	3,017,527	2,540,640	14
45	Glass and glassware	1,960,196	2,150,609	1,928,483	2,352,709	2,443,904	2,604,717	2,505,417	14
46	Wool, manufactures of	1,512,457	1,723,128	1,987,938	2,015,064	2,119,518	2,219,106	2,219,815	12
47	Lamps, chandeliers, etc	961,638	1,113,290	1,502,888	1,570,125	1,954,991	1,875,850	1,827,116	10
48	Spirits, distilled	2,671,273	1,904,991	1,601,467	1,968,767	1,525,225	1,842,757	1,816,287	10
49	Zinc, and manufactures of	2,017,191	2,146,629	1,710,211	3,085,245	2,780,199	2,141,574	1,666,012	09
50	Hay	2,580,622	828,485	1,052,705	1,089,505	1,116,307	976,287	1,464,010	08
51	Marble, stone, and manufactures of	1,761,696	1,465,744	1,580,790	1,281,219	1,460,561	1,413,121	1,148,996	07
52	Starch	656,705	813,041	1,340,282	1,410,572	1,490,797	1,136,465	1,142,054	06
53	Malt liquors	1,290,063	1,178,740	854,119	1,012,868	1,110,776	1,215,140	1,020,171	06
	All other articles	31,761,904	22,567,281	26,770,512	30,512,483	30,908,594	28,784,582	28,461,024	2 10
	Total	1,455,481,861	1,492,211,483	1,435,170,117	1,491,744,041	1,717,953,183	1,851,718,014	1,814,786,157	100 00

*Corn oil cake not included

of the Bureau of the Census, had prepared a map* of the United States showing the location of the establishments, both principal and subsidiary, manufacturing sulphuric acid, those making explosives, and those engaged in wood distillation, each being a typical industry, and the sulphuric acid industry being generally accepted as of fundamental importance.

From this chart it appears that 13 States and Territories, being the seven already named (Arkansas, Idaho, Montana, New Mexico, North Dakota, South Dakota, Utah) with Iowa, Nebraska, New Hampshire, Nevada, Oregon, and Wyoming, containing 7,648,000 out of the 76,303,387 inhabitants of the continental United States in 1905, or over 10 per cent of the whole, did not possess a single establishment devoted to any one of these industries.

Considering sulphuric acid only, which is so important an industry that it has frequently been referred to as an index of the state of civilization of a people, we find that 23 States and Territories, namely, the 13 just enumerated, together with Delaware, District of Columbia, Kentucky, Maine, Minnesota, Missouri, Oklahoma, Washington and West Virginia, containing 19,562,200 population, or 25.6 per cent of the total, did not possess a single sulphuric acid plant within their borders.

*This map will be printed in a later issue.—ED.

Turning now to the East, we find that 11 out of the 13 original colonies, viz: Connecticut, Georgia, Maryland, Massachusetts, New Jersey, New York, North Carolina, Pennsylvania, Rhode Island, South Carolina and Virginia, contained 30,695,000 population, or 40.2 per cent of the total, and 100 sulphuric acid factories, or 67.1 per cent of the total number existing in the country. Analysis of the statistics of the separate States shows that the number of these establishments does not follow the population, Georgia, for instance, with about one-fourth the population of New York, having twice the number of sulphuric acid factories that New York had.

I am aware that the number of establishments in an industry is an unsafe criterion as to the magnitude or importance of that industry, but this feature has been chosen as lending itself most easily to graphic demonstration. I have, therefore, assembled, by geographic divisions, in Table VII, data for the quantity of sulphuric acid produced, and we find that inspection of this leads to much the same result as to that which was drawn from the consideration of the distribution of the establishments.

All investigations show that there is an enormous extent of fairly well-populated area in this country yet awaiting development by the chemical engineer.

As a field in which costs may be diminished, attention may

TABLE V—IMPORTS OF MERCHANDISE, BY PRINCIPAL ARTICLES AND CLASSES, YEARS ENDING JUNE 30, 1902–1908

Order of Magnitude, 1908	Articles	1902	1903	1904	1905	1906	1907	1908	
								Value	Per Ct. of Total
		Dollars	*Dollars*	*Dollars*	*Dollars*	*Dollars*	*Dollars*	*Dollars*	
1	Sugar	55,061,097	72,088,973	71,915,753	97,645,449	85,460,088	92,806,253	80,258,147	6.72
2	Chemicals, drugs, and dyes	57,723,622	64,351,199	65,294,558	64,779,559	74,477,137	82,997,914	73,237,033	6.13
3	Cotton, manufactures of	44,460,126	52,462,755	49,524,246	48,919,936	63,043,322	73,704,636	68,379,781	5.72
4	Coffee	70,982,155	59,200,749	69,551,799	84,654,062	73,256,134	78,231,902	67,688,106	5.67
5	Silk, unmanufactured	42,635,351	50,011,050	46,100,500	61,040,053	54,080,504	71,411,899	64,546,903	5.40
6	Hides and skins	58,011,168	58,031,613	52,006,070	64,764,146	83,882,167	83,206,545	54,770,136	4 59
7	Fibers, vegetable, etc., manufactures of	39,036,364	39,334,521	40,308,837	40,125,406	51,437,581	67,422,557	54,467,572	4 56
8	Wood, and manufactures of	21,445,599	28,746,271	26,984,353	29,564,323	36,532,706	42,969,941	43,527,982	3.64
9	India rubber and gutta-percha, crude	25,652,977	31,004,541	41,049,434	50,729,873	46,035,685	60,206,418	37,753,266	3.16
10	Fruits and nuts	21,480,525	23,726,636	24,435,854	25,937,456	28,915,747	35,867,160	37,354,742	3.13
11	Fibers, vegetable, etc., unmanufac ured	31,545,962	34,462,513	37,814,285	38,118,071	39,360,290	42,239,358	35,493,083	..97
12	Silk, manufactures of	32,640,242	35,963,552	31,973,680	32,614,540	32,910,590	38,653,251	32,717,668	2.74
13	Iron and steel, manufactures of	27,180,247	51,617,312	27,028,312	23,510,164	29,053,987	40,587,865	27,607,909	2.31
14	Tobacco, and manufactures of	17,706,493	20,579,120	20,073,346	22,145,846	26,590,766	30,192,375	27,267,913	2.28
15	Tin, in bars, blocks, and pigs	19,461,850	23,618,802	21,486,311	23,378,471	30,932,998	38,117,459	25,205,061	2212
16	Copper, manufactures of	10,968,948	17,505,247	18,215,442	19,942,511	25,835,502	39,428,687	24,462,663	2.05
17	Wool, unmanufactured	17,711,788	22,152,961	24,813,591	46,225,558	39,068,372	41,534,028	23,664,938	1.98
18	Spirits, wines, and malt liquors	15,246,640	17,171,617	16,662,702	17,652,323	19,257,590	22,104,235	20,771,804	1.74
19	Wool, manufactures of	17,384,463	19,546,385	17,733,788	17,893,663	23,080,683	22,321,460	19,387,978	1.62
20	Oils	9,300,198	12,283,957	11,179,442	11,593,520	13,723,948	17,068,777	18,292,191	1.53
21	Furs, and manufactures of	15,623,601	15,301,912	14,763,002	18,306,302	21,855,682	21,884,034	15,918,149	1.33
22	Diamonds and other precious stones	23,348,225	31,479,223	22,964,119	33,313,931	40,247,010	42,278,901	16,642,634	1.39
23	Articles, the growth, etc., of the United States, returned	5,815,628	7,170,573	9,899,470	9,079,124	11,134,912	11,833,983	16,599,766	1.39
24	Tea	9,390,128	15,659,229	18,229,310	16,230,858	14,580,878	13,915,544	16,309,870	1 37
25	Cocoa, crude, and leaves and shells of	6,656,504	7,820,087	8,873,709	8,577,649	8,697,515	13,376,562	14,257,250	1 19
26	Cotton, unmanufactured	11,712,170	10,892,591	8,541,510	9,414,750	10,879,592	19,930,988	14,172,241	1 19
27	Leather, and manufactures of	11,317,785	11,294,167	11,100,215	11,666,233	15,140,926	20,359,533	14,127,347	1 18
28	Earthen, stone and chinaware	9,680,156	12,512,052	12,054,014	11,659,723	12,877,528	13,706,790	13,427,969	1.12
29	Fish	8,527,097	8,635,583	9,889,697	10,498,076	11,607,602	12,335,988	12,292,770	1.03
30	Paper, and manufactures of	4,223,125	4,733,036	5,319,086	5,623,638	6,998,761	10,727,885	12,223,058	1.02
31	Meat and dairy products	3,510,696	4,703,536	4,197,466	4,253,414	5,117,054	6,768,432	8,877,183	.74
32	Vegetables	7,039,835	4,581,355	7,008,602	3,983,272	5,092,932	5,728,472	8,289,068	.69
33	Toys	4,023,670	4,232,074	4,977,389	4,964,457	5,887 863	6,993,561	7,209,423	.60
34	Breadstuffs	2,580,295	3,023,160	3,247,503	6,557,347	4,521,667	8,892,968	7,138,214	.60
35	Copper ore	14,003,840	3,385,524	3,466,381	4,892,961	6,727,861	8,296,328	7,057,080	.59
36	Metal, and manufactures of, not elsewhere specified	6,223,383	7,057,202	7,092,125	7,050,118	7,888,565	10,325,446	6,768,637	.57
37	Glass and glassware	6,205,052	7,255,879	6,583,168	5,948,839	7,507,823	7,596,631	6,570,123	.55
38	Seeds	3,252,152	2,831,279	3,587,469	3,457,619	5,388,043	6,404,776	6,371,470	.53
39	Books, maps, engravings, and other printed matter	4,133,215	4,323,938	4,529,187	4,589,858	5,599,948	6,451,309	6,036,693	.51
40	Coal, bituminous	5,310,450	10,562,185	5,043,824	3,713,748	4,367,750	4,184,541	5,123,862	.43
41	Fertilizers	2,426,758	3,100,276	3,503,726	4,524,700	4,446,360	5,341,430	4,970,461	.42
42	Hats, bonnets, and hoods, and materials for	3,050,478	3,871,278	3,963,043	4,379,473	4,571,184	6,820,259	4,852,548	.41
43	Rice	2,926,921	3,061,473	3,073,430	2,010,966	3,082,203	4,392,146	4,798,553	.40
44	Animals	4,624,531	4,533,845	3,129,609	3,337,484	3,914,382	4,344,282	4,777,459	.40
45	Household and personal effects	2,934,244	2,886,007	3,040,523	3,263,384	3,941,875	5,835,354	4,446,187	.37
46	Feathers and down, crude, not dressed, etc.	2,032,566	2,476,659	2,742,018	3,036,791	2,970,260	4,401,131	4,360,721	.37
47	Matting for floors, etc	3,817,866	3,780,050	3,609,795	3,600,088	3,831,436	3,769,202	4,333,044	.36
48	Art works	3,516,536	4,310,315	3,286,262	4,381,324	4,908,782	5,867,265	4,310,767	.36
49	Cork wood, or cork bark, and manufactures of	2,464,934	2,567,580	2,295,138	2,738,319	3,313,306	4,063,982	4,249,006	.36
50	Lead, in ore	4,497,360	4,073,099	3,374,661	3,616,476	3,534,876	3,352,534	4,147,720	.35
51	Paper stock, crude	2,770,255	3,015,084	2,900,713	3,796,595	4,374,464	5,580,528	3,675,926	.31
52	Spices	3,685,242	4,815,125	4,366,008	4,583,356	5,188,116	5,113,000	3,591,537	.30
53	Hair, and manufactures of	2,055,536	2,775,084	2,727,062	3,428,404	3,854,349	3,604,599	3,501,012	.30
54	Iron ore	2,362,544	2,351,278	1,593,279	1,670,683	2,728,854	3,360,449	2,949,462	.25
55	Clocks and watches, and parts of	2,460,324	2,672,310	2,990,474	2,966,495	3,105,136	3,593,173	2,922,142	.22
56	Cement	1,478,452	3,607,666	2,052,952	1,355,696	1,394,447	3,778,114	2,100,598	.18
57	Bristles	2,047,331	2,654,604	2,367,301	2,370,498	2,695,746	3,261,877	2,097,777	.18
58	Jewelry, and manufactures of gold and silver	2,642,345	2,007,433	2,048,821	1,303,652	1,739,951	1,779,527	1,672,275	.14
	All other articles	50,313,903	57,875,712	52,552,007	57,111,870	73,919,068	88,063,181	74,169,712	6.21
	Total	903,320,948	1,025,719,237	991,087,371	1,117,513,071	1,226,562,446	1,434,421,425	1,194,341,792	100.00

be called to the savings of waste. So much has been said on this subject that I hesitate to dwell upon it lest I weary you. But I venture to suggest that one remedy for waste, which has not been so markedly dwelt upon as it deserves, is by a change in location, and I take as an example of this the gas industry.

I have long looked upon our present custom of transporting coal long distances to be converted into gas as uneconomic, for a not inconsiderable quantity of coal is burned to provide the energy with which to haul this coal. Not only that, but, since the gas constitutes but a very small percentage by weight of the coal, there is a considerable waste in hauling the coke, with its ash and the by-products. Further, to provide for emergencies, large stocks of coal must be accumulated in advance at the gas

TABLE VII—QUANTITY OF SULPHURIC ACID PRODUCED IN THE UNITED STATES BY GEOGRAPHIC DIVISIONS 1905 AND 1900

Division	1905	1900
	Tons	*Tons*
United States	1,869,437	1,548,123
North Atlantic	768,647	734,669
South Atlantic	549,593	520,575
North Central	349,906	153,979
South Central	141,107	87,665
Western	69,184	51,235

works, and as coal, particularly gas coal, begins to deteriorate as soon as it is removed from the mine, there is a very considerable loss going on all the time from this cause. Further, as the by-products or residuals are now purchased in the crude state in relatively small quantities at the different gas works, a large part of their value is consumed in collecting and transporting them to central refineries.

By producing the gas at the mine and shipping it by pipe line the cost of haulage on the coke, with its ash and crude by-products, is saved. The wastage of coal by weathering is saved. The cost of collection and transportation of the crude residuals is saved. Such coke as is not needed for industrial purposes can be converted in producers into gas which, by means of internal-combustion engines, can be used in generating electricity for distribution, and the ash from this coal can be put into the mine for use as a filler in place of coal.

It is evident that gas can, under these circumstances, be made and delivered at a much less cost than is the case at present, though it may be necessary after long travel to enrich it near the point of consumption. Furthermore, the valuable areas now occupied by gas plants in our cities can be given up to more concentrated industries and cheap country lands be substituted for them.

I venture further to suggest that frequently an urgent reason for saving waste is to suppress a nuisance, for I do not hesitate to assert that the existence of a public nuisance is evidence of the existence of an economic waste.

Almost at the outset of my professional life, in 1872, I became involved in the famous Miller's River nuisance case, and it fell to my lot to examine, on behalf of the citizens of Cambridge, Mass., the large slaughtering houses which were believed to be the cause of the nuisance, and to study the operations going on within them. The conditions were very complex and there were a variety of causes which led to the creation and maintenance of this most horrible and most extensive nuisance, but among other causes I found that the slaughtering houses had permitted much valuable blood and offal to escape into the stream and that, at that time one establishment alone was pouring into the river, in the water in which it had steamed its hogs, over 5 tons of gelatinous matter per week, and this was done in ignorance of the existence of this matter in the tank waters.

What I have found to be true regarding matter, I have also found to be true as regards energy, and I cite as an example the nuisance of "cannonading" in blasting, which is proof in itself of the use of unnecessarily excessive charges of explosives.

But in urging the abating of a nuisance or advising the saving of waste or the conserving of resources, we should not fail to point out that it can usually be accomplished only with added expense and that a profit can rarely be realized unless the operations are carried out on a quite considerable scale. In fact, it seems to be an economic law that only the rich can really save; that "to him that hath shall be given," for the poor must pay the price of much subdivision and the consequent cost of much handling and a multiplicity of containers.

In fact, we should make it plain that the advocacy of the saving of waste in manufacture and of conserving our resources necessarily implies the use of great aggregations of capital and the carrying on of large-scale operations under a single management. It means the application of methods such as have been applied with great success in the manufacture of hog products or in the refining of petroleum. In dealing with coke at the census of 1905 I found that of the 37,376,251 tons of coal coked in the United States in that census year only 3,317,585 tons, or 8.9 per cent, were coked in by-product ovens, and I estimated from the yields and values of the by-products which were recovered that had all the coal been coked in by-product ovens there was a possible saving of $37,492,136[2]. This is an enormous amount to save in a single industry in a single year, and if the saving could be made an accomplished fact it would go far toward wiping out that humiliating account against us in our imports of "chemicals, drugs and dyes." But I have never failed to recognize the fact that this could only be accomplished by those controlling large capital, and that it meant the "killing off" of a large number of minor establishments, and I have further recognized the fact that the apparent savings set forth could not be realized until the charges against the more costly plant had been satisfied, nor until the market had been so readjusted that it could absorb this greatly increased output of by-products.

As an example of the commercial advantage resulting from the abating of a nuisance I cite the instance of Ducktown, Tenn., whose smelters have for decades been notorious offenders. I will not repeat to you the details by which their devastating sulphurous fumes have been converted into valuable merchandise since they have been so well set forth in current literature, but will simply note that, by report, this saving has resulted in the suspension of a number of the sulphuric acid works in the contiguous region, and I am ready to believe this report to be true, for I look upon this result as a natural consequence of the operation of a wholesome law in economics.

However, all of the endeavors avail but little so long as we remain a dependent nation, which the quantity of manufactured "chemicals, drugs and dyes" imported by us indicate that we are, and especially while we import over $7,000,000 worth of coal-tar products and nearly $2,000,000 worth of ammonium sulphate as we did in 1908, and yet allow $37,000,000 worth of the by-products produced in the coking of our coal to be wasted. It is evident that there is still a wide opportunity for the employment of the chemical engineer in developing our chemical industries.

I find that I have been led to devote my attention to the chemical industries of the United States when you have asked me to treat those of America. I have, however, limited myself, not because I consider our country America, but because of the limited amount of information that I have been able to secure relative to the other countries in North and South America. Such as is available for Canada is found in a paper by Dr W. R. Lang, published in the *Transactions* of the Canadian Institute for 1904, from which it appears that, in 1903, salt was produced in the Dominion to the value of $334,000, and arsenic, in 1901, to the extent of 1,347,000 lb.

Sulphuric acid was produced in Quebec, Ontario and British Columbia, but neither the number of factories nor the extent of the output is given. However, in treating of the plant at Ontario, which produced about 15 tons of acid per day, it is

[2] Bull. 66, U. S. Census of Manufacturers: 1905, p. 18.

stated that imported brimstone was used as the raw material, and this in the face of the fact that Canada bounds in pyrites.

The wood distillation industry flourishes in that country, the plant of the Lake Superior Power Company being, it is said, the largest retort plant in the world, but no statistics of production are supplied. Ammonia liquor was produced to the extent of 235,000 lb. of 28° B strength, a larger part of it being exported.

Soap was produced by some 15 concerns, employing about 2000 hands, the value of the product in 1902 being approximately $3,000,000. Glycerine was obtained from the soap lyes, one works being capable of treating 10,000,000 lb. of lye annually. Petroleum refining was carried on at Sarnia, the factory being able to treat 60,000 barrels of crude oil per month. Calcium carbide was made in two works, carborundum and graphite in one. There was a limited manufacture of fine and heavy chemicals. This about completes the tale for Canada.

My efforts to obtain information relative to the Central American and South American States have been less successful, though I have searched the literature and consulted officials from and to these countries. "The Statistical Abstract of Foreign Countries," recently published by Mr. O. P. Austin, Chief of the U. S. Bureau of Statistics, covers the exports and imports of these countries for a decade, and it appears to be the only authoritative and detailed report concerning them, yet a painstaking search of the tables of exports for each of these Central American and South American countries shows no other chemical items than borate of lime, iodine, and nitrate of soda from Chile; charcoal from British Guiana and Argentina; fermented and distilled liquors from several of the countries, especially from the West Indian Islands; and dyestuffs and extracts from a number of States.

Literature relating to the commercial resources and industrial activities of the Pan-American republics, other than the United States, is apparently quite meagre, and information regarding their industrial activities appears not to have been collected either by the countries themselves or by students of commerce and industry. It does appear, however, from what information can be obtained, that the resources of these countries are in an undeveloped condition and that these countries present an almost virgin field for development by the chemical engineer.

I have myself attempted to inspire one such development, for at the outset of the undertaking of the construction of the Panama Canal by the United States, I advised that dynamite, which has been consumed in enormous quantities in the excavation work, and the manufactured "raw" materials of its manufacture, be made upon the Isthmus. The easy access to the nitrate of soda deposits of Chile, making but a brief water transportation necessary for delivery, and the existence of pyrites in great abundance in the vicinity of the Isthmus, making the production of sulphuric, and hence mixed, acids easy and simple were a few of the many advantages which would follow the adoption of this plan. But not the least would be the civilizing influence which chemical manufacture always exerts. It is unnecessary to say that up to the present I have been unsuccessful in my endeavors to introduce chemical manufactures into the Central American States, but I trust that you, who have done me the honor to listen to me, may succeed where I have failed.

GEORGE WASHINGTON UNIVERSITY,
WASHINGTON, D. C.

The Colorado School of Mines senior classes in mining and metallurgy are now on the annual trip of inspection to the larger mining and metallurgical centers of the West. The trip began April 25 and will end May 23. The Salt Lake City district in Utah and the Butte and Anaconda districts in Montana are the principal points visited. The more important centers in Colorado are usually inspected on short trips at other times during the year.

The Tube-Mill Circuit and Classification.*

BY G. O. SMART.

In view of the increased attention which has been paid to classification for tube-milling, it is of interest to note the results of work carried out on the Rand for the past year. It is now generally admitted that for most efficient tube-milling it is necessary to maintain constant conditions as regards tonnage and moisture in the pulp entering the tube-mill.

A general arrangement of the tube-mill circuit and classification system as used on the Rand is shown in Fig. 1. With this system only those particles in the tailing pulp can overflow the classifier and pass away to the cyanide plant, which are not sufficiently heavy to fall through the classifier and so pass into the tube mill. Thus the particles of richer and specifically heavier pyrites are reduced to a finer state of division before leaving the crushing plant than the poorer and lighter silicious grains, so that the tube-mill works mostly on that portion which best repays to grind. To further obtain full efficiency of the tube-mill the outflow should contain a fairly large proportion of coarse grains, which will be again elevated to the classifiers and return to the mill, while the portion which has been sufficiently finely comminuted by its first passage through the tube-mill will overflow the classifier. The circuit, as shown, also serves as an efficient trap for floured mercury and amalgam.

The rapid increase in the number of cone classifiers employed on these fields, and elsewhere in mining districts, in place of the old-fashioned wooden pyramidal spitzkasten and spitzluten, is due to their greater efficiency, inasmuch as the length of their overflow, which is more than three times as large as that in the usual pyramidal classifier, reduces the velocity of the stream in inverse proportion, so that more efficient settlement is obtained. The use of conical classifiers with peripheral overflow launders naturally involves a central inflow, in place of the side inflow and overflow over the opposite side usual with pyramidal classifiers. This arrangement reduces the velocity of the overflowing pulp and consequently allows more tranquil and better settlement.

The cone classifiers, shown in Fig. 2, are considerably larger than those in use elsewhere on these fields, being 6 ft. in diameter and 9 ft. deep. This is in accordance with the principle of employing units proportional to the size of the plant, and likewise because each tube-mill having but one classifier, its superficial settling area must be sufficient to ensure the settlement at once of all grains in the pulp entering which require further crushing. The principle of operating this classifier differs from that of other hydraulic classifiers in that it must be operated with a considerable depth of settled sand. It will likewise be seen that the upper and lower portions of this sand are separated by a disk near the bottom, of which the edges are 2½ in. from the side of the cone.

In its original form this disk was serrated and was supported by the sides of the cone, but a plain disk supported by projecting arms extending to the sides of the cone, as shown in Fig. 2, is now used. When a wooden disk is used weights are required, also shown in Fig. 2, in order to keep the disk in position until the cone fills up with sand. This disk, or partition, is termed a diaphragm on account of its dividing the upper portion of the cone from the lower and, as will be seen, its function is entirely different from baffles as used for diverting the course of the stream of fluid pulp. While supporting the shifting mass of descending solids, it permits of a continuous flow between its periphery and the sides of the cone, and at the same time prevents a vertical channel from the top of the cone forming through the mass of solids to the outlet, which would allow thin fluid pulp to descend and escape.

Further, the diaphragm relieves the pressure of the mass upon the outlet at the apex of the cone, and, therefore, should be made fairly strong. The relieving on the outlet of such di-

*Abstract of a paper read before the Chemical, Metallurgical and Mining Society of South Africa. The full paper is published in the February (1910) issue of the *Journal* of the Society.

rect pressure prevents the undue packing of the solids at the apex, which would result in the pulp issuing very slowly, even with a free vertical discharge. This slow rate of underflow would necessitate a very large outlet at the bottom to pass the required tonnage, with consequent liability to formation of a central channel, or bore, and sudden discharge of a very large and unmanageable volume of fluid pulp. Such an underflow, if unchecked, speedily results in the discharge of all the solids in the cone owing to increased velocity. Further, if any attempt is made to run even a moderately thick pulp underflow from a classifier without using a diaphragm solids settling on the sides are liable to periodically slip and fall like miniature landslides, and thus, even if the outlet is not completely choked, the tonnage and ratio of water to solids is continually varied.

Since it is obviously impossible to observe what is happening inside a cone classifier the foregoing explanations are advanced from continuous observation upon the behavior of such appliances. The action of the diaphragm is difficult to understand

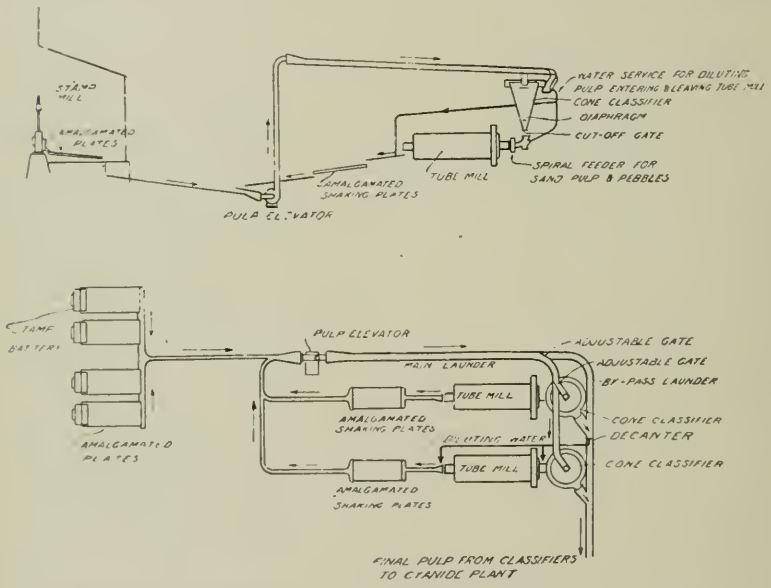


FIG. 1.—TUBE-MILL CIRCUIT, WITH SINGLE CONE CLASSIFIER, AND DIAPHRAGM FOR EACH TUBE MILL.

and is subject to different explanations, but as regards the result that it ensures a steady underflow, in large volume, of thick pulp there is no doubt. The fact that the particles of sand must descend round the periphery of the diaphragm ensures a more uniform settlement of the solids in the cone than if no diaphragm were used.

The upper surface of the settling solids is concave, the edges being higher at the side of the cone. Since the coarser grains naturally settle nearer the center than the finer grains, and since the former constitute the bulk of the settled particles, it follows that in the upper portion of the cone, the faster downward movement of the descending solids is nearer the center than the side of the cone. At the same time particles of insufficient sizes or density to settle near to the center, but which the velocity of the current is insufficient to transport over the edge, settle before overflowing and gradually creep down the sloping side of the cone.

If the ratio of water to solids in the pulp entering the classifier decreases, the finer particles in the settled solids accumulate at the side near the top until the cross-sectional area of the basin is of such dimensions that the velocity of the pulp flowing over its edge, and over the edge of the cone, is sufficient to carry away all particles of such a size as are not specifically heavy enough or large enough to descend. But if with a decrease in the volume of water entering the classifier there is likewise a decrease in the tonnage of solids, or in the average size of particle in the pulp, both the overflow and underflow under these conditions are correspondingly finer. If, however, while the volume of water is decreased, the tonnage and coarseness of the solid particles remains the same, and merely the ratio of water to solid reduced, then the average fineness of both overflow and underflow remain constant.

Similarly, if the volume of pulp entering the classifier is increased, but the tonnage of solids and average size of particle remain the same, only the ratio of water to solid being raised, the cone again automatically adjusts itself to the changed conditions and yields the same grading of overflow and underflow as before. This is due to the area of the basin being increased by extending its edges toward the side of the cone and thereby reducing the velocity of the overflow stream at the point where the solids are discharged to the same as before. If the average size of particles in the pulp entering the classifier be greater, both overflow and underflow are proportionately coarser.

In the practical operation of classifiers, it is not only necessary that a thick underflow of the desired tonnage be secured, but that there should also be an equal volume of pulp of the same consistency delivered to each classifier, as otherwise classifiers receiving more than their proportionate share of the total pulp or solids will overflow a coarser product than others receiving less. It is, therefore, necessary that the overflow of each classifier should be sampled and tested, to secure uniformity. From the foregoing it will be apparent that unless the volume of pulp entering is greater than the capacity for which the cone was designed, which will result in the velocity of the stream being greater than will allow of the settlement of the bulk of the desired particles, it follows that the classifier will automatically adjust itself to all conditions. The object of the horizontal baffle shown near the top of the cone, Fig. 2, below the inlet, is to prevent the force of the stream disturbing the settled solids and to divert the flow of pulp in a horizontal direction towards the periphery of the cone.

Owing to the low percentage of moisture and consequently viscous nature of the underflow it is essential that it should have the freest possible discharge from the outlet of the classifier. Any attempt to discharge the underflow through a pipe with a right-angle turn, or through a curved pipe attached to the bottom of the cone, or even through a vertical pipe of the same diameter as the usual outlet, will be unsatisfactory. The best method at present known for discharge of the underflow is a free vertical fall from the outlet at the apex of the cone through a nozzle, as shown in Fig. 3. It will be observed that this nozzle is practically a continuation of the sides of the cone, and terminates in a sharp edge so that in no portion are there parallel sides. The friction is consequently reduced to a minimum. This nozzle constitutes the wearing portion of the outlet,

and consists of a simple piece of hard cast-iron, which lasts about three weeks, and can be readily replaced at the cost of a few pence.

To shut off the underflow of the cone when required, or to regulate the flow, several devices are suitable, one of which, as shown in Fig. 3, does not obstruct the free vertical discharge. The diameter of the outlet to the classifier, shown in Fig. 3, is 2 3/16 in., which will pass about 400 tons of sand with 26 per cent. of moisture in 24 hours. This large size is required on

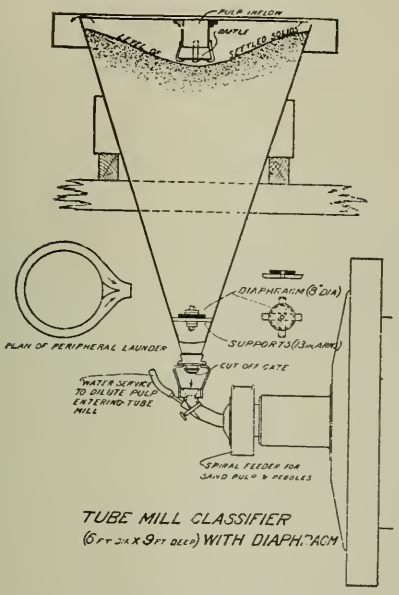

PULP INFLOW
OUTLET
LEVEL OF SETTLED SLIMES
DIAPHRAGM (8" DIA)
SUPPORTS (13 in. APART)
PLAN OF PERIPHERAL LAUNDER
CUT OFF GATE
WATER SERVICE TO DILUTE PULP ENTERING TUBE MILL
SPIRAL FEEDER FOR SAND PULP & PEBBLES
TUBE MILL CLASSIFIER
(6 FT 2 IN X 9 FT DEEP) WITH DIAPHRAGM

FIG. 2.—TUBE-MILL CLASSIFIER WITH DIAPHRAGM.

account of the slow rate of the flow of the viscous pulp. Incidentally, the wear on the nozzle, by abrasion, caused by such slow rate of flow, is considerably less than with the usual type of spitzlutten, delivering pulp at a greater velocity.

The thick underflow described contains too little moisture for efficient tube-milling, and is delivered at so slow a velocity that the method of feeding the tube mill by means of the horizontal jet, as commonly employed, is impracticable. To ensure the delivery of such pulp into the tube mill it is necessary to employ preferably a spiral form of feeder shown in Fig. 2. The additional water required to thin down the pulp for efficient tube-milling is, of course, of some assistance in the delivery of the underflow into the interior of the tube mill. This water may be derived from the mill water service, or may be conveniently obtained from the upper portion of the stream in the tailing launder, in the latter case this water also serves to dilute tube-mill outflow before amalgamation, and is withdrawn from the upper portion of the tailing pulp, by means of a small adjustable decanter, placed in the launder, with an intake opening facing upstream, Fig. 1. The amount of water required is exactly regulated by valves, or cocks, on the pipes from the decanter.

With the contemplated removal of amalgamated plates from the stamp mill, and with very coarse crushing, the importance of continuous and reliable classification becomes very great, so as to prevent any but the very finest gold overflowing the classifiers and to ensure that nearly all the metallic gold liberated by crushing, shall reach the amalgamated plates in the tube-mill circuit. The settling area of a 6-ft. diameter cone, if 28.3 sq. ft., so that with the high ratio of tube mills to stamps, needed with coarse stamp milling, the settling area in the classifiers becomes proportionately large, and the liability for coarse particles of metallic gold to overflow the classifiers is thus reduced.

In consideration of what has been already stated, and by reference to the diagrams shown, it will be seen that several advantages are secured by the method of classification described. In the first place, the simplicity of the general arrangement and the fewness of the units involved are obvious, and consequently the initial cost of installation, attendance, and the liability to derangement are much reduced. The large outlet at the bottom of the single cone for such tube mill precludes the possibility of choking when an inclined ¾-in. mesh screen in the tailing launder is used, in place of the usual ¼-in. mesh, which latter necessitates continual labor in removing ore fragments and debris, under penalty of disastrous results. The saving in elevation by the use of a single cone may be of great importance, especially in an existing plant, when the fall available is insufficient to permit of the installation of the usual spitzlutte, launders and dewaterer for additional tube mills.

This has been illustrated at the Simmer and Jack, where the satisfactory installation of a sixth tube mill was thus rendered possible, without change of launders or pumping arrangements, as also at a mining company in Mexico. At this latter plant an additional tube mill (5 ft. x 24 ft.) was installed, and as little fall was available, there was erected as the tube mill classifier, a cone 4 ft. 10 in. deep and 4 ft. in diameter, with a diaphragm 8 in. from the apex, allowing 1.5 in. annular space between diaphragm and cone. This classifier was run completely filled with sand, and was fed from the battery at the rate of 240 tons of tailing per 24 hours, produced from screen of 256 holes per

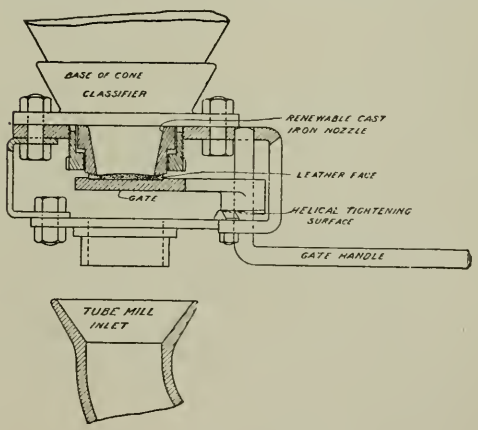


FIG. 3.—DISCHARGE OF UNDERFLOW.

square inch. The following were the results obtained from the classifier:

Gradings.	Inflow.	Overflow.	Underflow.
+ 30	18.5	0.5	34.5
+ 40	10.0	1.0	16.5
+ 80	21.5	8.0	33.5
+100	1.0	1.5	1.5
+150	12.0	11.5	8.0
+200	3.5	9.9	2.0
—200	33.5	68.5	3.5
Percentage of moisture.	90	93	34

The foregoing overflow is fairly fine, but if a finer final product were required, it would be necessary to reduce the volume of pulp entering the cone, or to employ a larger classifier. As under the conditions of this plant 38 per cent. of moisture gives a considerably higher tube-milling efficiency, than higher or lower dilution, a satisfactory tube-mill product was in this way secured, whereas the usual classifiers had failed to

reduce the moisture in the underflow below 48 per cent.

The following table represents the effect produced upon the efficiency of classification for tube milling, by a recent change at the Simmer and Jack plant, whereby the classification for four tube mills was converted from the ordinary pyramidal spitzlutten, launders and dewaterers, to a single cone per tube-mill single cone classifier having previously been installed for the other two tube mills.

		Pyramidal Spitzlutten and Dewatering Cones. 1st to 8th Dec., 1909.		Single Cone Classifiers with Diaphragms. 1st to 8th Jan., 1910.
Tons ore milled		12,208		17,934
Grading analyses of tube-mill feed	+60	61.8	+60	67.1
	+90	22.3	+90	20.7
	−90	15.9	−90	12.2
Grading analyses of mill tailing	+60	43.8	+60	45.2
	+90	11.8	+90	11.7
	−90	44.4	−90	43.1
Grading analyses of final pulp entering cyanide plant	+60	10.9	+60	7.4
	+90	16.0	+90	15.9
	−90	73.1	−90	76.7
Relative internal capacity of tube-mills		100		102.5
Tons—90 product per tube-mill, 24 hours		115.2		126.5
Relative tons—90 product per unit of tube-mill capacity		100		107.3

This comparison of grading analyses and efficiency would appear still more favorable, were it not that only two-thirds of the classification plant was changed, the remaining third having been changed previously to the single cone system. It will be seen from the foregoing that the tons of −90 mesh product per tube mill per 24 hours were raised by 11.3 tons, or allowing for 2.5 per cent. greater internal capacity of tube mills owing to longer average life of liners, the tube-milling efficiency was increased 7.3 per cent. This is due to the better separation in the classifiers of the coarser particles requiring regrinding, and to the thicker and steadier feed of pulp. The low percentage of −90 mesh product entering the tube mills from the single cone classifiers will be noted, as also the reduction of the +60 product from 10.9 to 9.4 per cent. in pulp entering the cyanide works. This is due to the fact that the low percentage of moisture carries down very little finely divided solids or slime.

Owing to the less labor and attention required by the classifiers, two men were transferred to other work.

New York Section American Electrochemical Society.

At the meeting of the New York Section of the American Electrochemical Society held on March 16 Dr. Fritz Zimmermann, of Newark, N. J., presented a paper on the decomposition of carbon monoxide in presence of catalytic agents and in the blast furnace, while Dr. Reston Stevenson of the College of the City of New York lectured on chemical reactions at electrodes. The former paper was published in our April issue, page 182, while the latter paper will be found elsewhere in the present issue.

The following officers of the section were elected for the ensuing year Chairman, Mr. Elmer A. Sperry; secretary-treasurer, Mr. H. B. Coho. A motion of Dr. Baskerville, expressing the gratitude of the section to the retiring secretary-treasurer, Mr. Alois von Isakovics, for his indefatigable work of many years, was unanimously carried.

New York Section American Chemical Society

The program of a largely attended meeting of the section, held on April 8, was a symposium of papers on leather as follows. General outline of the industry, by Allen Rogers; the processes of bating, by Alan A Claflin; vegetable tanning materials, by John H. Yocum; recent advances on chrome tanning, by Otto P Amend; the coloring of leather, by F. E. Atteaux; oils used in the leather industry, by Edgar A. Prosser Previous to that F D. Dodge read a paper on methods of analyzing essential oils.

The next meeting will be held on May 6.

Modern Electroplating Plant.

At the works of the National Cash Register Company, Dayton, Ohio, is to be found an electroplating plant which for size, completeness, and general cleanly operating conditions may well be called a model, and quite on a par with the other high-grade equipment of the "National" company.

Practically all of the metal work used in connection with cash registers is plated, the outside of the cabinet for appearance's sake and the interior mechanism for protection against rust.

The Layout.—The plating department is on the fifth and top floor of one of the buildings, and occupies a floor space 540 x 60 feet. The sides of the building are composed mostly of windows 8 x 8 feet each, and these, together with the elevation, give abundance of light and ventilation. The latter is further improved by ten powerful exhaust fans in the side walls and four in the ceiling of the dip room.

In the main room are 105 plating tanks, each equipped with individual rheostat, voltmeter and ammeter, thus giving absolute control of the pressure and the current. Ninety-three tanks are devoted to "wire work"—that is, the plating of comparatively large pieces, rods, plates, etc., which are hung suspended by copper wires twisted around the cathode rod. The other twelve tanks are fitted with wicker-work baskets, which are slowly revolved in the electrolyte. Each basket has capacity for two or three pailfuls of small inside parts, bolts, nuts, etc.

In addition to the plating outfit, several cleaning tanks are provided. These are connected direct to the line, without meters or rheostat. The castings to be cleaned are cathodes, as in the plating process, and the violent bubbling of hydrogen speedily loosens up the oil, polishing powder, scale, and other discolorations. Iron and steel only are treated in this way, as an automatic method for satisfactorily cleaning bronze does not yet seem to have been discovered. Bronze castings are soaked in soapy water, scrubbed with a hand brush and dipped in weak lye water.

The acid dipping room, 20 by 60 feet, contains one elliptical dipping tank at which two gangs of men can work simultaneously, and one straight tank for one gang.

Pickling tanks for hydrofluoric, sulphuric and muriatic acid pickles are in a smaller room set apart from the other sections, and the fumes are carried off by fans.

The laquer room, 120 by 60 feet, is used in coating the register cabinets or outside casings. It contains ten glass ovens 7 by 20 feet, steam heated, for baking the laquer.

A complete chemical laboratory in another building has rendered valuable service in the plating department, and has also effected a considerable economy in the operation of the entire works.

Solution for Various Finishes.—The following plating solutions are employed:

For nickel, the double sulphate of nickel and ammonia is used, 30,000 gallons being in use.

For copper, (a) copper cyanide and (b) copper sulphate are employed, the quantities being 2600 and 6000 gallons, respectively.

For silver, the cyanide of potassium and chloride of silver is used, 800 gallons being the amount of solution.

For zinc plating, 4000 gallons of commercial sulphate of zinc acidulated with a little sulphuric acid are used.

For cleaning, crude potash and carbonate of potash and carbonate of soda are employed.

The total amount of standard solutions in use is 47,200 gallons, besides several hundred gallons of other solutions which are maintained for experimental purposes.

Notes on Methods.—The method of running through a piece is about as follows: The finished casting is received from the machine shop and dipped in a mixture of sulphuric and nitric acid to remove the scale and stains; then, if a cabinet part, it is sent to the polishing room to be scratchbrush polished and buffed It is then returned for plating.

Nickel-plate can be applied direct to any metal without a preliminary coating of copper, as was formerly thought necessary.

A very low voltage is being used for nickel plating, 1¾ volts on wired work. This is to avoid pitting and to make a softer coat which will take the polish better. The time needed is from one hour on cabinets to three hours on interior parts which receive considerable wear.

Zinc, on the other hand, can be put on very rapidly, 6 volts being used on wired work and about ten minutes being required.

One tank of zinc plating work will keep one man constantly busy. Eight to ten bolts are used on barrel galvanizing.

Copper plating requires first a cyanide solution to act as a "binder" that is to form a preliminary alloy to which the first coating will adhere. This goes on slowly, but about one minute on brass castings will be enough for the coat. Iron and steel take longer.

A transfer is then made to the faster acid sulphate solution for the heavy run.

PLATING PLANT OF NATIONAL CASH REGISTER CO.

When the copper is to receive an oxidized finish the piece, after removal from the sulphate tank, is washed in clear water and dipped in pentasulphuret of potassium (the commercial "liver of sulphur") long enough to get a blue-black color.

It is taken out smutty and is then scratchbrushed with a brass brush in order to bring out the luster of an oxidized finish.

The high parts of the work are then finally relieved with a felt or cotton buff wheel so as to bring out the contrasting copper effects.

Silver plating is used largely on the counter wheels to give a greater contrast with the black lettering. The silver coating is put on in about five minutes after a heavy base coat of nickel of some half-hour's time has been deposited.

Low-Voltage Generators.—The electrical equipment to supply the large amount of current needed is rather interesting, as it includes machines of various makes and different age.

Six direct-current motor-generator sets are used, the line voltage running 225 volts and the speeds from 450 to 850 revolutions per minute on the different sets.

The list is as follows:

Amps.	Volts.	Winding.	Maker.
4000	6-12	Compound	Jantz & Leist Electric Co.
3000	6	Shunt	Zucker & Levett & Loeb Co.
1500	8	Compound	Jantz & Leist Electric Co.
1400	6	Shunt	Eddy Electric Co.
1000	6	Compound	Burke Electric Co.
400	6	Compound	Jantz & Leist Electric Co.

The Zucker & Levett & Loeb 3000-ampere set has been in satisfactory service for the last seven years, and the 1400-ampere Eddy set for 16 years. Each machine has required practically no repair, and is apparently as good as when new. The other machines have not yet had as good an opportunity to demonstrate their long life.

The latest addition to the equipment, as well as the largest and most recent in design, is the 4000-ampere machine built by the Jantz & Leist Electric Company, of Cincinnati, Ohio. The overload capacity of this unit is 50 per cent, which brings the output to 6000 amperes. There are several interesting features connected with this machine, which may be mentioned.

The set, together with switchboard and control apparatus, is installed in a small frame housing erected especially for it on the roof of the main building, where it is removed from the acid fumes and has a generous supply of fresh air. The arrangement has proved so satisfactory that the other machines are to be moved up also. The motor-generator set is constructed in three sections, a 100-hp, 225-volt motor driving the two generators, one on either side, through flexible leather couplings. The normal rating of each generator is 2000 amperes at 6 to 12 volts, and it is compound wound. This winding has been found more satisfactory than straight shunt, as it is impossible to locate all the supplied tanks near enough to the generators to prevent a considerable voltage drop on full load.

A few advantages of the double generator arrangement are the following:

Better efficiency on light loads, as one section may supply all the current and the other run with the field off, so that the latter then remains unloaded.

Two voltages furnished, which is necessary when work of different classes, such as zinc and nickel, is being done.

High voltage obtainable. The two machines may be conveniently connected in series to furnish 24 volts. This may be useful in cleaning wood, though as at present arranged 12 volts is used and considered satisfactory.

When the rearrangement of the various generators is completed it is intended to use one end for cleaning and the other for plating operations.

Distinctive features of the design of the set might be mentioned, such as the elimination of the heavy cables found on most machines by the ring support for brush holders, use of carbon-gauze brushes instead of copper, etc., but it will be sufficient to note that since its installation some months ago the machine has been running with perfect satisfaction.

The plating department of the National Cash Register Company employs 98 men and is in charge of Mr. William Fraine, through whose courtesy we have been enabled to inspect the plant and secure the information for this article.

Improving the Output and Efficiency of Existing Electric Furnaces.

By Carl Hering.

The following case sometimes arises: Given an existing furnace, can it be improved in output or efficiency without a complete change of design? The object of the present article is to describe simple methods and rules by means of which it may readily be determined whether such improvements can be made by mere changes in the electrodes, and if so, how to calculate them, and whether it is worth while making them.

The electrode loss increases with the current, but is independent of the voltage; hence, the total voltage of the furnace should be made as high as possible and the current as small as possible.

The essential part of an electrode is the necessary part within the furnace wall, hence the terminal should be as close to the furnace wall as possible as every inch of electrode between the wall and the terminal is a useless, wasteful resistance. If the electrode requires to be fed, it is preferable, whenever possible, to feed it through the terminal so as to reduce the part between terminal and wall as much as practicable.

The other end, which may be projecting inside of the inside wall, does not necessarily represent an entirely dead resistance, because the heat generated in it either is added to that in the furnace or, if at a lower temperature, it at least tends to keep the furnace heat from flowing out. Such interior projections are not really parts of the essential electrodes, which are, or ought to be, only those parts which are within the walls.

The part within the walls should be surrounded with heat insulating material. The most perfect heat insulator is a body which is kept as hot as the body which it insulates. Hence, one of the best heat insulators would be a furnace wall in which the heat gradient is the same as that in the electrode. The necessarily poor thermal contact between the electrode and the surrounding wall will tend to act as a very high thermal resistance, which aids in this heat insulation.

The outside terminal should, in general, be allowed to get as hot as the conditions permit; the hotter, the less the electrode loss, but the larger the electrode.

Considering the length of the essential electrode to be that part within the walls, plus a part of that within the terminal, say, half the length of the terminal, and perhaps a short part of that portion which may project into the interior of the furnace, then if the electrode tends to burn out the interior of the wall, this length is too great, or, what is the same thing, the section is too small, or the current is too large. If, on the other hand, the furnace product is found to be chilled around the hot end of the electrode, the length is too short or the section too large, or the current too small. In either case the total electrode loss is greater than it need be. And when the product is being chilled by the electrodes the output of the furnace can be increased by lengthening the electrode or decreasing its section or increasing the current, as these changes increase the temperature of the hot end; hence, both efficiency and output can then be increased in this way.

The length being usually fixed by other considerations, the proper section in square inches is calculated by multiplying this length in inches by the current in amperes and by the factor s in the accompanying table for the given temperature drop of that furnace. With that new cross-section there will be practically no chilling of the product nor burning out of the walls around the electrode and the loss will be practically the least possible.

If the existing section is nearly equal to this value, then great accuracy in calculating it or in changing it to this exact size is unnecessary, as the loss increases but slightly for relatively much greater differences in the cross-section, provided the section is near to the best one; this is not the case, however, when the section departs considerably from its correct value.

	Temp. drop C.°	Temp. drop F.°	s	e	i
Carbon	1600	2912	0.00052	2.5	1920
	1800	3272	0.00048	2.7	2070
	2000	3632	0.00045	2.9	2200
	2500	4532	0.00040	3.3	2500
	3000	5432	0.00037	3.7	2700
Graphite	1600	2912	0.00022	1.5	4600
	1800	3272	0.00021	1.5	4700
	2000	3632	0.00021	1.5	4800
	2500	4532	0.00020	1.5	5000
	3000	5432	0.00020	1.5	5100

This minimum loss in watts for one electrode is then equal to the current multiplied by the factor e in the accompanying table for that range of temperature. For intermediate ranges of temperature it is sufficient for most practical purposes to make a proportionate correction of both e and s.

If it is impracticable or undesirable to make this change of the section of the electrode, as it may involve new terminals, the same results can be accomplished by changing the length instead. In that case, the new length should bear the same relation to the old section as the old length did to the new section. In other words, the results will be the same if the ratios of the lengths to the corresponding sections are the same.

Or, if it is more convenient to change the current instead of either the section or length, then the same result may be accomplished by making the current equal to the product of the factor i in the table and the existing section in square inches, divided by the existing length in inches. This, of course, changes the input and may, therefore, require a different source of current.

To find out whether there is any chilling action or unnecessarily high temperature within the walls, or to determine whether the possible gain in efficiency or output is worth making the changes, it is necessary to determine what the conditions of operation are in the existing furnace. This is not quite as simple to calculate because in the existing furnace there may be some heat flowing out of or into the furnace through the electrode.

Let L and S be the length and section of the existing electrode in inch units, and I the normal current which has been used; and let s, e and i again be the factors in the table. Calculate an auxiliary factor a so that $a = sL/S$. Then the total loss in watts through the cold end of each electrode will be

$$\text{watts lost} = \frac{e}{2}\left(\frac{1}{a} + aI^2\right)$$

and the watts of heat flow per electrode leaving or entering the furnace, if any, will be

$$\frac{e}{2}\left(\frac{1}{a} - aI^2\right)$$

which, if positive, means that this much of the power is leaking out of the furnace through each electrode, and is, therefore, chilling the product and is diminishing the output of the furnace; while if negative, it shows that it is flowing into the furnace and, therefore, is developing a higher temperature within the walls than in the furnace. The resistance loss, or I^2R loss, is equal to aEI^2 watts in each electrode.

By comparing the total loss thus determined with that which would be obtained after making the changes, it can be seen what the gain in efficiency would be and, therefore, whether it would be worth making the change. And by comparing the total watts set free in the furnace with those leaking out through the electrodes it will be seen how much the output could be increased if this outflow were stopped.

These last two formulas are not mathematically exact, but are no doubt sufficiently approximate for all practical purposes in the neighborhood of the correct proportions. It will be noticed that the current densities are not the determining factors and are not even involved; also that the two conductivities need not be known.

If the proportions are the correct ones for a normal current, it should be remembered that whenever the current varies it will either develop a high temperature within the walls or cause some chilling in the furnace. If the former is likely to injure the walls it is better to allow for a margin in the current at the expense of some chilling action. If the electrodes are carbon an appreciable improvement in the loss as well as a considerable reduction in the size and often also a more suitable shape can be obtained by using graphite, as the calculations given below will show; the reverse is probably never true. A correctly proportioned carbon electrode tends to be short and thick, sometimes impracticably so. Graphite electrodes tend to be more slender. Iron, and especially copper, are in turn better than graphite, but their substitution for carbon or graphite would generally require a complete change in the design of the whole furnace.*

Example.—The following numerical example will illustrate the above.

Given a 50-kw furnace operating with 1000 amp and 50 volts between outside terminals. Drop of temperature, 2000° C. Electrodes, carbon, 10 in. long and 20 sq. in. in section, they being supposed to have been calculated according to the old current-density rule of 50 amp per square inch. It is assumed that the voltage is fixed and cannot be further increased at the expense of the current.

From the table the value of s for carbon for 2000° drop is 0.00045, hence the new section for that length would be $0.00045 \times 1000 \times 10 = 4.5$ sq. in., or about one-quarter as large; the diameter would be about 2⅜ in. in place of about 5 in. From the table $e = 2.9$, hence, the total loss in each electrode would be 2.9 kw, or 11.6 per cent for both. Or, if it is desired to change the length instead of the section, the new length would be $10 \times 20/4.5 = 44.5$ in.; the loss will again be 2.9 kw.

The total electrode loss originally was 6.77 kw in each electrode, as obtained from the above formula, the quantity a being 0.000255, this left 36.5 kw available in the furnace; the total electrode loss was, therefore, 27.1 per cent, or over one-quarter, hence very high, more than twice what it could have been reduced to. Moreover, from the formula the leakage of heat from the interior of the furnace was 6.1 kw per electrode, which for both together amounts to just one-third as much as is available in the furnace itself, namely, 36.5 kw. This very large outflow of heat would undoubtedly chill a large part of the product in the furnace around the ends of the electrodes, hence reducing the hot zones greatly. The resistance loss in each electrode was only 0.66 kw.

The change to the smaller section would, therefore, result in a considerable saving of electrode material, a reduction of the total electrode loss of from 27.1 per cent to 11.6 per cent, a gain in available energy in the furnace of from 36.5 kw to 44.2 kw, or 21.2 per cent, and a large increase in the available hot space in the furnace as there would be no chilling action. Increasing the length to 44.5 in. would accomplish the same thing, except the saving in electrode material. The current density for the smaller electrode would seem to be very high, about 220 amp per square inch, but with a good water jacket this would probably not cause any trouble; if considered too high, then any desirable intermediate section could be used, in which case the length must be increased proportionately; the gain in power and output will, of course, be the same.

If instead of changing the section or length it is preferred to increase the current, then i from the table being 2200, the new current will be 4400 amp instead of 1000. This will increase the available heat in the interior of the furnace nearly five and one-half fold, making a 220-kw furnace of it, showing that the same furnace, just as it is, could have been increased enormously in output. This shows very strikingly how the older current density rule has misled us.

By using graphite instead of carbon the calculations will show that the electrode loss could be reduced to 1.5 kw per electrode, or nearly half that for carbon, or an additional saving of about 6 per cent; the section would be 2.1 sq. in., equal to about 1⅝ in. diameter, hence much smaller, and the available heat in the furnace would be increased about 6 per cent in addition. If, furthermore, the original carbon electrode were merely replaced by one of graphite of the same size, the proper current would be 9600 amp, or nearly 10 times the original, and as the efficiency would be considerably increased, the available energy in the furnace would be increased 13.1 fold besides making the total interior furnace space useful. While the current density would then be high, the writer has used still higher ones with graphite and experienced no trouble whatsoever.

PHILADELPHIA, April 7, 1910.

* Further information and details concerning the above methods of determining electrodes will be found in the following publications by the writer: *Proc.* Amer. Inst. Elec. Engr., March, 1910, page 285; two papers to be read at the coming meeting of the Amer. Electrochem. Soc.; *Transactions of* this society, Vol. 16, pages 265 and 317 (abstracted in this journal, Vol. VII, p. 514), this journal, March, 1910, page 128, and April, page 188; also a paper by A. E. Kennelly, *Proc.* Amer. Elec. Engr., March, 1910, page 267, on "Modifications in Hering's Laws of Furnace Electrodes."

Electric Smelting of Nickel Ore.

A New Electric Furnace Plant for Nickel Silicide in the South.

The development which is now being carried out by the Consolidated Nickel Company, at Webster, Jackson County, N. C., is particularly interesting as an example of the increasing industrial activity in the South, as well as a further instance of the advance of electric furnace methods into the realm of metallurgy.

The nickel ore deposits of Webster have long been known and for their utilization various processes have been proposed and tried in the past; they were essentially leaching methods, but all of them had to be abandoned, on account of excessive cost, after a shorter or longer trial.

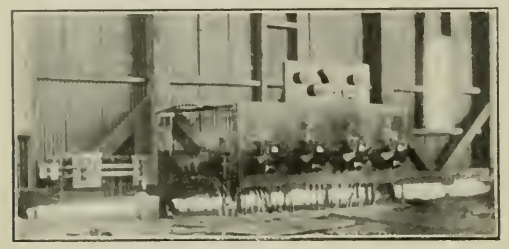
PLANT OF CONSOLIDATED NICKEL COMPANY.

The process now used is an electric furnace process, devised by Dr. Charles Theodore Hennig, of Brooklyn, after an extended investigation. The research in his laboratory was finished early in 1909 and the Consolidated Nickel Company, composed of New York capitalists, then began with the installation of electric plant and the erection of an electric furnace for testing the process on a large scale. In these tests, which have now been successfully concluded, electricity was generated by steam power, but in the ultimate commercial plant water-power will be employed, which is now being developed. For the information contained in the following article we are obliged to Dr. Hennig.

The Consolidated Nickel Company owns the mineral rights to 770 acres of land one-half mile from Webster; this town is

situated 3.5 and 4 miles from Dillsboro and Sylva, respectively, both places being on the Hurphy branch of the Southern Railway. The plant at Webster is to be connected by a spur with the Southern Railway. Hard wood is plentiful and labor is cheap, white labor predominating (farmers), the wages being about $1.50 per day of 10 hours. The cost of Tennessee coal is about $5 per ton at the plant, while wood costs $1.25 to $1.75 per cord delivered.

Nature of Ore.—The ore at Webster consists essentially of hydrated nickel-magnesium silicates, chiefly applegreen

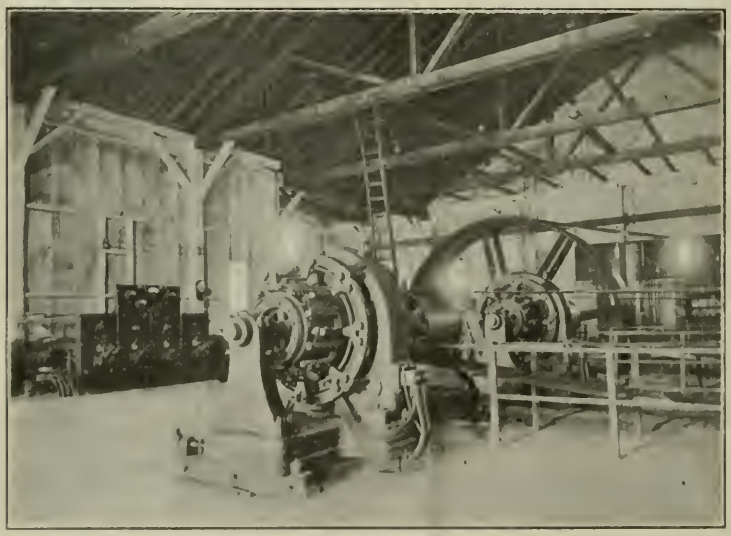


SWITCHBOARD.

garnierite and brown dunite. The garnierite is richer in nickel, containing in the average 5 per cent, while the nickel content in some samples has reached 12 per cent. Dunite is a brown earthy mineral containing 1.5 to 2 per cent nickel. The general average content of nickel in all ore is about 2 per cent. These are two analyses:

Garnierite: 7.3 per cent Ni, 37.5 SiO_2, 29.0 Al, 1.8 Fe, 10.0 Mg, 0.1 CO, 11.9 H_2O.

Dunite: 1.7 per cent Ni, 41.8 SiO_2, 6.7 Al, 8.3 Fe, 28.2 Mg, 0.02 CO, 10.9 H_2O.



PART OF POWER PLANT.

Both ores contain much muscovite and the brown variety contains some chromite.

Electric Smelting.—Dr. Hennig found that these low-grade nickel ores could be converted into a marketable product by a simple direct reduction process with coke in an electric furnace. On a laboratory scale it was thought necessary to add lime to the charge of ore and coke, but when working on the larger scale at Webster, lime was found to be unnecessary. The ore is crushed, when in large lumps, and then mixed with some 10 per cent of crushed coke to reduce the oxides and silicates.

The product of the electric furnace process is a silicide of nickel and iron containing some chrome, aluminium and magnesium and a small quantity of carbon (probably less than 1 per cent). The metal is extremely brittle and is easily crushed; it has a steel-gray color and a high luster on fresh fracture; the specific gravity is about 6.

ONE OF THE FALLS.

This nickel silicide is directly applicable for making nickel steel. As the nickel silicide is obtained from the ore in a single operation, it will be evidently cheaper than metallic nickel for nickel-steel making. The nickel silicide dissolves easily in molten steel.

The exact composition of the nickel silicide varies with the proportion of the charge and the ore, but the composition can be controlled quite readily. It is, of course, advantageous to produce a number of grades for different purposes.

Nickel silicide containing 10 to 30 per cent Ni, 20 to 30 per cent Si, 40 to 50 per cent Fe, 5 to 10 Al, 3 to 5 Cr, and 3 to 4 C, Mg, S, P can be made. It is proposed to market nickel silicide at 10 cents per pound up according to the nickel content and to make no charge for the other constituents.

The molten slag and metal are tapped into large molds or slag pots, and a clean separation is made due to the difference in specific gravity. It was originally intended to crush the slag and pass it over concentrating tables to recover the finely divided particles of nickel silicide, but it will hardly be necessary to do this on account of the clean separation now being made at Webster. In a small laboratory furnace the heat in the slag was smaller and the slag contained much metal; but with a very hot slag the metal readily sinks to the bottom and the slag is free from metal.

Electric Furnace.—The electric furnace is made of a sheet-steel shell, 5/8 in. thick, 6 ft. in diameter and 5 ft. high, and is bolted on a 3/4-in. plate, which is connected to one pole of the electric circuit. The bottom of the furnace thus serves as one electrode; it is formed of a number of iron rods cast into the baseplate to act as conductors; around these iron rods carbon

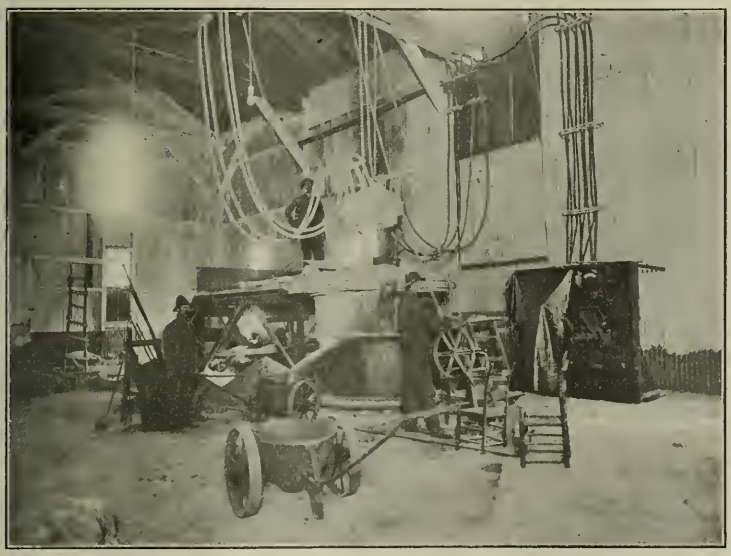
is rammed to form the hearth. The furnace walls are lined with firebrick, so that the inside diameter of the furnace is 4 ft.

The furnace is covered with a cap through which the other electrode, in form of a carbon block or rod, is suspended.

Power Plant.—At present the capacity of the plant is 1000 kw, generated entirely by steam, using wood for fuel. The boiler room, which is situated alongside the mill, contains four 200-hp Wicks return-tubular boilers, furnishing steam for one 1000-hp Hamilton-Corliss engine. This drives four direct-current generators, specially built by the Western Electric Company.

The generators have six poles and are wound to deliver a normal current of 6800 amp at a pressure of 50 volts. They are guaranteed to maintain an overload of 100 per cent for two hours and 150 per cent for 10 seconds without undue heating. As a sudden load of 50,000 amp is liable to occur, all connections, busbars, circuit-breakers, etc., were designed accordingly.

The main circuit-breaker on the furnace switchboard is the largest ever made in the United States, and the switchboard panel itself is made of slate 4 in. thick.



NICKEL SILICIDE ELECTRIC FURNACE.

As has already been mentioned, the electrical energy will be ultimately generated from water-power. Running through the property is the Tuckaseige River, a stream about 100 ft. wide, 2 ft. deep and having a fall of 10 ft. per mile. Two miles from the mine is an excellent dam site, capable of furnishing 2000 hp, and 12 miles away in an air-line is a fall of water from which 6000 hp may be derived. It is proposed to build a dam at the former point, and to install turbines at the latter. Current will be transmitted at a pressure of either 66,000 volts or 110,000 volts, and this will be transformed at the plant.

It is intended not to restrict the manufacture to the production of nickel silicide. By bessemerizing ferro-nickel is to be produced.

Chrome nickel alloys are also to be made, and with ores from another property owned by the company, it is intended to extend the manufacture to all kinds of nickel alloys and metallic nickel.

On the Equilibrium of the System Consisting of Lime, Carbon, Calcium Carbide, and Carbon Monoxide.

By M. deKay Thompson.

1. Introduction.

While the author of the following paper was working on the subject indicated in the title above, an article dealing with the same matter appeared in *Electrochemical and Metallurgical Industry*.[1] The present writer's results did not agree with those in the article referred to, and it was, therefore, thought best to publish a preliminary paper on the subject, which was accordingly presented at the October meeting of the American Electrochemical Society in New York. As the work has now been brought to a close, the following article will be made complete, including all of the preliminary publication that is necessary for clearness.

According to the phase rule[2] the substances taking part in the reaction $CaO + 3C \rightleftarrows CaC_2 + CO$ form a monovariant system; that is to say, for any given temperature there is a definite pressure of carbon monoxide which will preserve equilibrium. In order that equilibrium can exist the reaction must be reversible. The fact that this reaction is reversible has been shown by Rothmund[3] and others.[4] Rothmund also attempted to measure the temperature of formation of carbide by heating to different temperatures lime and carbon, and testing the charge immediately afterward to see if it reacted with water, giving off acetylene. The furnace used consisted of a carbon tube through which an electrical current was passed.

The assumption that must be made with regard to the partial pressure of the carbon monoxide is that it is constant and is due to oxygen of the air acting on the carbon tube, giving one-third of an atmosphere.[5] Unless the temperature is raised above that corresponding to one-third of an atmosphere, no carbide would be found. By repeated trials this temperature could be located within certain limits, if the above assumption is true. In this way Rothmund found 1620° C. as the temperature of formation. Similar experiments were repeated later by Rudolphi,[6] who found the temperature of formation to lie between 1800° C. and 1819° C.; that is, about 200° higher than Rothmund's value. The temperature measurements were made by an optical method, as were also Rothmund's. Finally, Lampen,[7] by a method similar to the above, using a Wanner pyrometer temperature measurements, found 1725° C. for the temperature of formation.

It seemed evident, from the poor agreement of these results, all obtained by the same methods, that some other method would have to be used in which the pressure of the carbon monoxide

[1] C. A. Hansen, this journal, 7, 427 (1909).
[2] See Findlay, "The Phase Rule," p. 16.
[3] *Zeitschr. f. anorg. Chem.* 31, 136 (1902).
[4] A. Frank, *Zeitschr. f. angew. Chem.*, 1905, p. 1734.
[5] Rothmund erroneously assumes the pressure of the carbon monoxide to be 1/5 atmosphere, probably because this is the partial pressure of oxygen in the atmosphere. Taking into consideration that every mole of oxygen produces two of carbon monoxide, ⅓ atmosphere is the result obtained.
[6] *Zeitschr. f. anorg. Chem.* 54, 170 (1907).
[7] *Jour. Am. Chem. Soc.* 28, 864 (1906).

could also be measured, as these differences might be due simply to different values of this quantity. It was the object of the following investigation to make these measurements.

2. Method and Results.

The method decided on was to heat the charge in a vacuum furnace connected with a mercury manometer and to measure the temperature of the charge and pressure of the carbon monoxide when equilibrium is reached. A small Arsem[1] vacuum

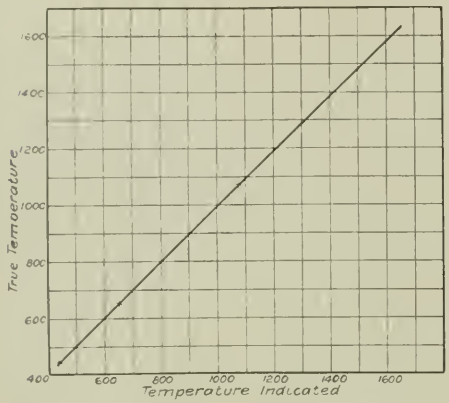


FIG. 1.—CALIBRATION OF THERMOELECTRIC JUNCTION.

furnace, made by the General Electric Company, was the apparatus used. It consists of a cylindrical bronze casting 24 cm inside diameter and 39 cm in length. Parallel to the axis in the center of the casting, and fastened to the lid, is a graphite helix, 27 cm in length, 5.1 in. outside diameter and 0.5 in. thickness of wall. The helix is clamped at each end by water-cooled electrodes. The lid is fastened to the casting with a number of cap-screws and a lead washer.

The whole furnace is immersed in water with the exception of a tower projecting from the center of the lid containing a mica window, making it possible to see the hot material held in the center of the spiral. The support for the crucible is a graphite rod held by the lower electrode, but insulated by lava rings. The lid also contains a pipe by which the furnace may be exhausted. A Geryk oil pump was used for obtaining the vacuum. The pressure could be read by a wooden scale divided in millimeters on a mercury gage completely evacuated and sealed off at one end, thus making a siphon barometer.

The temperature of the gas contained in the furnace is not constant, but all that determines the equilibrium besides the pressure, is the temperature of the solid substances and of the gas in contact with it. Of course, the pressure must be constant throughout the furnace.

In the first experiments the temperature was measured by a Wanner pyrometer, which rendered it necessary to replace the mica window by one of glass clamped between rubber and sealed up with paraffin. In calibrating the pyrometer a similar piece of glass was placed between the amylacetate standard and the instrument. The Wanner was found to be unreliable, however, apparently due to inconstancy in the amylacetate standard.[2]

The furnace was, therefore, calibrated by means of a platinum platinum-rhodium junction—that is, the temperature of the crucible was measured while the power was held constant. The temperature was then subsequently determined by measuring the power applied. Heating was furnished by alternating current with a frequency of 60 cycles per second. This was taken from a transformer switchboard so arranged that the voltage could be varied in steps of about 12 volts. For the finer regulation a carbon-plate rheostat, in which current regulation could be obtained by varying the compression on the plates, was found satisfactory. The terminals were copper boxes filled with water.

Fig. 1 gives the calibration of the junction. The galvanometer was a Siemens & Halske instrument made for this special purpose, but which did not read as high as the melting point of platinum.

TABLE I.—CALIBRATION OF THERMOELECTRIC JUNCTION.

	True temperature.	Temperature read directly from scale.
Melting point of gold.....	1065° C.	1075° C.
Melting point of aluminium..	655° C.	650° C.
Boiling sulphur	445° C.	440° C.

In Table 2 and Fig. 2 the calibration of the furnace is given. The power was obtained from voltmeter and ammeter readings. The ammeter scale read to 5 amp and was connected to a current transformer with a ratio of 60 to 1. This instrument was not calibrated. Two voltmeters with scales from 0-65 and 40 to 160 were used. These were calibrated so as to make them comparable with each other. The alternating-current instruments were of the Thomson type made by the General Electric Company.

In calibrating the furnace the wires of the pyrometer were protected by fused silica tubes, which extended up into the tower in the lid of the furnace. The tubes were covered at the junction by a short graphite tube. This projected through a hole in the cap of the crucible containing the charge and rested in the charge. The bare wires were brought out of the furnace

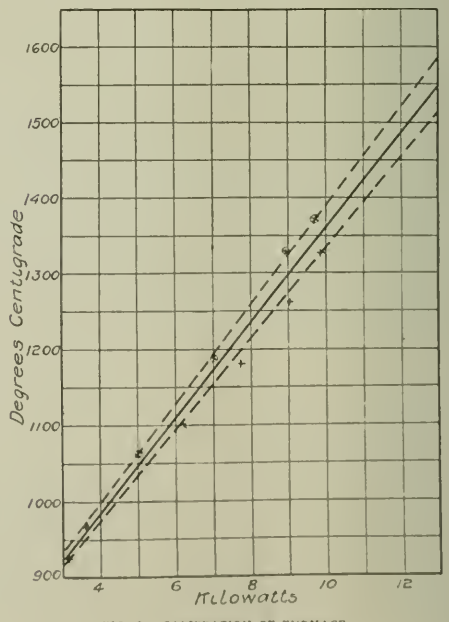


FIG. 2.—CALIBRATION OF FURNACE.

at the top of the tower between rubber washers; the furnace was then evacuated and the calibration taken.

The carbon shield surrounding the spiral was not used in these experiments on account of the fact that carbon absorbs a large amount of gas, which is not easily removed. It will be evident from the method of experimenting described below that its use would not be permissible.

[1] *Trans.* Am. Electrochem. Soc. 9, 163 (1906).

[2] The temperatures measured in the former article on this subject are [illegible] low, but the general conclusions there [illegible] are not affected.

TABLE 2.—CALIBRATION OF FURNACE.

Kilowatts.	Temperature by thermo-electric junction.	Remarks.
3.60	968°	
7.03	1185°	
8.98	1325°	First spiral.
9.71	1368°	
3.12	925°	
4.99	1062°	
7.10	1180°	Second spiral.
8.03	1225°	
9.00	1262°	
9.81	1325°	Second spiral
7.68	1180°	repeated on
6.15	1100°	following day.

In Fig. 2 a circle is put around those points taken with the first spiral. It is evident from this diagram that this method of obtaining the temperature is not as accurate as the Wanner pyrometer would be were it in good condition.

It will be seen that there is no regular difference in the calibration of the two spirals, except that all the points of the first coil lie on the upper dotted line, while some of the points for the second coil lie on the upper as well as the lower. This is probably due to the fact that the second spiral was calibrated more than once. It was thought best under the circumstances to draw the solid line midway between the two extremes and take this for estimating the temperature.

A further test was made to see how constant the temperature was throughout the length of the crucible. For this purpose the junction, protected by silica tubes, was lowered through the window in the tower into the crucibles and the furnace heated without pumping out the air. There was no lid on the crucible in this experiment. The results are given in Table 3.

TABLE 3.—VARIATION OF TEMPERATURE IN CRUCIBLE.

Distance of junction from bottom of crucible.	Temperature.
0.0 cm.	1220°
0.6 cm.	1220°
1.8 cm.	1225°
3.3 cm.	1225°
4.0 cm.	1220°
4.4 cm.	1215°
4.8 cm.	1205°

The power was 8.36 kw.

It is seen that without the lid and with no charge in the crucible the temperature is quite constant, which would be improved, if anything, when the charge is in the crucible and the lid in position.

The carbide used in the following experiments was made from Merck's lime and Acheson graphite powder in the form of turnings from graphite electrodes. Carbide was made by heating a mixture of the two in an arc furnace consisting of a graphite electrode and graphite crucible. By the loss-in-weight method[10] it analyzed 78 per cent pure. The impurities must have been carbon and lime, which were not harmful for these experiments.

The first experiments were carried out at from 1700° to 2000°, but no consistent results could be obtained. After a run at these temperatures it was found that the walls of the furnace were always lined with a white powder, whether lime and carbon were heated alone or when carbide was in an atmosphere of carbon monoxide. It was found when carbide was heated in carbon monoxide to about 1800° only graphite was left in the crucible and the white powder was formed on the walls. When carbide was heated alone in a vacuum the walls of the furnace were lined with a thin sheet of calcium, which easily peeled off and took fire when brought in contact with moisture. Graphite was left behind in the crucible. These two facts taken together show that calcium reduces carbon monoxide according to the equation:

$$Ca + CO = CaO + C.$$

Therefore, if carbide is to be produced, it must either be below the temperature where it breaks up into its elements or the velocity of the reaction

$$CaO + 3\,C = CaC_2 + CO$$

must be greater than the velocity of the preceding reaction. The latter is evidently the state of affairs in the manufacture of carbide, but equilibrium measurements could hardly be made under this condition.

From a number of experiments, which it is not necessary to reproduce here, it seemed that 1500° C. was about the highest temperature at which equilibrium could be measured. This conclusion was based on the quantity of white powder found on the walls of the furnace after runs at different temperatures. Some further experiments at about this temperature showed that it would be impossible to differentiate between the pressure of carbon monoxide and occluded gases that came out of the carbon spiral and the charge on heating in a vacuum.

It was therefore decided to heat the charge in some indifferent gas, which could be drawn off and analyzed for the amount of carbon monoxide present. Hydrogen was, of course, the only gas available. Nitrogen could not be used on account of the fact that it is absorbed by calcium carbide, forming calcium-cyanamide. Hydrogen would have no action on carbide,[11] but it does enter into an equilibrium with carbon monoxide according to the reaction

$$H_2O + C \rightleftarrows H_2 + CO$$

which is the reaction of water-gas formation. If an appreciable quantity of water were produced from hydrogen and carbon monoxide this would react with the carbide and form acetylene and in analyzing for carbon monoxide by absorption in cuprous chloride solution, acetylene would be mistaken for the former. It can be shown, however, that the quantity of water vapor formed is too small to have any effect. The free energy of this reaction is given by the equation[12]

$$\Delta F = -27{,}950 + 31.76\,T + 4.58\,T \log \frac{p_{H_2O}}{p_{CO}\,p_{H_2}}$$

where T is the absolute temperature and the p's are partial pressures. At equilibrium $\Delta F = 0$, therefore, placing the right-hand side of the equation equal to zero, and substituting for T its value 1773° absolute, we find that for 1500° C.

$$\frac{p_{H_2O}}{p_{CO}\,p_{H_2}} = 0.000324$$

If p_{H_2} equals about 90 cm of mercury, as it does in the following experiments:

$$\frac{p_{H_2O}}{p_{CO}} = 0.0029$$

or $p_{H_2O} = 0.003\ p_{CO}$, which is a negligible quantity. The temperature of the gas, however, is not all at 1500°, but falls off to the temperature of the water-cooled walls of the furnace. At 1000° C. $p_{H_2O} = 0.063\ p_{CO}$ which is still a relatively small amount. What actually happens is that at the higher temperatures where the velocity of the reaction is great, the equilibrium varies uniformly with the temperature, but as the gas reaches the cooler portions of the furnace, due to convection currents, it suddenly becomes chilled to a point where the reaction practically stops, leaving the concentrations at values corresponding to the higher temperatures.

(*To be concluded.*)

Mme. Curie has announced the isolation of an appreciable quantity of the rare element polonium. It is more energetically radiactive than radium. The element was extracted from pitch blende and one-half of the quantity isolated disappeared by disintegration in 140 days.

[10] Lunge, Chemisch-technische Untersuchungs Methoden, 5te Auflage, Band II, 711. The drying tube contained a layer of P_2O_5 besides one of $CaCl_2$, which the escaping gas had to pass first.

[11] Moisson, "The Electric Furnace," p. 211.

[12] Bodländer, *Zeitschr. f. Electrochem.* 8, 833 (1902).

Notes on Chemistry and Metallurgy in Great Britain.

(From Our London Correspondent.)

Compression of Fluid Steel.

At a recent meeting of the Sheffield Society of Engineers and Metallurgists, Mr. G. Rodger gave a lecture on "Fluid Pressed Ingots," illustrated with demonstrations of the pressure method of preventing "piping."

The lecturer described the methods adopted for preventing the formation of hollow spaces caused by the contraction of the steel when solidifying. Slow running, feeding with melted steel and the application of feeding heads, were methods generally adopted, but there was still some waste, and every ingot had to be topped, causing a loss of from 10 to 30 per cent. The pressure process provided an effective remedy. It consisted in using moulds, which enabled pressure to be brought to bear upon two opposite sides of the ingot as soon as it was sufficiently set to stand the removal of the distance pieces. The pressure was gradually increased until the ingot was able to bear the maximum pressure intended. He exhibited a wooden model, lent by Messrs. Davy Brothers, of a large press exerting a pressure of about 10,000 tons, and also a drawing of a smaller press capable of dealing with ingots from 200 lb. to 800 lb. The two machines for carrying out the process had been in regular and satisfactory use at Brightside Works for nearly six years. One was capable of dealing with ingots of 30 tons.

In connection with the same subject Von Paravicini states (*Stahl und Eisen*) that strewing cold sand, etc., on the tops of ingots after pouring, in order to check the formation of cavities due to the sudden cooling of the metal against the wall of the mould is not effectual, even when fuel is added at the same time. As a more efficient method he uses a crucible, the bottom half of which has been filled with finely broken coke and the upper half with sand. This is kept as hot as possible in a suitable heating furnace, and as soon as pouring is finished is inverted over the top of the mould, so that the hot sand comes in contact with the metal and is itself covered by the coke. This forms a heat store; the sand prevents the metal being carbonized by the coke, and at the same time helps the metal to retain its own heat. Consequently the solidifying surface of the metal is protected from further cooling, and the weight of the inverted crucible and its contents exerts considerable pressure on the molten steel. The weight of the waste head on ingots and castings is reduced by as much as 40 per cent.

Atmospheric Nitrogen Products.

Professor Crossley has drawn attention to the absence of enterprise in Great Britain in the electrical production of nitrates and other nitrogen compounds from the atmosphere. He pointed out that on the Continent there are many factories for these purposes, whilst in England practically nothing has been done. Not only will these nitrogen compounds be required to supplement or replace the natural nitrates now used as fertilizers—the supply of which cannot be indefinitely prolonged—but atmospheric nitrogen may also under certain circumstances become the only possible source of adequate supply for the manufacture of sufficient nitric acid to meet the requirements of our manufacturers of nitro-explosives, and various chemical products. The question is really serious; and it is to be hoped that the apathy with which it has hitherto been regarded here may soon give place to practical and energetic action.

The Le Chatelier Test for Cement.

At the last (March) meeting of the Concrete Institute Mr. D. B. Butler read a paper on "The Le Chatelier Boiling Test of Portland Cement." After favorably noticing the Bauschinger micrometer-calliper apparatus, he said the only other quantitative method of note for determining expansion was that devised by M. Le Chatelier, which was adopted by the British Standards Committee in 1904 as the standard method. This test had an advantage over the Bauschinger apparatus in that it was cheap and simple, and involved no costly apparatus; but whether it was equally accurate, was quite open to discussion. He had carried out many thousands of tests for soundness by the Le Chatelier method, and the conflicting results he had sometimes obtained had led him to the conclusion that where the condemnation of a cement for unsoundness was concerned, the results given by the Le Chatelier test should not be too rigidly interpreted. It was quite an open question whether a cement should be condemned as unsound because it would not withstand boiling water 24 hours after gauging, whether that 24 hours' preliminary hardening was in cold water or in air.

Considerably less than 20 years ago not 10 per cent of the Portland cement manufactured in England or elsewhere would stand the Le Chatelier test. Was, therefore, 90 per cent of the cement used 20 years ago unsound? This, in view of the hundreds of thousands of tons then used for important engineering work throughout the world, was rather a startling proposition; and having regard to the excellent condition of such work at the present time, was one that could scarcely be seriously maintained. Mr. Butler, in alluding to the present good state of cement work executed long ago, does not appear to have made allowance for the improvement which unquestionably occurs with age. Surely his allusion to the "mass of anomalies and conflicting results continually met with" (in the Chatelier test) is an unintentional exaggeration. At all events such unhappy results are rarely, if ever, obtained by those who adhere rigidly to uniform manipulative detail.

Electric Furnace Steel.

At the meeting of the Sheffield Society of Engineers and Metallurgists early in March, Mr. Gerald Hooghwinkel predicted that in the near future nearly all the steel required would be of special qualities containing some of the rare metals. Almost all these steels had been manufactured in the electric furnace in various countries, and before long no steel works would be complete without electric furnaces to supply at reasonable prices the ever-growing demand for special steels made to stringent specifications. The greatest demand would arise when the railway authorities had settled upon a new rail for main lines; for the present rails were quite unsuitable to stand for any great length of time the strains to which they were submitted. The German State Railway authorities had issued a specification for electric steel rails specifying a tensile strength of 99,000 lb. per square inch, against 85,000 lb. per square inch for basic rails, with a guaranteed life of ten years for track rails. The cost of the electric power was only a small fraction of the total cost of production, about 3 per cent to 8 per cent for hot and cold charges respectively. With a supply at a price between 1 cent and 2 cents per kw-hour the production of electric steel was economical, but the waste gases from blast furnaces and coke ovens were available for profitable power generation on a large scale.

The electrical production of steel castings both for high-grade and medium-grade products would replace the crucible method, to which it was obviously much superior; moreover, the high temperature of the cast could be utilized in manufacturing complicated parts. The total cost of production of steel castings in a one-ton Stassano furnace at the Bonn File Works was $45 per ton as against $75 to $100 per ton for crucible furnace steel. The difference was partly due to the cheaper materials, etc., whilst working costs were $16 less than for the crucible furnace. A certain quality of steel was being made in the United States for $90 a ton which commanded a much higher price in Sheffield.

Corrosion of Steel in Reinforced Concrete.

E. R. Matthews has found that a complete preliminary coating of cement effectually prevents corrosion provided that such coating be allowed to dry thoroughly before encasing with concrete having a porous aggregate. When a non-porous aggregate is employed the preparatory coating of cement is

not necessary. For work exposed to air and water broken stone and hard brick were found to be the best material; whilst porous substances like coke-breeze and slag are not suitable. Where water-tightness is required river sand should not be used in reinforced concrete. Rusted steel embedded in concrete was found to become brighter, even after exposure to water. Linseed oil, turpentine and similar preservatives would probably promote rather than prevent corrosion.

Ferrosilicon.

The recent Local Government Board's report on ferrosilicon perils (see March issue of this journal, p. 133) has evoked from R. Taussig the statement that in furnaces of high power the ferrosilicon now produced is practically pure and contains only minute quantities of arsenides and phosphides. Its quality is demonstrated by its stability on exposure to the atmosphere; and if storage in a moist atmosphere does not result in decomposition, the ferrosilicon will not have any tendency to evolve explosive or poisonous gases. Yet it is proposed in the report to exclude from the market such grades which contain approximately 50 per cent of silicon. Where ferrosilicon is produced in large furnaces the silicon content does not affect the passing of the arsenic and phosphorus into the slag; but in the small furnaces generally employed this refining does not occur.

Estimation of Water in Petroleum.

A rapid method of determining the amount of water in petroleum residues used for fuel, devised by Messrs. R. W. Roberts and A. Fraser, consists in adding calcium carbide and measuring the evolved acetylene. Tests were carried out (a) by taking two equal quantities of the oil, adding a weighed amount of water to one, acting on both with carbide and noting the difference between the volumes of acetylene produced; and (b) by adding excess of carbide to the oil until evolution of gas ceased, then running in a weighed quantity of water, and again measuring the acetylene given off. The average yield of gas, at 0° C. and 760 mm was 580 c.c. per gramme of water, which agrees closely with Dupré's results in his experiments on the estimation of moisture in cordite. The generating flask must be kept cool to avoid evolution of gaseous hydrocarbons from the oil. The acetylene is passed into a saturated solution of sodium chloride, also saturated with acetylene, contained in a jar furnished with an outlet tube, through which the volume of salt solution displaced by the gas passes into a suitably graduated jar.

Accumulators in the Tropics.

Mr. E. P. Harrison, of the Physical Laboratory, Presidency College, Calcutta, states, in a recent issue of *Nature*, that the difficulty experienced in hot countries in keeping small accumulators in working order is probably due to the cells being filled with dilute acid of density 1.190 at a temperature of 30° or 35° C. While this is a proper density to use in a climate where the temperature is 15° to 20°C., corresponding to a 20 per cent mixture, it is too high for a hot climate, where it really represents a 23 per cent mixture; a density of 1.170 or even 1.150 is more suitable in hot climates.

Turbine Drums.

A new method of manufacturing turbine drums for marine propulsion has just been adopted by Messrs. John Brown & Company, who have substituted a special type of rolling mill for the process of hollow forging with the hydraulic press. The mill will roll steel cylinders up to 15 ft. diameter, a length of 10 ft., and 30 tons weight. The firm is now making drums for the two great White Star liners being built by Messrs. Harlan & Wolff. These drums will be 12 ft. 2½ in. in diameter, 7 ft. 8 in. in length, and 24 tons in weight.

High-Pressure Gas Furnaces.

Mr. A. W. Onslow recently presented to the Society of Chemical Industry a description of arrangements of furnaces and injector burners in which constant and easily reproduced temperatures up to 2600° Fahr. can be obtained by the use of gas at carefully regulated pressures, varying from 100 in. of water downwards. The system is applicable to glass-making, lead smelting either by direct heat or by radiation, and to the tempering of axle trees and long steel bars. Large combustion chambers were not necessary with high-pressure gas; and by confining action of the air and gas to the object to be heated the consumption of gas can be reduced two-thirds. He referred to the use of high-pressure gas for the economical warming of buildings as carried out under Colonel Baynold's superintendence at the Royal Arsenal. These improvements gave better heating results at a lower cost as a consequence of the use of smaller combustion chambers and the simplification of apparatus.

Analytical Notes.

At a meeting of the London section of the Society of Chemical Industry early in March "The Rational Analysis of Clays" was the subject of a paper by Mr. W. C. Hancock, who said that the ultimate analysis of a clay showed the total amounts of the acid and basic oxides present, whereas the "rational" or "proximate" analysis was really a mineralogical separation of the constituents of clay into three species, viz., aluminous substance, quartz and felspar. Certain inferences could be drawn from each and served as guides in the employment of the clay. Examination of the portions soluble in acid and in alkali and of the insoluble residue, and comparison of the results with the ultimate analysis would enable the extent and limitations of the process to be judged. To obtain comparable results, standardization of method was necessary.

At a recent meeting of the Society of Public Analysts, Mr. E. R. Andrews, in a "Note on the Analysis of Ultramarine Blue," described a method of effecting its decomposition by agitation of the pigment with saturated bromine water. The result was that practically all the products of decomposition remained in solution. After removal of the silica the alumina, sodium, etc., might be determined in one portion of the filtrate; and the sulphur in another by direct precipitation with barium chloride.

A "Note on the Kjeldahl Estimation of Nitrogen in Fatty Substances," by Mr. J. A. Brown, suggested that to prevent the frothing which sometimes occurs in distilling off the ammonia when determining nitrogen in fatty substances, the acid contents of the flask should be diluted with about 100 c.c. of water, and that 50 c.c. of this should be boiled away before making alkaline and distilling.

Market Prices, March, 1910.

Copper: rose from the 1st (£59 10 s) to £60 10s. on the 3d; thence fell until on the 11th it touched £59. Since then flat and not inclined to alter, closing at £58 15s.

Tin: rose from £148 to £150 in the first three days; dropped rapidly to £143 during the next fortnight, and thence recovered to £144, closing at £146 15s.

English lead has been steady with slight downward tendency; opened £13 12s., closed £13 2s 6d.

Iron, Hematite: Has risen steadily since the 3d, opening 64/3, it closes at 66/-.

Scotch Pig: Has had an upward tendency, starting at 57/4, it closes at 57/6.

India Rubber, Para fine: Has risen throughout the month in a sensational manner. On March 2 it was 8/10½ per lb., on the 9th, 10/1; it closes at 10/5. During the week ending March 26th, smoked sheet reached the record price of 11/0¼. As a result, new companies are incessantly floated. It is probable that present prices are inflated, but, as sudden increase of supply is practically impossible, it is difficult to see how any considerable reduction can occur.

	£	s.	d.
Aluminium ingots	75	0	0
Alumina, Alum, lump, loose, per ton	5	12	6
Antimony Black Sulphide Powder	20	0	0
Borax, British Refined Crystal	16	0	0
Sulphate of Ammonia, f.o.b. Liverpool, per ton	12	10	0
Sal-Ammoniac, firsts, delivered U. K., per ton	42	0	0

Copper Sulphate, per ton	18	15	0
Camphor, 1-oz. tablets		1	7
Caustic Soda, 48 per cent ord., per ton	5	10	0
Bleaching Powder, 35 per cent	4	5	0
Shellac, standard T. N. Orange Spots, per cwt.	3	10	0
Sulphur, recovered, per ton	5	0	0
Carbolic Acid, liquid, 97/99 per cent per gal.		1	0
Hydrochloric Acid, per cwt.		5	0
Creosote, ordinary good liquid, per gal.			2¼

Naphtha, solvent 90 per cent at 160°C., per gal. 1 3½
Platinum, per oz., nominal 5 15 0
Copper Ore, 10 to 25 per cent per unit .9s 9d to 10s 3d
Tin ore, 70 per cent, ton £90 to £92
Mica, small slab, in original cases, per lb. 6d to 1/-

Note.—Of the chemicals it will be noted that with the exception of ammonia and copper sulphate, bleaching powder and shellac (all lower) and naphtha solvent (slight increase) the prices remain firm.

Synopsis of Metallurgical and Chemical Literature

Iron and Steel.

Special Steels; Nitrogen in Steel; Case Hardening.—Vol. I of the *Iron and Steel Institute Carnegie Scholarship Memoirs* (pages 1 to 59), contains a paper by Mr. Walter Giesen on "the special steels in theory and practice." The influence of nitrogen on steel is first emphasized. In the case of mild iron the tensile strength increases with the nitrogen content, while the elongation diminishes. At a critical nitrogen content, which is between 0.037 per cent and 0.041 per cent, the elongation disappears entirely and the steel becomes quite brittle. In the course of a number of examinations of ordinary steels, the author has found the critical limit to be 0.039 per cent of nitrogen, but this critical nitrogen content depends largely on the percentage of carbon in the steel. The author urges that in steel specifications the proportion of nitrogen should be expressly limited to a certain maximum; this is especially important for special steels. Vanadium is recommended to remove both nitrogen and oxygen from steel. The case-hardening of special steel is then discussed and it is emphasized that all parts that are to be hardened and tempered should be made of steel with a low content of carbon and nickel; while chromium steel should be used for bearings, silicon steel for springs and gear, and chromium-vanadium steel for all parts that will be subjected to considerable vibration, and at the same time are required to be hard. The author describes a reliable furnace for case-hardening. A good and cheap cementing medium is composed of 60 parts of ivory black (eborustum), 35 parts of leather-charcoal powder (free from extraneous admixtures), 3 parts of potassium ferro-cyanide ($Fe(CY)_6K_4 + 3H_2O$), and 2 parts of copper sulphate ($CuSO_4 \cdot 5H_2O$), or saltpeter (KNO_3). Carbon and barium carbonate in the proportion 3.2 : 1 ($8OC + 25BaCO_3$); 1.7 : 1 ($59.5C + 35BaCO_3$); or 1 : 1.7 ($35C + 59.5BaCO_3$), also furnish an extremely good cementing powder. Cyanides, too, have a carburizing action at high temperatures, but with these media the crucible must not be filled too full. The results are given of the case-hardening of various special steels. The author then gives a long account of a metallographical, microscopical, and micrographical examination of various special steels and their mechanical and physical properties (nickel-chromium steels, nickel steels, chromium steels, manganese steels, tungsten steels, titanium steels). The general conclusion as to the microstructure of these steels are summed up and a theory of the ferro-carbon alloys and high-speed cutting steels is given, also of the thermal and mechanical treatment of ordinary carbon steels. (High-speed steels; structural steels; structural changes in superheated steel; state of equilibrium of the "iron carbon" systems; alternating stresses, phenomena of hardening.)

Nickel-Steel Riveted Joints; Slip; Corrosion.—Nickel steel was used for the rivets of the Manhattan and Blackwell Island Bridge because of its great strength and other excellent qualities. It was used without extensive tests being made of the strength and other properties of joints riveted with nickel-steel rivets. For this reason an investigation by Dr E. Preuss on the strength of nickel-steel riveted joints, with special reference to resistance to slip, is of especial interest (*Iron and Steel Institute Carnegie Scholarship Memoirs*, Vol. I, pages 60 to 142). Ninety-three nickel-steel riveted joints of all kinds (made from four different kinds of commercial nickel-steel, delivered from stock) were tested. The rivet heads were of the shape generally used either in boiler-making or in bridge construction, according to whether the rivets were delivered by boiler makers or bridge builders. The tests lead to the following conclusions: The strength of nickel-steel riveted joints is about two to two and a quarter times as high as that of joints riveted with wrought-iron or mild-steel rivets. The advantages thus gained are self-evident. For instance, the nodal points in bridge construction are easier to make. As fewer rivets are used, the expenses for rivet material and for making the holes and making the rivets are reduced; while, as rivets of smaller diameter are used, the riveted plates or other parts are less weakened, so that they need not be so broad as they usually are. Though the price of nickel-steel is slightly higher than that of other rivet material, this is of no great importance in comparison to the advantage above mentioned; indeed, the higher price of nickel-steel is balanced by the smaller amount of the material used. Of the nickel-steel tested only that made in the electric furnace was too brittle to be used. It has an exceedingly high ultimate shearing strength, but this was of little use, as, especially in lap joints, the brittleness of the nickel-steel caused the rivet heads to break off before the ultimate shearing strength of the rivet shank had been reached. The remaining three kinds of nickel-steel were extraordinarily tough, and of the large number of rivets made of these only one rivet head was broken off, and in this case the rivet was most probably overheated.

With regard to slipping of the plates of a joint, the results of these tests have led the author to other conclusions than those formed by von Bach. The author, who measured the amount of slip with a mirror-reading apparatus, capable of measuring a slip of 1/5000 mm, found that for nickel-steel riveted joints the slip began asymptotically at loads much smaller than the working load of the riveted joint, whereas von Bach found that no slip took place up to a certain load, but that the plates slipped suddenly when the load was increased by about 1000 kg. In the author's opinion riveted joints should be so designed that the resistance to shearing alone is taken into consideration, as is generally the case. With regard to slip, nickel-steel and wrought-iron rivets were found to be equally efficacious. The slipping of butt joints was greater than that of lap joints of the same ultimate breaking strength. In view of the high shearing resistance of nickel-steel it seems advisable to choose, for a given thickness of plate, a rivet with a shank of smaller diameter than is taken for wrought-iron or mild-steel rivets. As nickel-steel is stronger and harder than the rivet material hitherto used, it requires a greater amount of work (something like 25 per cent greater pressure) to form it into rivet heads.

Some interesting observations were made on corrosion in sea water. According to the electrolytic theory of corrosion the electric potential between the rivets and the plates to be joined together should not be too high in order that the materials should not suffer corrosion by electrolysis. To determine the

influence of electrolytic action six nickel-steel rivets and six wrought-iron rivets were riveted into two mild-steel plates, which were then left for two months in salt water without any protective coating. The percentage of salt in the water was about that of sea water, namely, 3.6 per cent. The following reductions in weight took place in the two months during which the tests lasted:

Material.	Weight of beginning of test. Grams.	Loss of weight during test. Grams.
Mild-steel plate with nickel-steel rivets.....	2567	11
Mild-steel plate with wrought-iron rivets....	2794	6

The electric potential between mild steel and wrought iron and between mild steel and nickel-steel was directly measured, and it was found that the electric potential between nickel-steel and mild steel is higher than that between wrought iron and mild steel when submerged in the liquids used.

Gases Occluded in Steel.—In the *Iron and Steel Institute Carnegie Scholarship Memoirs,* Vol. I, pages 219 to 229, Mr. THOMAS BAKER reports on an investigation of the gases given off from steel when heated in vacuo in an electrically heated glazed-porcelain tube. The experiments were made with steel as cast and not with steel which had been subjected to further treatment, such as reheating or rolling—processes which may bring about changes both in the quantity and the nature of the gases occluded. Hydrogen and carbon monoxide form the major portion of the gases evolved on heating in vacuo; carbon dioxide and nitrogen only amounting to 2 per cent or 3 per cent of the total. An actual analysis of the gases drawn off from a piece of steel free from blowholes, when heated for 36 hours at a temperature of 1000° C. is as follows: CO_2, 0.48 per cent; H_2, 49.62 per cent; CO, 48.05 per cent; CH_4, 0.40 per cent; N_2, 1.45 per cent; the small amount of nitrogen present is rather unexpected, but repeated tests on the steels experimented on by the author have resulted in no increase of the percentage above mentioned. The presence of methane is unusual and is to some extent accounted for by the gases given off from the porcelain tubes employed.

In the experiments on sound steel it was found that hydrogen was given off at as low a temperature as 300° C., the rate of evolution reaching a slight maximum at 600° C., and then gradually falling off until a temperature of 900° C. is reached, when it again slowly rises with the temperature; this gas comprises over 90 per cent of the total gas evolved up to a temperature of 500° C. At this point carbon monoxide begins to be given off; its rate of evolution gradually increases until a temperature of 690° C. is reached, when it is so great as to cause a well-marked maximum. After passing 690° C. the evolution falls off and reaches a minimum at 800° C.; it then gradually increases with the rise in temperature, as in the case of hydrogen. At temperatures higher than 760° C. the gases given off consist of almost equal quantities of hydrogen and carbon monoxide. The total volume of gas given off amounted to 91.86 cu. cm, equal to 10.4 times the volume of the steel employed. Similar changes were observed in the gases given off from unsound steel. The total volume of gas given off amounted to 42.09 cu. cm, equal to 5.2 times the volume of the steel. The influence of aluminium in preventing blowholes seems to be due to the fact that it prevents the liberation of gas during the period of solidification and not to any action it may have in hindering the production of carbon monoxide.

Ferromanganese.—Ferromanganese can be produced from ore only by means of a reagent having a greater affinity for oxygen than manganese has, or for which manganese has a greater affinity than oxygen. Only aluminium and carbon have been proved available for this purpose. By means of the aluminothermic carbon-free ferromanganese is being produced commercially, but is naturally high in price. Mr. E. A. WRAIGHT has studied in detail other possibilities of making carbon-free ferromanganese. (*Iron and Steel Institute, Carnegie Scholarship Memoirs,* Vol. I, pages 143 to 160.) His first research relates to possibilities of decarburizing ordinary ferromanganese reduced from the ore by means of carbon and containing carbon. The following decarburizing agents were tried, either in fusion or cementation, or both: oxides, viz., manganese dioxide or pyrolusite, ferric oxide (hæmatite), silica, barium peroxide, alumina, zinc and copper oxides, lime, magnesia and titanic oxide, also ferrous and manganous carbonates (for FeO and MnO); the gases hydrogen, oxygen, and carbon dioxide; and the metals calcium and aluminium. Most of these failed to give any satisfactory reduction of the carbon content of the alloy, and even when the carbon was lowered the process was in no case commercially practicable. While the results were, therefore, entirely negative, one special observation should be mentioned, namely, that proof that at temperatures of 1300° C., 1400° C., and 1600° C. to 1700° C. manganese is not volatile, contrary to the statement of others.

A second series of experiments was made with a view of producing from the ore directly pure manganese free from carbon. First, manganese oxide was intimately mixed with such reducing agents as flour, sugar or starch, and the reduction carried out in a closed iron tube at a red heat. When ferromanganese was produced it was not free from carbon. Gaseous reduction of the ore was then tried, with carbon monoxide or coal gas or hydrogen as reducing agent, without the desired success. Attempts of substituting metals, for instance, calcium, to produce pure manganese from a mixture of manganese dioxide and calcium also proved futile. The final conclusion of the author is that the commercial and economical production of the carbon-free alloy is an impossibility. This failure may be attributed to several facts which have been definitely proved:

1. Manganese has a greater affinity for carbon than for any other element. Evidence of this is shown in almost every experiment that has been made. The only elements that can replace carbon have been shown to be aluminium and silicon. The use of the former entails heavy expense, and produces an alloy which very rapidly disintegrates; further, only about one part of carbon is removed by the addition of 10 parts of aluminium. Silicon, although it replaces carbon to a certain extent, nevertheless soon reaches a limit beyond which it is impossible to further lower the carbon content without enormously increasing the percentage of silicon, in which case a high-grade silico-spiegel is produced instead of carbon-free ferromanganese.

2. Garrison's statement that "in the reduction of manganese ore by carbon, carbide and not the metal is invariably produced," is supported by the results of coal-gas reduction of the ore, but this fact requires modification unless it is assumed that carbides other than Mn_3C exist. This appears to be quite a probable view.

3. Under oxidizing conditions carbon is only removed from the alloy at the expense of manganese. When oxygen, either in the form of air or as oxides, has been tried for decarburizing, manganese has been invariably oxidized, and, although carbon has been to some extent removed, this loss is out of the proportion to the manganese lost.

Gold and Silver

Agitating Cyanide Pulp.—The use of a false bottom of "silica sponge" in a cyanide tank is the new feature of an agitating and filtering device described in a paper read at the March, 1910, meeting of the Amer. Inst. Min. Eng. by E. G. SPILSBURY. The object of the bottom of silica sponge is to provide a medium through which air may be forced to agitate a pulp of ore and cyanide solution. The bottom is raised 4 in. above the tank bottom, thus forming an air chamber into which air is pumped. Air under a low pressure is forced upward through the porous diaphragm at the same time that the charge of pulp to be agitated is introduced above the diaphragm. The entire charge immediately becomes a seething mass, and on account of the fine division of the air as it comes through the diaphragm there are no direct currents anywhere visible. As

each particle of air is released after its passage through the silica sponge diaphragm, it is not subject to further pressure and merely seeks its way to the surface. As the depth of charge increases in the tank, the air pressure is increased to a point just sufficient to force the air through the head of pulp. The results obtained by the perfect mixing and agitation which follows are noteworthy. Extractions of 60 to 75 per cent are obtained in two hours, and an extraction of 99 per cent of the gold and 90 to 95 per cent of the silver is effected in 10 to 12 hours. The low consumption of cyanide is also an important feature of the process.

The silica sponge diaphragm has been found to be an excellent filter for slime pulp, although at first glance it might be regarded as a poor medium, likely to clog in a short time. A year's testing has shown that in the filtration of fine slime pulp, the cake forms promptly on the surface of the filter and does not penetrate and clog the interior. The filter now being prepared for the market has a capacity of 75 gallons of water per square foot of surface per hour under a pressure of 1 ft. head. The formation of a horizontal cake offers an opportunity to increase the thickness of the same to a point limited only by the porosity of the cake itself and the readiness with which it will take the wash water. By moving the filter into a vertical position and reversing the air current the cake is readily discharged.

Other uses for the silica diaphragm, which is an invention of J. E. Porter, of Syracuse, N. Y., are being sought in the arresting of flue dust in hot and acid fumes and in the making of catalytic plates for sulphuric acid manufacture.

Cyaniding Ores of Eastern Oregon.—In a brief note on some laboratory tests made on refractory ores of Eastern Oregon, Algernon Del Mar, writing in the *Eng. & Min. Jour.* for March 26, states it as his belief that an all-sliming system will prove successful on these ores. The precious metals are, as a rule, disseminated in a quartz or slate matrix. This was shown on a test of a representative ore in which the easily crushed portions assayed the same as the harder parts, while the small quantity of sulphide concentrates obtained did not assay twice as much as the original ore. The gold is not easily amalgamable. Cyanide processes have been tried, but not in their latest development. Experiments by the author showed that the finer the ore was ground, the better was the extraction. Financial reverses prevented the erection of slime filtration plants, so that only the laboratory tests are available as indicating the success of the all-sliming system.

Ammonia-Cyanide Extraction.—The alkaline nature of potassium cyanide and its selective action, permitting of the treatment of the most varied gangues, and allowing of the extraction of the values without much loss of the solvent, are assigned as the valuable features of potassium cyanide in the metallurgical industry by D. Mosher, writing in the *Pacific Miner* for March, 1910. Looking for a similarly valuable solvent for copper and zinc in ores carrying gold and silver, which would make it possible to handle these ores in a leaching process like cyanidation, the author believes that such a solvent is found in ammonia-cyanide. It is a stable, alkaline solvent, has selective action, extracting nothing from the gangue and only the ammonia soluble metals from the ore. Through the addition of cyanide to the ammonia, the extraction of gold and silver is also cheapened and simplified. The author expects that calcium cyanide will prove a source of cheap cyanogen, since he states that it can be made by heating cyanamide to a bright red heat in the presence of acetylene vapor or any gas rich in carbon. Cheap calcium cyanide with ammonia should make it possible to treat ore not now handled at a profit.

The chemistry of the ammonia metal combination shows that ammonia combines with metallic oxides and hydroxides to form soluble compounds. In this way silver, copper, zinc, nickel, cobalt, tungsten, uranium and vanadium form soluble ammonia compounds. The copper compound is given as $CuO \cdot (NH_3)_4 \cdot H_2O$; zinc as $ZnO \cdot (NH_3)_4 \cdot H_2O$; and nickel as $NiO \cdot (NH_3)_4 \cdot H_2O$. The solution is to be effected as much as possible in the cold, as the solvent power of the ammonia decreases with a rise in temperature and increases on cooling the solution down to the freezing point. In practice, it will take about ½ lb. to 1 lb. of NH_3 to dissolve 1 lb. of copper, zinc or nickel out of an ore. The recovery of the ammonia is an essential part of the process, and this is effected by the use of ammonia-tight apparatus. The recovery is said to be effected under less severe conditions than exist in ice making or in the Solvay process. In the boiling-out process the metallic oxides are precipitated as the ammonia distills off.

Copper.

Improvements at the Washoe Smelter.—Considerable interest always attaches to smelting and concentrating practice as carried out at the famous Washoe plant, in Montana, where operations are on a large scale and efficiency is kept at its highest, both financially and metallurgically. In the April issue of *Mines and Minerals* the latest changes and improvements at the plant are given. The concentrator is now treating 8000 to 9600 tons of ore per day. Heavier crushers are used and Huntington mills have been enlarged from 5 ft. to 6 ft. A larger-sized feed is sent to the coarse Hartz jigs and their screen area has been increased 58 per cent. In the McDougal roasters a new style of arm and rake have been introduced to facilitate repairs. In the new type the contact faces between arm and rake are protected against incrustation, so that broken rakes are easily removed. In the blast-furnace department considerable attention has been given to reducing air leaks in the air mains, bustle pipes and tuyeres. Another feature in this department is the reduction of the coke used, from 10.5 per cent to 9 per cent, by increasing the quantity of sulphides in the charge to give a sulphur content of 11 per cent to 12 per cent instead of 8 per cent to 9 per cent as formerly. In the converter department the principal change has been the use of second-class Butte ore for lining the converters. Formerly the Snowstorm (Idaho) ore was used for this purpose, being mixed with concentrator slime in the proportion of 2.9 tons ore to 1 ton slime. Second-class Butte ore is now similarly used and, although it contains more iron and sulphur than the Snowstorm, it is giving good service. Improvements have been made in the converter repair department which facilitate the relining operations. A Sanford portable welding machine has been introduced for autogenous welding and is used mostly in repairing the pressed-steel buckets of the slag conveyors.

Sintering Copper Ores.—An interesting discussion of the present status of copper smelting and of the need of a suitable process for sintering fine copper ores is given by Mr. Walter G. Perkins in the March, 1910, issue of the *Mining Magazine* (London). The author emphasizes the impossibility of divorcing the interests of the mine, concentrator and smelter, in view of the changing conditions which have led to the exploitation of very low-grade copper deposits. On account of the high silica and low iron and sulphur in copper concentrates from the low-grade "prophyry" deposits of the West, smelting by roasting and reverberatory practice is close to its limit, both technically and economically. The concentrator can make only a narrow saving lest the silica become excessive and the iron and sulphur prove insufficient for subsequent operations. If a greater saving is to be made more silica must go into the concentrate, and if this is true, improvements must be made in the smelting methods.

Reviewing the past improvements that have been made in reverberatory smelting in Montana and Utah, the author shows that there is still a limitation to the use of reverberatories and that they will not handle charges as refractory as those smelted in blast furnaces. He comes to the conclusion that "it is evident that improvements are demanded in smelting methods just as much as in water concentration if we are to save some of the 30 per cent to 40 per cent of the copper that goes over the

dump in the most modern of the large concentrating plants." And further: "The economic problem resolves itself into the cost of roasting in McDougal roasters with smelting in reverberatory furnaces, as against sintering followed by smelting in blast furnaces." Sintering machines have not yet solved the problem of successfully handling the fine concentrates now produced, but the author believed that "the only solution seems to be the development of sintering in place of roasting, and blast-furnace smelting in place of reverberatory."

Zinc and Lead.

Zinc-Lead Concentration.—The practice at the 400-ton concentrator of the United States Smelting Company, located at Midvale, Utah, is described in the April *Mines and Minerals* by Mr. LEROY PALMER. The ore treated carries about 9 per cent lead, 8 per cent to 9 per cent zinc, 13 per cent to 14 per cent iron and some copper, gold and silver. Two products are made; one, a lead concentrate carrying gold, silver and copper, which is smelted, and the other, a zinc-iron concentrate, which is subsequently submitted to an electrostatic separation. In the scheme of wet concentration adopted, all the ore receives at least a preliminary treatment on jigs of the Hartz type, from which lead concentrates and zinc middlings are produced. Nevell rolls, Colorado Iron Works cone classifiers and Sherman classifying tanks treat the product sent to Wilfley and Overstrom concentrating tables and Sherman slimers. The products of this system are lead concentrate, zinc concentrate, zinc-iron middling and tailing. The zinc-iron middling is further treated to prepare it for electrostatic separation. It is elevated to a dewatering wheel which sends the dewatered product to a bin and the water to a Callow tank. The bin product is reground in a Denver Engineering Works tube mill, which is an innovation that is found very satisfactory. Silex linings and flint pebbles are used, the linings lasting nine months and the pebbles wearing at the rate of ¼ lb. per ton of ore treated. Frenier pumps elevate the ground pulp to a settling tank. The overflow of the settler goes to a Callow tank, the spigot discharge of which is treated on a Sherman slimer, while the overflow goes to waste. The spigot discharge of the settler goes to a six-compartment Richard's pulsator classifier, the different products being treated on Overstrom and Wilfley tables. The mill is electrically driven and the power consumption is 0.4 hp per ton of ore treated. Concentration is on the basis of 1.83 tons of crude ore to 1 ton of concentrate. Extraction is 92 per cent to 95 per cent of the zinc and lead, 85 per cent to 90 per cent gold and silver, and 80 per cent copper.

Silver-Lead Smelting in Tasmania has never been a financial success, according to Mr. THEODORE KAPP, who describes the operation of the Tasmania Smelting Company in the April 2, 1910, issue of the *Eng. and Min. Journ.* The difficulties lie in the treatment of the low-grade ores at a profit. High-grade lead ores are more profitably shipped direct to European works, as the difference in freight between the ore and the silver-lead bullion is not great enough to counterbalance the costs of treatment in Tasmania. The introduction of the Huntington-Heberlein process of roasting greatly facilitated the treatment of low-grade ores, and while the smelter has not yet returned dividends to the investors, it has worked to the great advantage of the miners.

The Tasmania plant has automatic weighing and sampling departments, 12 Huntington-Heberlein roasters, three lead blast furnaces of the American type, 42 in. x 120 in. at the tuyeres, of which two are regularly in blast.

Mercury.

Mercury Reduction as practised at the Guadalupe mine in Santa Clara County, California, is described in the *Min. and Sci. Press* for March 26, 1910, by Mr. COURTENAY DEKALB. The mine has been operated intermittently since 1855. Cinnabar occurs in serpentine and is mined through a 2000-ft. adit reaching a depth of 700 ft. The furnaces are of the Litchfield type. Oil is used as fuel and 63 per cent of the total mercury recovered is obtained direct from the condensers, the remainder being extracted from the soot. Two blocks of furnaces are used, one for coarse ore, averaging 3 in. in size, and the other for the undersize. The furnaces are tall distillation chambers with inclined tiles so placed within that the ore fed at the top slides down in a zig-zag manner to the base, where it is drawn off hourly into cars. Outside the distillation chambers are the fire chambers. Heated gases pass from the latter into the former and upward through the ore, finally passing out near the top. The ore chambers at the Guadalupe furnaces are 40 ft. high and 3 ft. x 6 ft. in horizontal section. The upper part is constricted to a "throat" which is entirely filled by the ore being fed to the furnace. A dryer, independently fired, is above this constriction. The condensers are tandem brick chambers 12 ft. x 40 ft. in plan and 20 ft. high. From these the fume passes through flues expanding into smaller condensing chambers at intervals. Oil consumption is 1 bbl. per 10 tons of ore treated. The soot is treated on concrete tables, where it is hoed to facilitate the separation of the mercury. It is then mixed with ore and again fed to the furnaces. Labor consists of three men per shift of eight hours.

Electrochemistry.

Induction Furnace.—Concerning a new type of induction furnace, due to Mr. OTTO FRICK, the following information is given by Dr. F. Mollwo Perkin in the Lond. *Electrician* of March 18: Frick has designed a new induction furnace which, it is reported, has been tested and approved by the Krupp works in Essen. The furnace is said to have a capacity of 10 tons. The primary winding surrounds one arm of a rectangular transformer iron core; the same arm is also surrounded by the annular hearth. The other limbs of the core do not, as is usual in ordinary transformers, closely include the primary and secondary circuits of the transformer furnace. They are arranged at such a distance from them that the leakage fields, when the furnace is working, do not pass in any essential degree through the iron. By combining the primary winding facing a horizontal side of the hearth and a magnetic core, it is possible to use a primary coil having a flat, dish form. A coil of this design has a high magnetic reluctance in the leakage field which surrounds it, consequently the self-induction of the furnace is reduced.

Carbide and Cyanamide.—In a review of progress in electrochemistry in 1909, in the Lond. *Electrician* of March 18, F. MOLLWO PERKIN makes the following remarks: One has not heard so much in reference to the advances in the manufacture of cyanamide, but it is understood that in a good many works plant is being put down to convert the cyanamide into ammonium sulphate. This is produced by blowing steam over the cyanamide, and passing the ammonia so produced into sulphuric acid. The conversion of cyanamide into ammonia would rather point to its not having been found so satisfactory a fertilizer as was originally hoped. As to calcium carbide it is stated that in Australia the use of carbide is increasing rapidly and whereas the imports about five years ago were not more than 2500 tons they are estimated to have been 10,000 tons during 1909. The carbide is not only employed for illuminating purposes, but to work sprayers for killing blight and other purposes. In this case the carbide is placed in a strong iron vessel, water is allowed to act upon it and the pressure so produced (up to 100 lb. per square inch) is used to drive the insecticide through a nozzle.

End of Electrolytic Gold Precipitation in South Africa.—The Lancaster West Company, Transvaal, South Africa, has just completed new extractor-boxes for the zinc precipitation process, and the change from the Siemens-Halske system will be made immediately. According to the March, 1910, issue of *The Mining Magazine* (London), it is believed that this transition represents the final abandonment of electrolytic precipitation on the Rand. For some years before the war the process

had keen support, and many of the journals of the scientific societies indicate the great weight of opinion in its favor, owing to the saving in cyanide and the fineness of the bullion produced, even though it cannot be said to have ever gained a position of equal popularity with the zinc method. Under present conditions, with the improved efficiency of zinc precipitation and clean-up arrangements, the Siemens plant has had no chance of support, except in the case of the Lancaster, which until recently feared the capital expenditure involved in the alterations.

Rapid Electro-analysis with Stationary Electrodes was suggested about a year ago by Prof. J. T. Stoddard, of Smith College. The results obtained aroused considerable interest and some criticism, and caused a number of other investigators to look into the possibilities of Stoddard's method. In the March 31, 1910, issue of the *Jour. Soc. Chem. Ind.*, Dr. T. Slater Price and T. C. Humphreys present the results of their attempts to repeat Stoddard's work. Their procedure was as follows: In each case the amount of metal in a given volume of solution was determined electrolytically, using a rotating cathode, by a method which was known to give quantitative results. The analysis was then carried out according to Stoddard, using stationary electrodes and a current strong enough to heat the solution and increase the circulation effect produced by the gas bubbles coming off the anode. The electrolytes used were copper sulphate, nickel ammonium sulphate, silver nitrate, cadmium sulphate, zinc sulphate. The authors were unable to secure complete depositions in the period of time claimed by Stoddard, and came to the conclusion that for rapid electro-analysis stationary electrodes are not so reliable and satisfactory as rotating.

Those of our readers who are concerned with electro-analysis, especially in commercial practice, will be interested to compare these conclusions with those expressed in our correspondence columns in this issue by Allen P. Ford, metallurgist for the Crane Valve Company, Bridgeport, Conn., who gives practical conclusions favorable to the stationary electrodes of Stoddard.

Plating upon Aluminium.—The following process of E. C. Szarvasy for plating upon aluminium is mentioned by F. M. Perkin in the Lond. *Electrician* of March 18. The bath used is anhydrous methyl-alcohol with the copper or nickel salt dissolved in it. The surface of the aluminium is cleaned with a metal polish containing grease, the object of the grease being to protect the surface of the aluminium from atmospheric oxidation. The metal is then placed in the bath, the alcohol dissolves off the fat and the metal is then deposited in an adherent form on the aluminium.

Recent Metallurgical and Chemical Patents

Iron and Steel.

Alloy Steels.—Mr. Samuel S. Wales, of Munhall, Pa., patents an alloy steel which is claimed to have increased strength, great resistance to shocks and a high tensile strength. It contains carbon, manganese; nickel below 4 per cent; chromium, 0.5 per cent to 2 per cent; vanadium, 0.1 per cent to 0.5 per cent; titanium, 0.05 per cent to 1 per cent. (952,391, March 15, 1910.)

Another alloy steel patented by Mr. Wales and especially suitable for armor plates, with or without a carburized face, contains: carbon; manganese; nickel below 4 per cent; chromium below 2 per cent; vanadium 0.1 per cent to 0.5 per cent; copper less than 1 per cent. The addition of the copper within the limits named increases the tensile strength of the metal and increases the reduction in the area when tested to destruction. Both these patents are assigned to the Carnegie Steel Company. (952,392, March 15, 1910.)

Boron Steel for Transformer Sheets.—Dr. W. R. Whitney, of the General Electric Company, has found that the addition of the elements boron or beryllium increases the resisitvity of iron to a high degree, at the same time improving its mechanical properties and making a valuable steel for electric transformer sheets. The increased resistance causes, of course, a reduction of the eddy-current losses. The proportion of boron in the high-resistance alloy may vary from about 0.2 per cent to 5 per cent boron, but for most purposes the lower limit of boron content is preferable. The steel should be as free as possible from sulphur, phosphorus or other ingredients. The following suggestions of Dr. Whitney as to the rationale of the process are interesting. He believes that the resistivity of iron is increased by the presence of alloying elements to a degree inversely to their atomic wights, because of the increased number of molecules furnished by an element of low atomic weight per unit weight of alloy. In other words, a given weight of alloy, containing given percentages of iron and alloying element, will have a higher resistance if the atomic weight of the alloying element is low, as a given percentage of the element introduces a relatively greater number of molecules than the same percentage of an element of higher atomic weight. (952,299, March 15, 1910.)

Special Steel.—Mr Charles R Bryson, of Pittsburgh, Pa., patents a steel containing "chromium, 6 per cent; nickel, 3 per cent, tungsten, 0.5 per cent; silicon, 0.3 per cent, carbon, 0.6 per cent; iron, 89.177 per cent; phosphorus, 0.014 per cent; manganese, 0.4 per cent; sulphur, 0.009 per cent; vanadium, trace." The following properties are claimed for it: Excessive hardness, combined with toughness, and capability of resisting abrasion when brought into repeated contact with white or red-hot iron or steel. Also the peculiar property of non-cracking or fracturing when at a red heat and suddenly quenched in cold water, and furthermore, the property of non-deterioration when coming into repeated contact with white or red-hot iron or steel. For these reasons the steel is claimed to be "especially adaptable for the manufacture of butt or lap weld or seamless tubing, which art necessitates the use of balls, bell rings and stretcher bars, adjuncts commonly used in the production of the several methods of making metal tubes." (952,069, March 15, 1910.)

Cupola.—The construction of a cupola which is intended not only for melting, but for refining, is patented by Mr. Henry Johnson, of Saxilby, England. The cupola, which may be of the ordinary type with or without a receiver, is formed with a bath at the bottom capable of holding a suitable quantity of molten metal. A set of high-pressure blast twyers are provided in the side walls at the bottom of the bath, connected to a suitable air compressor or pressure blower, for the purpose of refining the metal. The bottom of the bath may be formed with a tapping hole through which the metal remaining at the end of a blow may be run off. A set of twyers are provided at a higher level than the blast twyers and are connected to a suitable fan or blower, these low-pressure twyers being approximately at the same level as the overflow, and being used to mix air or steam with the burning gases and support the combustion. (952,260, March 15, 1910.)

Combination Steel Process.—A new combination process, terminating in electric furnace treatment, has been devised by Mr R. H. Wolff. The only claim of his patent refers to "the charging of a dephosphorized molten steel from a basic converter or open hearth furnace on top of a basic slag, for the purpose of desulphurizing and deoxidizing the metal thoroughly and rapidly and then recharging this metal promptly in an electric furnace." The treatment in the electric furnace is made simply for the purpose of finishing and resting the metal, so that any gas or slag left in the metal may rise to the top. (946,302, Jan 11, 1910.)

Gold and Silver.

Pulp Agitator.—In order to keep the particles in suspension in cyanide or other similar solution, R. W. GORDON, of Denver, Colo., has recently patented a novel device. The machine, as shown in Fig. 1, comprises a tank, in which a revolving conveyor raises the settled particles from the bottom of the tank discharging them at the top where they again settle to the bottom. The settling sands are caused to gather at a certain place on account of the construction of the lower wall of the tank and the insertion of a cone in the bottom of the tank. In operation the ore pulp is fed in any suitable manner. The rotation of the conveyor constantly elevates the settled particles of ore and discharges them at the top. They then settle both at the center of the tank, within the conveyor frame, and

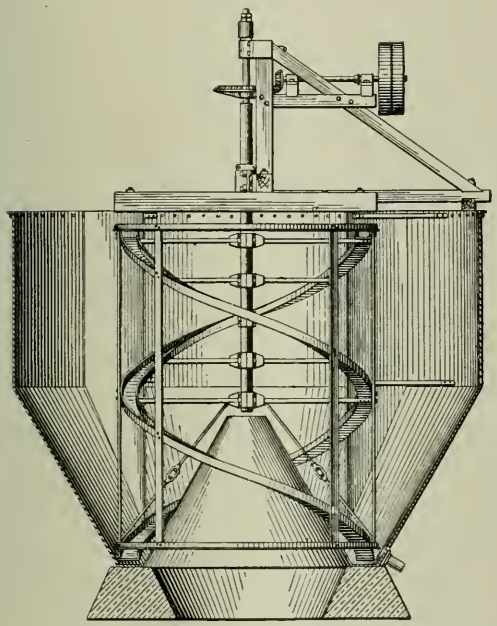


FIG. 1.—PULP AGITATOR.

in the outer part of the tank. This gives rise to three vertical currents, one upward in the path of the conveyor and two downward on each side. The rotation of the body of the pulp solution is prevented by baffles. The device is claimed to be particularly adapted for use ahead of filters, as it will deliver a homogeneous feed either continuously or intermittently. (954,511, 1910.)

Gold and Silver Extraction.—The mechanical treatment of slimes is the subject of a patent granted to A. F. CROSSE, of Johannesburg, Transvaal, South Africa. The principal features of the process were described in this journal for February, 1910, page 72. In brief, the process consists in treating the slimes with potassium cyanide solution in a cone-shaped agitator, obtaining a concentrated pulp underflow and a clear overflow liquid which is conducted directly to zinc boxes for precipitation, after which it is returned to be remixed with the pulp until practically the whole of the precious metal content is abstracted. The invention seeks to reduce the time required and to eliminate the necessity for filter presses. (951,940, March 15, 1910.)

Agitating Apparatus.—In order to thoroughly agitate sands or slimes containing gold and silver in order that they may be thoroughly leached by cyanide or other solvent, R. S. TOWNE, of New York, has patented an apparatus for agitating and aerating that class of material. The pulp is fed into the top of a leaching tank having a cone-shaped bottom and is circulated through a central shaft by means of a jet of air which carries the metal from the bottom of the tank upward through the circulation shaft from which it is discharged back into the tank. The circulation shaft has apertures at various heights which can be closed or opened as may seem desirable. Several different forms of tank are described in which the circulation shaft may be either within or without the main tank. (952,222, March 15, 1910.)

Zinc.

Improvement in Smelting Furnaces.—In order to avoid waste of ore at the time the smelting furnace is charged, C. A. WETTENGEL, of Caney, Kan., has patented a metal apron to be attached in front of the retorts in such a way that any of the fine ore which might otherwise go to waste when the retorts are charged is collected in such a manner that it is readily saved. A portion of the apron is removable so that the ash chute may be opened at the time the furnaces are discharged. (953,405, March 29, 1910.)

Tin.

Detinning Tin Scrap.—The application of a freezing process for detinning tin scrap has been made the basis of a patent recently granted to WALTER J. PHELPS, of Baltimore, Md. It has been found that when tin-coated scraps are subjected to extremely low temperature the tin has a tendency to separate from the steel or other metal coated by it. The apparatus which the inventor uses in accomplishing this end is an insulated drum in which rotates a perforated cylinder carrying the tin scrap. The axle or shaft of the cylinder is hollow and communicates with a number of nozzles placed along its length through which liquid air or some similar material is conducted. As the drum revolves and the liquid air acts on the tin the small particles fall through the perforations of the drum and are collected below. (952,351, March 15, 1910.)

Detinning.—Another novel detinning process, invented by Mr. CHARLES J. REED, of Philadelphia, consists of the following steps: The charge of tin scrap (tinned iron sheet scrap) is heated in a revolving drum in contact with particles of magnetite in a finely divided condition. When the charge is heated up to about the melting point of tin, the magnetic particles wipe off the fused or softened tin in pulverized form. Either simultaneously with this operation or subsequently in a second step the pulverized tin is removed from the scrap by means of a current air and is collected in a settling chamber or in textile bags. In the last step the residual mass is screened so as to separate the magnetite particles from the steel scrap. The distinction of this process from other detinning processes is that it is entirely mechanical. (950,115, Feb. 22, 1910.)

Electric Furnaces.

Radiation Resistor Furnace.—A batch of interesting patents has been granted, partly to Mr. JOHN THOMSON, of New York City, partly to Mr. F. A. J. FITZGERALD, of Niagara Falls, and partly to both jointly for a novel type of resistance furnace. All these patents are assigned to the Imbert Process Company, of New York. The common idea underlying all the designs is to heat the furnace charge by radiation from a resistor, which is generally placed above the charge.

The design of the resistor, which is made of carbon, varies in the different furnace designs, but in all cases great care is taken to counteract such deformations of the resistor (due to heating and cooling) which would change the uniformity of generation of heat in the resistor. For instance, if the resistor is an arch built up of wedge-shaped carbon blocks originally in uniform contact, then upon being heated, the arch would expand and would rise along its crown and the wedge-shaped carbon blocks would open at the upper surface of the arch and

the current would be confined to the under surface of the arch. To obviate this difficulty Thomson forms the wedge-shaped blocks so that when primarily set the lower portions of their faces are not in physical contact. Hence, when the arch is heated and expands and rises, the lower portion of the blocks will be forced into physical contact and the entire mass of the resistor will be in approximate balance electrically. (950,882, March 1, 1910.)

Another design of Mr. Thomson covers a resistor in form of a flat arch composed of wedge blocks. (950,881, March 1, 1910.)

In another design of Mr. Thomson, the resistor, in form of a horizontal plate, is built up of wedge-shaped sections, alternately reversed, so that on heating a direct right- and left-hand thrust due to expansion is produced and presses the upper sections upward, while on cooling and recontraction they fall down again. In general, the resistor plates are arranged in form of a loop or a series of loops folded back and forth and generally connected in direct circuit. This makes series-parallel connection of the different parts of the loop easy. (950,877, March 1.)

In another design of Mr. Thomson each section of the resistor consists of a plurality of grooved plates. When such plates are laid together, with the rounds of one plate in the hollows of another, the contacts will be in lines, and when a series of such plates are set between terminals, the several parts are interlocked and the resistor as a whole is self-sustaining, its weight and end-thrust being taken wholly by the terminals. Any side of such a resistor may readily be given greater compression than the opposite side, thereby correspondingly increasing the current density and temperature. (950,879, March 1.)

Another patent of Mr. Thomson refers to the application of the last named principle for the construction of a resistor rod in the center of a revolving furnace. (950,880, March 1.)

In another construction of Messrs. Thomson and FitzGerald the resistor is built up of round carbon rods, resting on a layer of refractory brick. The carbon rods are arranged in various layers. Since the mechanical pressure is strongest in the lowest layer, which rests on the refractory brick, there would be a danger of excessive heat along this layer. It is, therefore, advantageous to insulate the ends of the rods from each other where they rest on the refractories so that no current will flow through those parts of the rods, and hence no heat will be generated there. The chief advantage of the use of carbon rods is their low cost. (951,086, March 1.)

Another interesting resistor construction of Messrs. Thomson and FitzGerald is in form of a central cylindrical rod for a revolving furnace. The resistor consists of a cylindrical casing of recrystallized carborundum (a strong and very dense structure which may be heated to about 3600° Fahr. without damage, but is an excellent conductor of heat). Within this cylindrical carborundum casing granular or powdered carbon is provided, which represents the resistor proper. (950,878, March 1, 1910.)

To prevent the destruction of the highly-heated carbon resistor by the oxygen in the atmosphere Mr. FitzGerald proposes to introduce producer gas on account of its reducing nature into the chamber around the resistor. (950,905, March 1.)

This principle may be employed to construct a furnace which is partly heated by fuel and partly by electricity. The resistor consists of carbon rods resting upon a layer of refractory brick which forms the top of the melting chamber. The fuel gas is introduced into the upper chamber containing the resistors. The fuel gas is then mixed with air for combustion, the flame gases passing through the furnace in zigzag form to get a high thermal efficiency. Whenever the economical apex of the fuel heat is reached, the electric current is switched on to the primarily heated resistor and the higher range of temperatures is obtained from electrical energy. The fuel heating may also be made to reduce the loss of heat, produced from electrical energy. (950,904, March 1.)

An illustrated article from the FitzGerald and Bennie Laboratories, on this furnace and its use in zinc metallurgy, is reserved for our next issue.

Alumina Abrasives and Refractories.—The "artificial emery" process—first essentially a simple fusion of bauxite in the electric furnace—has undergone an interesting development in recent years in the hands of the Norton Company. Further evidence of this evolution is a batch of patents granted to Mr. Lewis E. Saunders, of Niagara Falls, N. Y., and to Mr. George N. Jeppson, of Worcester, Mass., all being assigned to the Norton Company. All the patents relate to special effects which the addition of certain impurities has on pure alumina.

A material consisting of alumina and titanium oxide (for instance, 94 per cent Al_2O_3 and 6 TiO_2, obtained by fusing in an electric furnace a mixture of alumina and rutile) is, according to Mr. Saunders, exceedingly hard, tough and massive, and of a granular or crystalline structure and has abrasive qualities of a high order. Its melting point is rather low compared with that of pure alumina. (954,767, April 12, 1910.)

A composition of alumina, silica and titanium oxide, free from iron, is, according to Mr. Saunders, useful as an abrasive and as a refractory material for the production of cast or molded articles. For instance, a product containing 92.5 per cent alumina, 3 silica and 4.5 titanium oxide, is very hard and tough, but is quite fusible as compared with pure alumina and is therefore more easily cast. It is brown in fracture and color and has a marked massive or adamantine character and excellent abrasive qualities. (954,768, April 12, 1910.)

A composition of alumina, titanium oxide and ferric oxide (containing, for instance, 1 or 2 per cent of titanium oxide and at least 2 to 4 per cent iron oxide, but no silica) is, according to Mr. Saunders, of blue or blue-black color; the crushed and graded abrasive has superior holding qualities for glue, etc., and is therefore particularly adapted for the preparation of abrasive paper or cloth; it is also particularly suited in the qualities of temper and hardness to the class of work to which abrasive paper or cloth is applied, as, for example, finishing leather and certain classes of metal work, more particularly disk grinding. The method of producing this material is both interesting and simple. In the old process, where the object is to reduce the impurities, bauxite is calcined to expel the water of hydration, and the material is then fused in an electric furnace between carbon electrodes, with the addition of carbon to the charge for the reduction of the oxides of iron, silicon and titanium. In the new process of Mr. Saunders commercial bauxite, containing both iron and titanium, is ground and calcined, and is then melted in an electric furnace with precaution to avoid strongly reducing conditions, that is to say, no carbon is added to the charge, and the charge is melted as rapidly as practicable between graphite electrodes. (954,766, April 12, 1910.)

Mr. Jeppson has found that the reduced impurities from the bauxite may be made to act as a bond for the refractory particles. Thus iron oxide and silica in bauxite will be reduced to ferro silicon and at a high temperature under oxidizing conditions this yields ferrous or ferric silicate or both, which are highly efficient bonding agents when properly disseminated through the mass. Mr. Jeppson prefers to incorporate with the product containing the reduced impurities a small proportion of another bonding agent, the function of which is to dissolve or flux and thereby disseminate the oxidation products of the reduced impurities, throughout the mass to be bonded. The procedure is as follows: Commercial bauxite is calcined in a rotary kiln and then melted in an electric furnace between carbon electrodes, with the addition of 5 per cent of carbon, so as to reduce the compounds of iron, silicon and titanium. The product is solidified, crushed and the nodules and detached particles of ferrosilicon are removed by hand-picking followed by magnetic separation. The material is then subjected to an oxidizing roast for the purpose of converting the remaining reduced impurities into oxidized products and is then separated into the required number of grades and sizes. The resulting

aluminous material is superficially coated with the oxidation products produced during the roasting operation, and is then mixed with water and 1 per cent of fire clay or high-melting point ceramic bond. The mass is molded or formed into refractory bricks or other desired shapes and fired in the usual way. The product is strong and extremely refractory to heat. (954,809, April 12, 1910.)

An abrasive for special uses such as the internal grinding of articles of steel, iron or the like is made, according to another process of Jeppson and Saunders, as follows: Alumina substantially free from oxides of iron, silicon, or titanium, is melted in an electric furnace between graphite electrodes, the charge being fed more rapidly than is the practice with impure alumina. It results from this relatively rapid feeding and from the use of graphite electrodes as distinguished from ordinary carbon, that the product undergoes but little reduction, possesses an essentially crystalline structure and is not contaminated with large amounts of aluminium carbide, the effect of the presence of which is to cause the mass to disintegrate on long exposure. The resulting pig or mass, usually nearly or quite white in color, and usually of a very marked crystalline structure, is broken up, crushed and subjected to an oxidizing roast to remove traces of carbide and to improve its bonding qualities. The particles are then graded and molded into the desired articles or implements by means of an appropriate bond, preferably ceramic bond. A suitable bond is a mixture of one part of ball clay with one part of feldspar, four ounces of this mixture being user for each pound of abrasive grains. Wheels so bonded possess the characteristic that under conditions of use the abrasive grains fracture in the bond, thus always presenting fresh and keen cutting points and edges to the work (since the grinding process is essentially a cutting process.) In this respect the wheel differs radically from those heretofore prepared from impure or partially purified bauxite or other commercial forms of alumina, the grains of which become dulled and glazed, and then break out from the bond without fracturing to any important extent. The bond is carefully prepared to admit of this effect, without which the efficiency of the wheel would be greatly diminished, since the work would be burned and the wheel would require frequent renewal of its cutting face by dressing. (954,808, April 12, 1910.)

Induction Furnace.—An induction furnace of the "disk-transformer" type is patented by Mr. C. W. SÖDERBERG, of Ski, near Christiania, Norway. The arrangement is shown in Fig. 2 in horizontal and vertical cross-section. The primary windings are distributed on four coils as a_1 a_2, inclosing both of the vertical parts b_1 and b_2 of the iron core. The secondary armature consists of a metal bath m_1 m_2, contained in a hearth m arranged between the primary coils and formed with an intermediate portion, and two semi-annular channels e_1 e_2, which together with the intermediate portion inclose both of the vertical pieces of the iron core. By the arrangement of four coils the leakage of the power lines is cut down. The lower primary coils are cooled by currents of air which pass through channels d_1 d_2 in the foundation of the furnace. (954,827, April 12, 1910.)

Induction Furnace with Electrodes.—A modified induction furnace for reduction processes is patented by Mr. K. A. F. HIORTH, of Christiania, Norway. The induction furnace is of the same construction as usual, with a ring-formed channel as secondary for the charge. The essential novel feature is the possibility of breaking the continuity of the ring channel, by inserting at some point a dividing wall. To get electric connection between the two sides of the wall, electrodes are placed at the two sides. They may be connected together either directly or through an auxiliary secondary coil wound around the magnetic iron core. The necessity of special electrodes is avoided if the dividing wall reaches only to the top of the molten metal bath proper, the electric connection being obtained by the slag, above the dividing wall. (954,804, April 12, 1910.)

Protection of Electrodes.—To protect those parts of a carbon electrode which are not embedded in the charge from burning or wasting away, Mr. H. C. HARRISON provides them with a protective coating or envelope. He dips the electrode in a bath containing some such material as common salt or fluor-spar, which has itself a comparatively low melting point, and

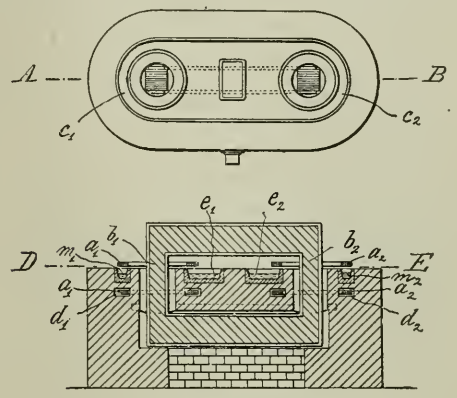


FIG. 2.—INDUCTION FURNACE.

which when in use forms a compound inert at the temperature to which it is to be subjected either as an oxidizing or as a reducing agent. If the electrode itself is made hot and dipped into a molten bath of common salt, for example, kept well fluid considerably above its melting point, after a short interval the electrode may be withdrawn and nearly all the superfluous salt will flow off leaving a glaze over the whole surface, which penetrates into the body of the electrode itself, the distance depending, among other things, on the porosity of the electrode, the length of the immersion and the fluidity of the bath. Another way is to dip the electrode into an aqueous solution of calcium chloride $CaCl_2$, which, after being heated in the air, forms, with the impurities present, a slag or coating consisting largely of CaO, and which will last as long as the electrode lasts. The patent is assigned to the Electro Metallurgical Company. (951,990, March 15, 1910.)

Vacuum Resistance Furnace.—In a vacuum furnace for exceedingly high temperatures, designed by Dr. A. O. APPELBERG, the principal feature is the use of a heating resistor of tungsten. This may be pressed into the desired form from finely powdered tungsten, but for convenience in manufacture it is desirable to add a small quantity of binding material to the tungsten powder and then squirt or mold the mixture into rods and blocks necessary for the construction of the furnace. A 50 per cent glucose solution is a suitable binder and may be added to the tungsten powder in the proportion of about five parts by weight. When glucose is used about 10 per cent of tungstic oxide is added to the mixture, so that when the mixture is heated up in the furnace there is enough oxygen present to eliminate all carbon and yield a refractory body of pure tungsten. This mixture when first molded or pressed is rather brittle and inclined to crumble, but by heating it in a vacuum to about 1300° C. it is rendered strong and relatively tough. After this treatment it can be machined and otherwise manipulated much like artificial graphite. Before the furnace is put into regular operation it is advisable to heat these tungsten parts to a temperature considerably higher than 1300° C. in order to drive out all vaporizable components, such as unreduced tungstic oxide, but this treatment can be conveniently carried out in the furnace itself after the latter has been assembled. A series of insulated superimposed rings of tungsten is used as a shield to prevent dissipations of heat toward the metal walls of the chamber and to reflect radiant heat on the crucible located

within the heating space. Since the entire vacuum chamber is free from carbon, very high temperatures may be produced without danger of contaminating the charge with carbon or carbides. The patent is assigned to the General Electric Company. (953,774, April 5, 1910.)

Amorphous Tungsten Powder.—The use of tungsten in the manufacture of electric lamp filaments has given rise to a great number of patents bearing on the production of tungsten in a suitable form. A recent process of Johannes Schilling, of Colonie Grunewald, near Berlin, Germany, depends on the tendency of tungsten compounds to break up into finely divided particles on being highly heated in the presence of ammonia. The resultant powder may be readily mixed with suitable binding agents and is designed to form a stronger filament. The inventor uses preferably meta—or para—tungstates of ammonium. The salt is heated in a closed tube to a temperature slightly above that of the disintegration of the salt. Hydrogen and ammonia gas are simultaneously led through the retort to produce a reducing atmosphere. The ammonia driven out of the salt is either wholly or partly dissociated and the nascent hydrogen acts as a very strong reducer and reduces the oxides of tungsten to the elementary substance. (950,869, March 1,

Electrolytic Processes.

Cleaning Articles.—Mr. Howard I. Manley patents the following method of cleaning metallic articles. He uses a receptacle of tin containing a solution of 97 per cent of sodium bicarbonate and 3 per cent of salt in the proportion of 1 oz. of this mixture to 1 gal. of water, heated to not less than 212° Fahr. The articles to be cleaned are introduced into the bath and are brought into direct contact with the tin surface. The cleaning of the article is effected "by a sort of electrolytic action which takes place on immersion thereof in the bath, the foreign matter which ordinarily tarnishes metal articles being precipitated to the tin surface." This explanation may hold for the cleaning of gold, solid and plated silver, nickel-plated and brass articles, for which the method is stated to be suitable. But it hardly explains the cleaning of cut-glass, glass, tile, etc., for which the method is also said to be applicable. (951,824, March 15, 1910.)

Mercury Cathode Cell for Sodium Chloride Electrolysis.—In our Vol. VI, p. 159, we noticed a patent of Mr. Jasper Whiting, of Boston, Mass., for a mercury cathode cell, in which there is no continuous circulation of the mercury or amalgam cathode, but the cathodic mercury is maintained in a quiescent state during the deposition of the sodium, but is transferred at regular intervals to an oxidizing compartment. Some modifications of this process and apparatus have now been patented by Mr. Whiting, but since the inventor is to present a paper on his process at the coming meeting of the American Electrochemical Society, we defer our report till then. (951,228 and 951,229, March 8.)

Electric Discharges Through Gases.

Producing Hydrocyanic Acid.—It is well known that hydrocyanic acid can be produced by passing a mixture of nitrogen and hydrogen over highly heated carbon. The apparatus for its production was described in our issue for February, 1910. (Vol. VIII, No. 2, page 10.) It can be appreciated, however, than an industrial application of the process would necessitate a cheap and simple method of producing the necessary gases, since the production of pure nitrogen and hydrogen would be too expensive. A solution of the problem is proposed in a process devised and patented by Dr Otto Diffenbach and Mr. Wilhelm Moldenhauer, in which they use Dowson gas, or producer gas, in their apparatus referred to above. This gas may have the following composition.

CO	27.6 vol. per cent
H	15.3 vol. per cent.
CH_4	1.4 vol per cent
CO_2	3.9 vol. per cent
N	51.8 vol. per cent.

When blown into the electric arc furnace with a velocity of 100-500 liters per kw-hour, a considerable concentrated HCN gas is produced. In passing a cooling device and absorption apparatus the HCN is removed and the waste gases are led to the gas container and may be used again. The expression "Dowson gas" is used to refer to a gas produced according to the formula

$$C + O = CO$$
$$C + H_2O = CO + H_2$$

starting from air and steam passed over the heated carbon. (954,080, April 5, 1910.)

Fixation of Atmospheric Nitrogen.—A new process devised by Mr. Henry Howard, of Boston, Mass., for producing nitrogen oxides from air by arc discharges, is particularly interesting because it fulfils very well the two conditions which should be of fundamental importance if the purely thermal theory of oxidation of atmospheric nitrogen is correct: (1) the use of as high a temperature as possible for oxidizing the nitrogen, and (2) as quick a cooling as possible of the formed nitrogen oxides to prevent their dissociation. According to Mr. Howard's process the air is compressed, and is then subjected to the action of an electric arc which causes a great increase in pressure; it is then allowed to expand while doing sufficient work to absorb as much as practicable of its heat-energy, whereby rapid cooling is secured; this cycle is then repeated. In practice, the energy of the expanding gas will preferably be utilized for useful purposes, as for generating a portion of the electrical energy required for heating the gas, effecting a corresponding saving in the outside power required. The principle may be applied in various ways, as, for instance, in a machine operating on the principle of an internal-combustion engine, either of the four-cycle or two-cycle type. In the former case the first stroke of the piston serves to draw the gaseous mixture into the cylinder, and the second to compress it therein at about the moment of maximum compression or shortly before it is subjected to the heating effect of a powerful electric arc whereby its pressure is greatly increased. During the following or power stroke the heat-energy of the expanding gas is converted to a large extent into mechanical work with coincident chilling of the gas, further quantities of heat being dissipated by other means if desired. The final stroke of the cycle ejects the oxidized gas. In the two-cycle type the oxidized products are ejected and the fresh mixture admitted in the course of a single stroke. The principle may also be applied in a continuous manner by first compressing the gaseous mixture, and then passing it through a combustion chamber wherein it is strongly heated by a continuous or substantially continuous arc discharge, thereby increasing the volume of the mixture; the hot gases are then allowed to expand and do work as, for instance, by passing them through a turbine, the condition for the effective accomplishment of the reaction being that the expansion of the entire body of highly heated oxidized product shall follow the reaction as to minimize the dissociation thereof. (952,248, March 15, 1910.)

Ozone Generator.—In an ozonizer of Mr. Frank M. Ashley, of New York, the electrodes between which the silent brush discharge takes place consist each of a number of bell-shaped metallic vessels, inserted one within the other, the oxygen or air to be ozonized being introduced through an axial opening in the central bell of one of the two electrodes. The rims of these two sets of bells are opposite each other, and the brush discharge passes from each rim to the opposite rim. (951,789, March 15, 1910.)

Welding.

Copper Solder.—As a solder or brazing material for connecting the confronting ends of two pieces of copper, for instance, of a trolley wire, a composition consisting of 38 gr of copper and 35 gr of borax is patented by Messrs. A. Vastag and S. Horvath, of McKees Rocks, Pa. (952,389, March 15, 1910.)

Aquadag as a Lubricant.

The readers of this journal are familiar with the merits of oildag as a lubricant, but up to this time little has been said about aquadag. Both oildag and aquadag are inventions of Dr. Edward G. Acheson and to him the world is indebted for these superior lubricants. Both contain deflocculated Acheson graphite, which remains in suspension. Oildag continues to win very pronounced favor, and it is evident that it is a general lubricant of great value.

Those who have tested aquadag as a lubricant for thread-cutting and machine-shop work in general have already come to appreciate its value. In Fig. 1 two pieces of pipe are shown. The particular machine on which these threads were cut was feeding with a very poor die. The factory had no other, neither could it procure one soon enough to fill a special order, so they were trying to do their best with a worn-out die, hoping at least to cut a sufficient number of end pieces to fill the order.

The piece of pipe at the left shows the kind of threads they were cutting with lard oil as a cooling lubricant. About 11 out of 12 pieces stripped as per illustration, and this despite the extra precaution that was taken to cut the thread by two passages of the die, instead of one. While in the midst of this work, they took all the lard oil out of the machine and put

FIG. 1.—COMPARISON OF THE USE OF LARD OIL (AT THE LEFT) AND AQUADAG (AT THE RIGHT) AS A LUBRICANT FOR THREAD CUTTING

in aquadag. They were astonished as well as gratified to find the die cut a wonderfully good thread with the aquadag as a cooling compound, and this, moreover, was done with one passage of the die, instead of two, as they were doing with the lard oil. The thread cut with aquadag as a lubricant is shown at the right. The threads cut with aquadag are cleaner than with lard oil, and much more definitely shaped. To use the foreman's words, "the threads cut are as good as those on a tap die."

The Niagara Machine Company, of Niagara Falls, N. Y., is another concern that has tested aquadag and determined its merits. Some weeks ago this company was cutting off in the lathe a large number of cold-rolled steel rods of average diameter about 1 in. Records showed that the average number of cuts the cutting-off tool would do, with one sharpening, was 60 when using an ordinary soap compound.

Having been asked to try deflocculated graphite in the form of aquadag, they mixed a small amount of it with their cutting compound and were greatly astonished, as well as gratified, to find that the tool did not blunt nearly so rapidly as before; in fact, the cutting-off tool actually made 980 cuts with only one sharpening. Fig. 2 is a reproduction of the 60 cuts made representing the maximum cutting done without aquadag, while Fig. 3 represents the actual 980 cuts made by the same tool in the same machine, the only difference being the addition of deflocculated graphite to the compound. It was observed that not only did the tool in the second case do about 16 times as much work as in the first instance, but also that the ends were very much smoother and the work was much better.

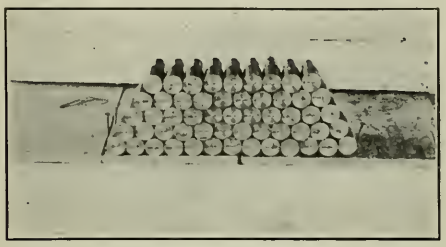


FIG. 2—RESULTS WITHOUT AQUADAG.

Still another point of merit associated with aquadag for machine-tool operation is that in boring and reaming a large number of holes in bronze bushings, it has been found that with ordinary cutting compound the hole is always about 0.0002 of an inch smaller than the reamer used, but through the addition of aquadag to this compound it has been found with a hole practically the full size of the reamer is cut.

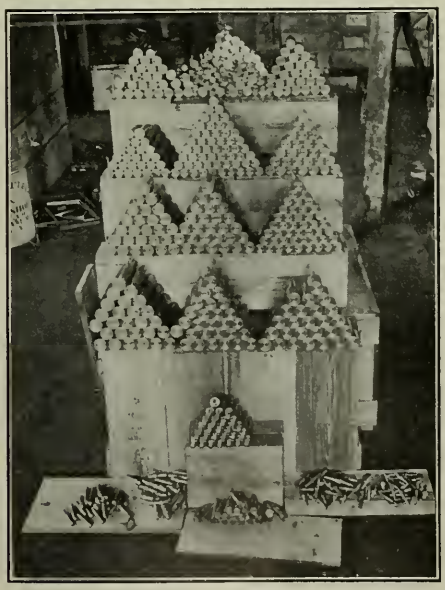


FIG. 3.—RESULTS WITH AQUADAG.

The general effect noticed in the use of aquadag in machine shop work is that it seems to afford considerable reduction in horse-power consumed, while there is considerable saving in the life of tools, and the work turned out is superior. Aquadag has been tried in practically every machine operation, namely:

lathe threading, lathe turning, die threading, boring, reaming, cutting off, planing, etc.

There are yet several points to be worked out in connection with the percentage of graphite to be used for the different purposes; whether it shall be mixed with compounds already on the market, or whether a separate aquadag solution shall be employed. All these questions are receiving careful consideration.

It might be added that aquadag has a wonderful field of usefulness before it as a lubricant for air compressors. Present practice is to use oil, the inflammable vapors of which cause serious explosions. The Refrigeration Committee of the National Fire Protection Association recently made this report:

"It is recommended that wherever in any system graphite can be used as a lubricant in compression cylinders, that it be done. This especially applies to air compressors used in dense air systems and for air compression for purposes other than refrigeration, as the temperature of the compression cylinders undoubtedly rises in some air systems to a point where inflammable vapors may be generated from lubrication oil."

Thus, a surprisingly wide field of usefulness opens before aquadag for the use suggested, and there can be no question but what it will afford high efficiency, undoubtedly saving many lives which might be lost through the use of more dangerous lubricants.

As the world comes to know the remarkable value of these lubricants containing deflocculated Acheson graphite, the greater must be the appreciation of the research work which resulted in their invention.

A New Hot Blast Stove.

There is not a regenerative stove on the market that does not involve special shaped brick in its construction, and some designs even go so far as to build entire linings of special brick, not one of which is listed in a brick manufacturer's catalog.

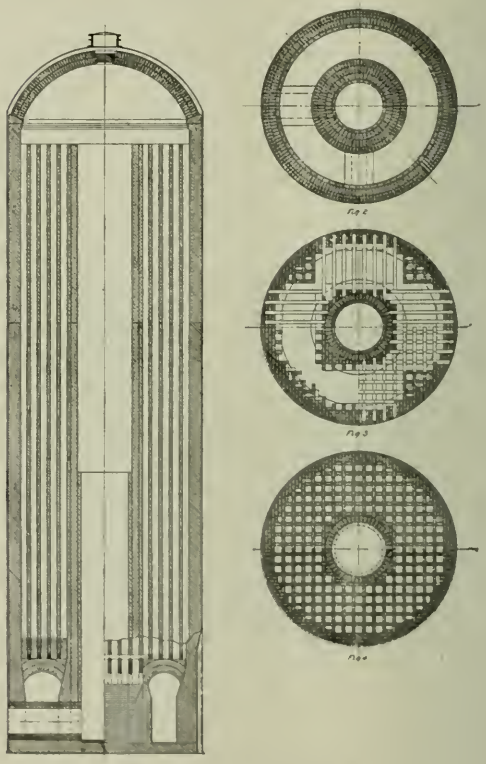
In order to build linings for stoves of this design it is frequently necessary for the brick manufacturer to make details of each particular brick and each brick must take its respective position in the lining or there is a misfit.

To manufacture special shapes other than are listed in brick manufacturers' catalogs, the manufacturer makes an extra charge on account of making new molds, extra labor and the time his drying floors and kilns are occupied for such special work. Delivery is also affected where special shapes are used, as they are very seldom found in the manufacturer's stock.

It is a well-known fact that in the manufacture of firebrick of standard size and shape, better results as to quality are obtained than is possible in the manufacture of brick of special shape larger in size than a 9-in. straight brick. The reason is that the manufacturer's equipment is best adapted to the making of standard shapes, and his employees are familiar with their manufacture, and that it is possible to make standard brick more accurately to dimension and to burn the smaller sizes of brick more uniformly throughout their entire section than is possible with a large or crooked special shape.

After a stove of this type has been in use for some time, and certain parts wear out by continuous use, it is always necessary to obtain duplicates of the special brick originally used, and this frequently necessitates considerable delay to obtain this material.

Mr. Alfred C. Nelson, having experienced considerable trouble in the building of stove linings where a large number of specials are used in their construction, has recently constructed a stove lining, using only such brick as are listed in the brick manufacturers' catalogs, with the exception of the dome plug, the tile supporting the checker, and the dome skew back. The arches shown in Fig. 1 were spanned in the earlier installation of this stove by using two 13½-in. x 6-in. x 2½-in. brick placed side by side. In place of these brick a specially designed 13½-in. x 6-in. x 4½-in. brick, with notches cut in each end, 9 in. apart to make a more substantial support for the checker work, as shown in Figs. 1 and 3, has been substituted. In order to make a more firm support for the dome the specially designed skew back above referred to is used. Outside of these two special shapes and the dome plug, the total of which figure about one-quarter of 1 per cent of the brick used in the construction of the stove lining, all brick that go to make up the lining can be bought from the brick manufacturer's stock, and of these about 85 per cent are standard 9-in. straight brick, 9 in. x 4½ in. x 2½ in., and 14¾ per cent consist of standard 9-in. keys, wedges, arch and circle brick such as a brick manufacturer usually carries in stock in large quantities and can ship immediately.



FIGS. 1, 2, 3, 4.—HOT-BLAST STOVE.

Patents were issued on a stove of this design June 22, 1909. Fig. 1 shows a vertical section of the lining. Fig. 2 shows horizontal section taken at the bottom of the lining just above the various openings. A wall 22½ in. thick is used for supporting the stove, this wall being made of keys, straights and circle brick. Fig. 3 shows a horizontal section taken in quarters just below the arches, top of arches and top of special brick spanning the arches. The arches (Fig. 1) are 18 in. deep and are made of two courses of wedge brick. Fig. 4 shows horizontal section taken of two courses of brick in the checkers, one just above the other, and indicates how every joint is broken, and each brick is absolutely bonded with adjacent brick without a break, making it practically impossible for any part to become disarranged, making a strong, solid structure.

The amount of heating surface compares favorably with other stoves now in use, and in some sizes gives a greater area. In

plants where the gas is thoroughly washed it is found desirable to use a smaller checker. This stove can be laid up to excellent advantage with a 6¾-in. square checker, using standard brick the same as in the case of the 9-in. checker, and the amount of heating surface will be somewhat greater than with the 9-in.

This type of lining has already been adapted to stove shells having a diameter of 18 ft., 19 ft., 20 ft. and 22 ft. There are stoves now in satisfactory operation at four different plants using this style of lining, and a number of stoves are now in process of construction.

This lining is adaptable to any two-pass stove regardless of its openings, and whether it has formerly been lined with side or central combustion chamber.

Mr. Arthur G. McKee, engineer, Rockefeller Building, Cleveland, Ohio, with whom Mr. Nelson has been associated for some time, is now introducing this stove to the furnacemen of the country.

Electrostatic Ore Separation.

When particles of minerals are brought into contact with one pole of a sufficiently high-voltage source of electricity they assume the electric charge with varying degrees of rapidity, depending on their electric conductivity. This varying degree of conductivity, together with the fundamental principle that bodies carrying electric charges of unlike polarities attract one another, while bodies carrying charges of like polarities repel one another, forms the basis of electrostatic ore separation.

If a mixture of mineral particles be subjected to the influence of an electric charge, there will be a difference in the readiness with which that charge is acquired by the various minerals, and there will also be a difference in the time in which the minerals will become discharged when brought in contact with a conducting surface.

Minerals may be broadly classified into good and poor conductors. If a mixture of good and poor conductors be brought into contact with a surface highly electrically charged, the good conductors immediately become charged to the same condition and are repelled from it, while the poor conductors require a longer time to acquire the charge and if the electrical conditions are right may cling to the surface for a considerable period. Or if such a mixture be in some manner first charged with one kind of electricity and then suddenly brought into contact with a surface charged with the opposite polarity, the good conductors will instantly lose their first charge and be repelled from the surface while the poor conductors, carrying an opposite charge, will not lose it so quickly and will therefore be attracted by the surface until their first charge is neutralized or until they are brushed off. Again, instead of moving the mixed minerals progressively into regions of different electrical conditions, the surface upon which they rest may be suddenly changed in its electrical polarity, whereupon the good conductors will follow the change more rapidly than the poor conductors.

To utilize these principles is the object of the Huff electrostatic ore separator built by the Huff Electrostatic Separator Company, 60 India Street, Boston, Mass. The illustration, Fig. 1, shows the application of the principle, and Fig. 2 shows the finished machine as it is now used in several plants. In Fig. 1 the path of the better conductors is represented by (1) and that of the poorer conductors by (2) with certain types of electrification, while with other types of electrification the path of the better conductors is (2) and the path of the poorer conductors is (3).

The separator is built in cast-iron sections, two electrodes to each section. This arrangement permits of flexibility in construction and convenience in shipping and handling. A four-section machine is 8 ft. high, 6 ft. 6 in. long and 18 in. wide. All parts charged to high potential are insulated from the grounded machine and carefully shielded from accidental contact. The electrical fields are concentrated and confined almost entirely to the spaces in which the separations take place.

One improvement in the Huff process is the use of special generators similar to the dynamos of an electric lighting plant to produce the desired electrical wave, thus displacing the static machine with all of its attendant difficulties which formerly made electrostatic separation so troublesome. This electrical apparatus is entirely independent of atmospheric conditions and provides uniform and steady operation of the plant. The strength of the separating field is constant, and

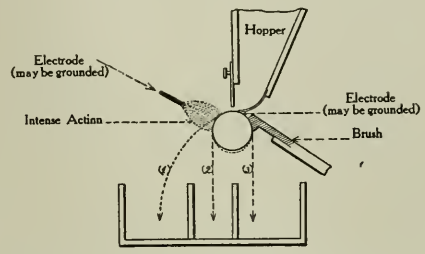


FIG. 1.—PRINCIPLE OF ELECTROSTATIC SEPARATION.

can be increased or decreased at will.

Ores for electrostatic separation must be dry and preferably free from dust. Fine crushing is not always necessary, nor is fine sizing, though coarse sizing is usually advisable. Power and labor costs are small items in the operation of the Huff machines. A ¼-hp machine is sufficient to turn the electrodes of the machine illustrated and to supply the electrical power.

As stated before, minerals may be divided into two classes—good and poor conductors. Minerals of the one class may be separated from those of the other and in many cases minerals in the same class may be separated from each other. If two

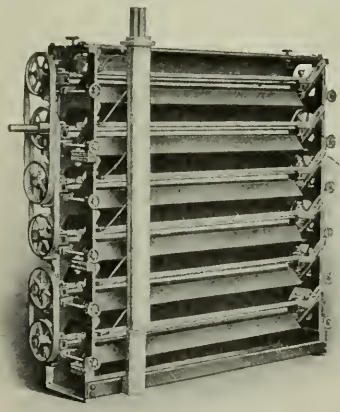


FIG. 2.—ELECTROSTATIC SEPARATOR.

minerals cannot be separated from each other in their natural condition, the conductivity of one may be altered by artificial means, such as by heat, or by chemical or electrochemical action. Thus, a poorly conductive carbonate may be converted by roasting into a conductive oxide. The following is a general classification of the common minerals according to their conductivity:

Good Conductors.

Most sulphides, such as pyrite, galena, chalcopyrite and chalcocite.

Native metals, such as gold and copper.

Some oxides, such as magnetite and hematite.

Certain arsenic and antimony compounds of copper and silver, such as pyrargyrite and tetrahedrite.

Biotite.

Graphite.

Poor Conductors.

Most silicious minerals and rocks, such as quartzite and sandstone; granite, porphyry and andesite; epidote, garnet and calamine.

Most carbonates, such as calcite and siderite.

Most sulphates, such as barite and gypsum.

Zinc sulphide, as sphalerite or blende.

Electrostatic concentration has some unique and exclusive fields of operation, such as general concentration of ores in dry and arid regions; the separation of copper and lead-silver ores from heavy gangues like garnet, barite and epidote; the concentration of carbonates of copper in limestone; the separation without previous roasting of zinc minerals from complex ores; and the concentration of molybdenite, graphite and monazite.

One of the most important metallurgical problems to which the Huff electrostatic separator is being applied is the treatment of complex sulphide ore containing zinc. A great many Western ores contain zinc, which has always been a source of trouble and loss to both miner and smelter of copper and lead-bearing ores. The latter finds the zinc detrimental to profitable smelting and the former has to stand the penalty which the smelter is obliged to impose. Finally, the zinc itself is not saved, thus increasing the total loss in mining and smelting.

It has been the successful application of electrostatic separation which has solved the problem of the miner with zinc in his ores. By this process the lead, copper, gold and silver bearing minerals may be separated from the zinc, thus producing two materials each suited for its special treatment, and much sought after by different smelters. The lead or copper smelter is no longer handicapped by a small percentage of zinc, and the zinc smelter is provided with a new supply of clean high-grade zinc concentrates.

Generators for Electrometallurgical Work.

By M. Berthold.

The steadily increasing demand for low-voltage generators for electrometallurgical work has been the stimulus in the

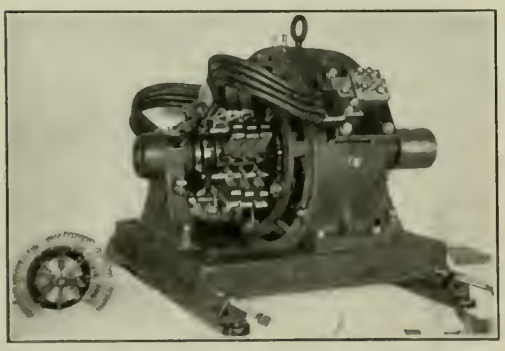


Fig. 1.—Low Voltage Dynamo for Electrolytic Work.

further development of such apparatus by the Ideal Electric & Manufacturing Company, of Mansfield, Ohio. Some details on the principal features of their designs are given in the following notes.

The principal considerations in the production of low-voltage generators, in particular those of very large ampere capacity, center upon the following points:

Heating.—The temperature elevation above the surrounding atmosphere is limited to 35° C. in any part of the machine when the generator is operated at proper speed continuously under full load. All parts of the machine, such as the armature core, winding and commutator, are well ventilated; besides all conductors are worked at low-current density. Thus, the in-

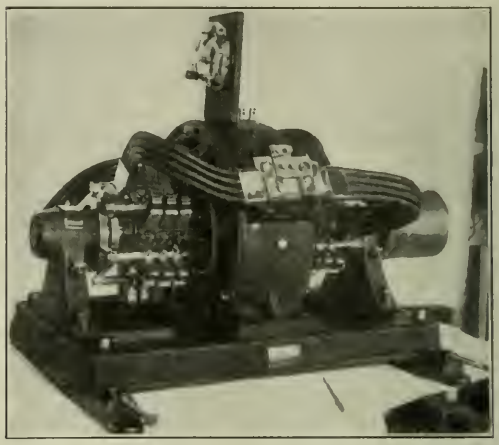


Fig. 2.—Low-Voltage Dynamo for Electrolytic Work.

sulation is best preserved which furnishes a guarantee against any service interruption.

Special care has been taken to eliminate completely the local heating of current-carrying contacts for frequent temperature changes due to the operation on light and heavy loads and subsequent shutdown tend to loosen the contacts. The surfaces are carefully finished, liberally proportioned and made easily accessible for inspection.

Armature Winding, Commutator and Brushes.—One or two commutators, according to the rating, are provided, and a construction of the most durable form for this sensitive part of the machine is selected. Sparkless commutation of the large currents is essential and the satisfactory working depends upon the electrical design as well as upon the construction of

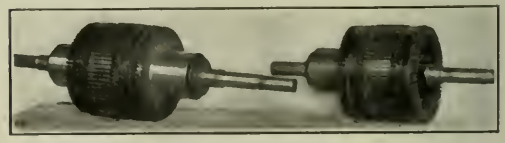


Fig. 3.—Armature and Commutator Construction

the commutator and brushes. The commutator is designed with a large number of segments in order to reduce the local currents which flow in the armature coils undergoing commutation. Because of the extremely low resistance of the armature winding, a slight irregularity in the shape of one or another coil leads to an unbalancing of the armature circuit or circuits. By the method employed in machine forming all the coils to like form and length, the armature circuits are made perfectly symmetrical and are thus equalized with mechanical precision.

The current is conducted to the terminal boards by means of the brushes contacting with the holders through large surfaces. Copper leaves carry the current to suitable clamping devices on the stems, independent of the spring mechanism

necessary for the adjustment of the brush tension. Thus extremely cool operation is maintained even at comparatively heavy loads maintained during long periods.

Brush Support.—The brush-holders, brushes, stems and cables, representing considerable weight, are supported on the frame in preference to supporting same at the extreme end of the bearing housing. This construction provides also for a greater freedom in the accessibility desirable for the adjustment of the brushes during operation, and eliminates any vibration necessarily so injurious to the proper working.

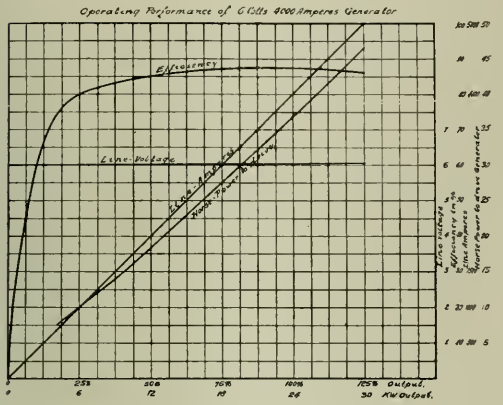


FIG. 4.—TEST CURVES OF GENERATOR.

Voltage Regulation and Excitation.—The standard machines are furnished as compound-wound generators and the compounding is so adjusted as to practically maintain constant the voltage within the guaranteed rating, when the generator is operated at practically constant speed. All machines up to the largest ampere capacities and above 5 volts at the terminals are self-excited.

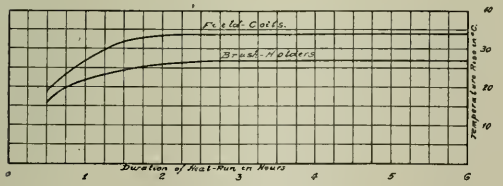


FIG. 5.—TEMPERATURE TEST.

Fig 1 and Fig. 2 illustrate machines of the standard type above 2000-amp line current. Fig. 3 gives a detail view of armatures showing the commutator construction and the armature winding.

The curve sheets Fig. 4 show the operating performance of a 4000-amp and 6-volt, compound-wound generator, while Fig. 5 gives a record of the maximum temperature rise of this generator attained during continuous full-load operation.

The performance given in the above curve sheets apply in general to all the standard ratings and each generator, after completion, is subjected to a full-load test.

Smelters in Colorado.—The Traylor Engineering Company, Allentown, Pa., is erecting two smelting plants in Colorado for private smelting companies. One is at Alma, near Leadville, and is being built for a company owning and leasing mines there. The other is at Red Mountain, near Ouray, in the San Juan country.

System for Utilizing the Heat of Blast-Furnace Slag.

When it is considered that one ton of molten slag from a blast furnace contains (on the average) sufficient available heat to produce upward of half a ton of steam (say, 30,000 cu. ft.), any system that can utilize that amount of energy in the form of steam power must be of such economic importance as to merit the consideration of the management of smelting works generally.

Prior to the advent of the low-pressure (1-lb. gage) steam turbine various attempts have been made to utilize the heat energy contained in molten slags in the form of high-pressure steam, but the practical difficulties experienced in admitting the molten material and removing the cooled or granulated product from a closed high-pressure steam generator have been the chief reasons why such an important and economical source of power has not been utilized.

In order to render practical any system for utilizing the heat contained in molten slag from iron, copper and lead furnaces it is, in the opinion of Mr. Claude Vautin of London, the inventor of the present system, necessary that the following conditions be fulfilled:

(a) The use of the steam at or about atmospheric pressure.

(b) The continuous admission to, and removal of the slag from the steam generator.

(c) The prevention of any escape of steam from, or of the admission of air into, the generator at the point of slag admission and removal.

The first of the above conditions is made possible by the utilization of the modern low-pressure steam turbine, the great possibilities of which are daily receiving the closer attention of all engineers.

The second is fulfilled by means of a specially designed and constructed trapped-slag inlet for the admission of the molten slag, and of a suitable trapped elevator for the removal of the granulated slag from the steam generator.

The third is made possible by means of a trapped-slag inlet so constructed as to freely permit the molten slag to enter the generator without the escape of the low-pressure steam therefrom, and at the same time to preclude the possibility of air entering same, thus ensuring the efficient working of the condenser connected with the low-pressure steam turbine.

Description of System and Plant.

The system consists in continuously bringing molten slag into water contained in a closed vessel, the slag being chilled or granulated and continuously removed from the vessel. The low-pressure steam produced is led either to a low-pressure steam turbine or into a heat accumulator (e.g., of the Rateau type) connected with the turbine and condenser.

The plant, therefore, consists of a steam generator, a low-pressure steam turbine connected with an electric generator, and a condenser; the addition of a heat accumulator is recommended in cases where variation in the supply of molten slag to the steam generator is likely.

The steam generator consists of a closed vessel containing water, to which is suitably attached a trapped inlet for the admission of the molten slag, an elevator for removing the granulated slag, and a centrifugal pump. The latter is for the purpose of applying boiling water from the generator to the granulation of the slag, and will be referred to later.

The steam turbine, electric generator and condenser, being well-known machines, need no detailed description.

The form of the generator can be varied to specific requirements. Generally one of the following types is recommended:

(a) An iron or steel vessel consisting of one vertical and three sloping walls coming to a point which forms the boot of the slag elevator.

(b) A boiler shell of the necessary diameter and length, containing a trough into which the molten slag falls from the

trapped inlet, where it is struck by a stream of hot water from a centrifugal pump and effectively granulated. This form of generator is provided with a slag elevator at the farther end.

The inlet to the generator varies with the class of slag to be treated; if the slag flows freely it will consist of an inner lip over which the flow into the steam generator takes place and a partition arranged so that the slag passes underneath it; the level of the slag is adjusted to a height such that a steam pressure in the generator of about 1 lb. per square inch is balanced by a head of about 8 in. of molten slag. This form of trap, besides preventing any escape of steam, precludes the admission of any air into the generator, a precaution which, as previously stated, is necessary for the best working of the condenser connected with the steam turbine.

Should the slags under treatment be viscid or "quick setting," a modified design is employed which obviates the necessity of a slag trap, while effectively preventing the escape of steam from, or the admission of air to, the generator.

The elevator for the removal of the granulated slag from steam generator is arranged so that any air carried down by the buckets is prevented from entering into the generator and from thus decreasing the efficiency of the condenser.

The slag-trap inlet is easily removable from the generator for repair; each generator is provided with two trap inlets, so that while one may be under repair the second is in operation; any stoppage in action is thus prevented.

Means are provided to prevent the escape of steam from the generator during the removal or repair of an inlet. The shape and construction of the slag inlets is such that the inner lip over which the slag flows into the generator can be readily barred and kept open.

It is the low pressure which the Vautin system employs that allows the heat contained in molten slags to be utilized for power production. This will be readily appreciated when it is pointed out that a steam pressure of, say, 1 lb. (gage pressure) is balanced by a head of about 2 ft. of water in the slag elevator casing and by about 8 in. of molten slag in the slag-trap inlet.

The granulation of the molten slag is effected by causing it to be brought into contact with a stream of hot water drawn from the generator and delivered at a high velocity by a centrifugal pump. This method of granulation prevents any condensation of the steam generated, as both are practically at the same temperature.

In order to provide an even and continuous flow of slag into the generator, where two or more blast furnaces are in operation, it is advisable to add a slag storage system, in the form of a reverberatory furnace, into which the slag from the blast furnace is tapped from time to time, or dumped from slag cars. From this storage the molten material is run continuously into the generator. The fuel here necessary to maintain the slags at the temperature at which they flow from the blast furnace is comparatively small, but little more than the radiation losses requiring to be made good. In lead and copper smelting works the reverberatory storage hearth also acts as a settler and permits of the recovery of a considerable amount of matte (or metal).

The granulated slag from the generator may be fed from the elevator to a belt conveyor and removed to the dumps, a system usually more economical than the usual slag-car method. The cost of operating the Vautin system is determined solely by the running and up keep of the slag elevator, centrifugal pump and slag-trap inlet, plus the ordinary interest and depreciation allowances. Any good type of low-pressure turbine can be used in connection with this system of steam generation.

Tests have proved that no ill effects upon the blades of steam turbines of the Rateau type are produced by the small amounts of sulphuretted hydrogen which sometimes arise from the granulation of certain slags in water.

Thermal Data.

The following thermal data are submitted by the inventor. The available heat contained in the average copper and lead blast-furnace slag is from 318 to 350 calories per kilo. From this must be deducted the heat retained by the slag removed from the steam generator at a temperature of 100° C., with a specific heat of 0.18, or a total of 18 calories. Figuring from a basis of 1000 kilos of slag per hour at its lowest available heat of 318 calories per kilo we have $1000 \times 318 = 318{,}000$ calories available heat, from which must be deducted $1000 \times 100 \times 0.18 = 18{,}000$ calories retained in the slag, leaving 300,000 calories for conversion of water into steam of atmospheric pressure at sea level. As each kilo of water at 15° C. will require $85 + 537 = 622$ calories, it should be possible to produce $300{,}000 \div 622 = 482$ kilos of steam for 1000 kilos of slag per hour. Assuming a consumption of steam in a good, modern low-pressure turbine to be 15 kilos per hp-hour, we have $482 \div 15 = 32$ hp.

From this figure deduct, say, 8 hp to cover all possible contingencies, such as variation of temperature of slag, radiation losses in steam generator, etc., and we have left 24 hp per ton of slag per hour, or 1 hp per ton of slag per day of 24 hours.

Saving in Coal Bill.

The following figures by the inventor show the estimated saving in the coal bill of a copper or lead smelter generating 1000 hp by means of coal as compared with the cost of generating the same power from 1000 tons of slag per day. Assuming the consumption of 1.5 kilos (3 lb.) of coal per hp-hour and a cost of 20 shillings ($5) per metric ton (or 2000 lb), Mr. Vautin calculates the cost of 1000 hp for a year:

Steam from coal ($8760 \times 3 \times 1000 \times \$5$)	$65,700
Steam from slag (estimated by the inventor)	7,500
Estimated saving in cost of fuel per year	$58,200

The Use of Recording Velocity Gages in Boiler Plants.

By F. Hymans, M.E.

The introduction of economy in the boiler-room has created a demand for an instrument that will give an accurate record of the process of combustion and of the performance of the mechanical stoker or fireman. The apparatus described in this article is intended to solve this problem. The present-day draft gage has proved entirely inadequate. Under certain weather conditions, for instance, the wind may exercise a suction and the draft gage would indicate a vacuum even though the damper is entirely closed. The varying thickness of the bed of fuel, the varying constitution of the fuel, the formation of clinkers offer a varying resistance to the passage of air through the grate, and though the draft gage may indicate one and the same vacuum, the amount of gas that passes through the flues may be wholly different.

The recording velocity gage of the "Hydro" Manufacturing Company, 721 Lewis Block, Pittsburgh, Pa., overcomes the difficulty by measuring the difference of pressure of the combustion gas at a point above the grate surface and at a point close to the damper. This differential pressure is the pressure that forces the gases of combustion through the flues; it is solely dependent upon the velocity of the gases and it is, therefore at once a measure for (1) the velocity of the combustion gases, (2) the amount of the combustion gases. And since the latter stands in intimate relation to the amount of fuel and air, the indications of the instrument are further a measure for (3) the amount of fuel burned on the grate, and (4) the amount of air used in the process of combustion.

This illustrates at once the fuel-saving possibilities of the instrument and still more so when it is remembered that the velocity of the combustion gases is a direct measure for the

time required to pass through the flues, i.e., for the time the gases have been in contact with the heating surfaces. Therefore excessive differential pressures are indicative of high stack temperatures and of loss.

In how far the instrument may be used as a check on the

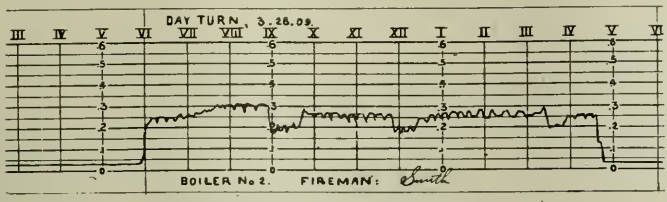


FIG. 1.—RECORD CHART OF HAND-FIRED BOILER.

fireman, Fig. 1 clearly shows. Working hours at the factory where the diagram is taken are from 6:30 a. m. to 5 p. m. The charts shows that the fireman began getting up steam half an hour earlier, at 6 a. m., the damper being closed during the night and the differential pressure being 0.225 in. At 6 a. m. he opened the damper, the differential pressure increasing at once. After that the differential pressure gradually increased and the chart shows in nearly regular intervals downward pointed curves. Such a point occurs when the fire door is opened, i.e., when fresh fuel is added to the grate.

The curve would point upward if the damper is not closed simultaneously with the opening of the fire door, as the opening of the fire door permits the air to enter direct, eliminating the resistance of the grate against the passage of the air, and thereby permitting a greater amount of air to enter the flues. This the instrument indicates by an increase of the differential pressure and the curve points upward.

On the chart, Fig. 1, the curves properly point downward, showing that the fireman decreased the amount of air entering the furnace by partly closing his damper. The pointed curves are, therefore, a fine check on the regularity of the combustion process. Between 9 a. m. and 10 a. m., the differential pressure drops for a long time, indicating the cleaning of the fire from clinkers with a simultaneous greatly closed damper. Thereafter the combustion process continues regularly as indicated by the downward pointed curves and at about 12 m. the drop of the differential pressure indicates a second deposit of clinkers.

Between 1:30 p. m. and 3 p. m., the points of the curves are less sharp, indicating the fire door remained open longer than previously. This was made necessary as the fireman at that time had to use wood and waste of the factory for fuel. Between 3 p. m. and 4 p. m., the chart indicates another deposit of clinkers. At 4:30, half an hour before close of business, the differential pressure drops again to 0.025 in., indicating that the fireman has banked his fire for the night.

While the foregoing is an excellent example of the check the instrument permits of the work of the fireman, it is equally useful for mechanical stokers for the simple reason that whatever the type or make of the furnace, the speed and amount of the combustion gases should be a minimum for the best

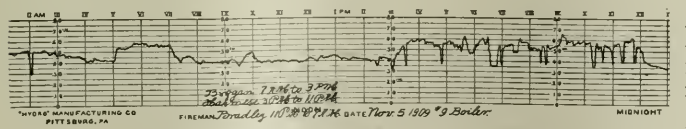


FIG. 2.—RECORD CHART WITH MECHANICAL STOKER.

results. With this point in view a still more important use can be made of the instrument which of necessity must result in prevention of waste. A few observations will in any boiler plant determine the minimum differential pressure that is required to maintain the steam pressure under the varying loads. These once determined the fireman or mechanical stoker should be made to regulate the combustion process in accordance. The result will be the maintenance of the most economical velocity and amount of combustion gases resulting in the most economical consumption of fuel.

The diagram in Fig. 1 is made of a hand-fired boiler and it might be expected that the curves would show a much more regular aspect with mechanical stokers. That this is not the case is shown by Fig. 2 being a diagram of a 350-hp boiler with inclined type of mechanical stoker. The boiler was attended by three men and the curves plainly show how much the personal equation enters into the handling of even a boiler equipped with a mechanical stoker.

Brogan beginning at 7 a. m. brings the differential pressure quickly down to about 0.4 in. at 7.30 a. m. and practically maintains same at this height until 3 p. m.

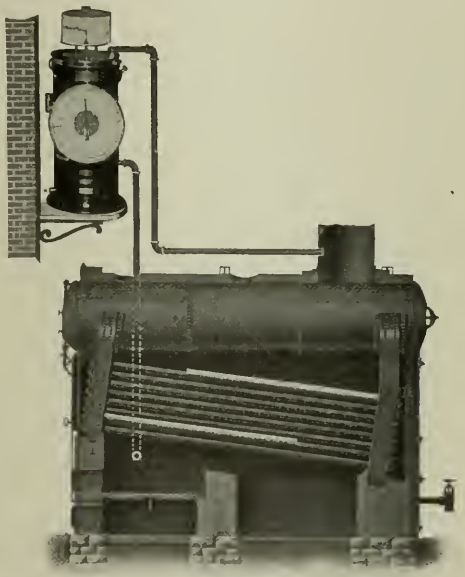


FIG. 3.—RECORDING VELOCITY GAUGE FOR BOILER PLANT.

At 3 p. m. Hahmese takes the job and right away gets in trouble. The differential pressure quickly increases and shows sudden drops at about 3.30, 4.50, 6.40, 7, etc. This is caused by the fireman closing the doors of the ashpit, which was the only means of regulation employed in this plant. From 6:40 to nearly 7 p. m., that is, for nearly 20 minutes, it was necessary for him to throttle the air supply in order to reduce the head of steam, although this was one of the periods that the peak-load occurs and the record is therefore a plain indication of waste.

At 11 p. m. Bradley goes on duty and no doubt the bad showing of his record between 11 p. m. and 12 p. m. is due to the unsatisfactory operation of his predecessor; from 12:30 a. m. to 7 a. m. his record shows up fairly good. Of course, with the increase of load more coal must be burned, which would in turn result in an increase of the differential pressure.

Therefore, a mere increase of the differential pressure is not necessarily an indication of waste, but when the curve shows so many sudden drops as is the case between 3:30 p. m. and 12 p. m., it most assuredly is. It shows that the combustion process has been accelerated too much, which is naturally coupled with waste of coal. With the further knowledge that the peak-load occurs between 7 a. m. and 10 a. m. and 5 p. m. and 9 p. m. and comparing the average differential pressures of these periods it is easy to deduct where waste has occurred.

Fig. 4 shows the construction of the apparatus, which is ingenious in its simplicity. It consists of a cylindrical vessel *g*, a cover *a*, which dips with its outer rim into a liquid contained within the annular space between vessel *g* and cylinder *h*. This liquid acts as a seal for the gas within vessel *g*. The working parts of the instrument consist of the bell *b*, to which is attached a float *c* guided upon a rod *d* and submerged in the operating fluid. Gas pressure is introduced under the bell through nozzle t_1 and above same through nozzle t_2, the differ-

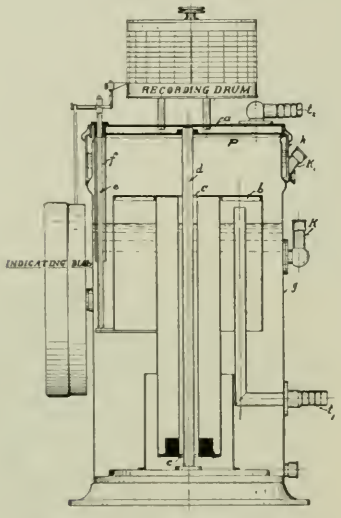


FIG. 4.—CONSTRUCTION OF RECORDING VELOCITY GAUGE.

ence causing the movements of the bell which are carried outwardly to the recording drum containing the clockwork of the instrument by means of rod *e* working through a tube *f* attached to the cover *a*.

The beauty of this arrangement is evident through the fact that the areas of bell and float stand in definite relation to the indications of the instrument. The standard construction is such that a differential pressure of 1 in. water represents a movement of 4 in. of the bell, and this being obtained without the use of magnifying levers justifies the claim of extreme sensitiveness and accuracy of indications for the instrument. In order to make the indications plainly visible to the fireman the rod *e* is further connected to a dial indicator magnifying the actual pressure 40 times. Workmanship of the instrument is in line with its other high-class features; in fact, the scientific principle of its operation, its construction and its usefulness merit a wide recognition for the instrument

"Tycos"-Rochester is the title of a neat publication issued by the Taylor Instrument Companies, of Rochester, N. Y. The designated place of publication is given as the "Thermometer City," and the contents are devoted to an exposition of the variety of recording apparatus made by the company. The pamphlet is sent regularly to interested parties on request.

Repairs of Broken Castings by Reinforced Brazing.

BY W. G. SWANEY.

The importance of being able to repair a broken casting on the spot without loss of time is so evident that there is no need to discuss it further; the difficulties of this problem are well known.

Some four years ago certain men began experiments of brazing broken castings, and these proved so practical and successful that the Reinforced Brazing & Machine Company, of Pittsburg, Pa., was formed to carry on this business. The process as perfected is entirely original with this company, and by it the spelter is introduced directly into the pores of the iron on

FIG. 1.—6000-LB. HORIZONTAL PUNCH WITH HEAD BROKEN.

both sides of the brazed portion, thereby not only reinforcing the metal adjoining the fracture, but giving a firm hold on the body of the metal beyond the fracture back into the healthy strong metal.

The range of work which the company can undertake is very wide and covers all sizes of castings, from the smallest to the largest, this being evidenced both from the illustrations shown herewith and the fact that it has brazed successfully broken castings from a waffle iron up to a 27-ton engine bed. Many of these, too, have been of both a delicate and intricate character and necessitated great care to preserve the original lines of the

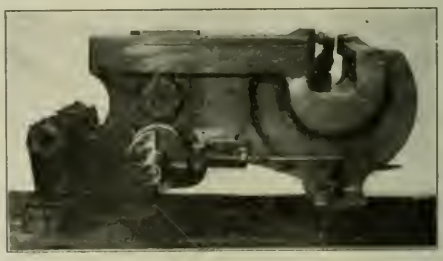


FIG. 2.—REPAIR OF FIG. 1.

casting, but in every case they have been put in as good a condition, and in many cases better, than prior to the break, and are serving their purpose to-day.

It often happens that the original casting was not strong enough for the purpose it served, and broke through weakness. In these cases not only does this company unite the fractured portions, but also introduces reinforcements of steel, thereby increasing the original strength of the casting and enabling it to hold up under its work.

The illustrations shown herewith are taken from a number of similar cases showing work which has been brazed and put back into service, and all of which without this treatment would have been consigned to the scrap pile, so that they give a convincing idea of the great benefit and value which the perfection of this process will be to the mechanical world.

There are many instances, however, to which the "reinforced brazing" process is not applicable, on account of the lightness of the pieces or the liability of warping or distorting the same while bringing them to a brazing heat. In such instances the oxygen-acetylene process is applied, and the results obtained have been highly satisfactory. To demonstrate this, the repair of automobile cylinders may be mentioned. This job is daily being made, and they are repaired so as to be quite as good as new without affecting the bore in any manner whatsoever. It is applicable to any style of castings up to 1½ in. in thickness.

Fig. 1 shows a 6000-lb. horizontal punch with head broken. Fig. 2 shows the same after being repaired.

Fig. 3 shows a Baldwin-Westinghouse mine-locomotive frame with the bumper broken through coupling and bumper broken from both side frames. Fig. 4 shows the repaired frame ready for service and entirely as good as new.

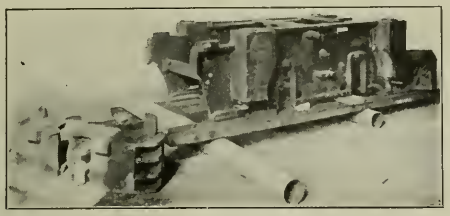


FIG. 3.—MINE LOCOMOTIVE WITH BUMPER BROKEN.

FIG. 4.—REPAIR OF FIG. 3.

FIG. 5.—BROKEN AUTOMOBILE CYLINDER.

FIG. 6.—REPAIR OF FIG. 5.

Figs. 5 and 6 show the results of oxygen-acetylene welding on Pope-Hartford automobile cylinders. These cylinders were cracked in the water jackets on all four sides, due to freezing, and were repaired without in any way injuring the bore.

The North Dakota School of Mines has recently installed a complete equipment for fuel and gas testing. A briquetting plant for lignite fuels and a gas-producer plant of the most approved type, together with a complete laboratory for fuel and gas analysis, comprise the installation.

Ozone Apparatus.

While the production of ozone has been studied for a long series of years by some of the most eminent chemical scientists, the industrial exploitation is comparatively of recent date. Nevertheless, in Europe, at least, there exists now a successful commercial ozone industry. The chief applications are, of course, purification of air and sterilization of water by means of ozone; both applications are being worked on a large scale, but ozone has also found application in breweries, tanneries, perfume factories, soap factories, lard factories, laundries, lace factories, distilleries, etc.

While this country has been somewhat slow in taking up the new development, considerable progress has recently been made along commercial lines. As the most recent instance, we notice the formation of the Standard Electro-Utilities Company, of Chicago, to give an impetus to a more satisfactory appreciation of the possibilities of ozone in this country.

This company owns the patents for the Vohr electric ozonizer, which is of very simple construction and which is stated to have a high efficiency. In order to accomplish an effective treatment of as large a volume of air as possible, the period of air contact in the Vohr ozonizer, through a rapidly revolving fan, which is one of the electrodes, is exceedingly brief. From 15,000 cu. ft. to 30,000 cu. ft. of air are treated per hour.

In the Vohr ozonizer the troublesome cleaning feature of some other types is eliminated. With no points to carbonize and no cleaning problem, the Vohr ozonizer is stated to remain practically permanently efficient. Further, one unit does the work of a number of separate units.

OZONIZER FAN.

The Standard Electro-Utilities Company have laboratory facilities, to which they invite the attention of those who may be interested in the subject of ozone. This company is equipped for any service which may require scientific data on this "coming field" of electrochemistry.

Electric Steel Refining.

The Crucible Steel Company of America has adopted the Heroult electric steel furnace for the production of highest grades of steel. A 5-ton furnace, to operate in conjunction with a 40-ton basic Wellman furnace, will first be installed at the Atha Steel Works in Harrison, N. J., and a duplicate of this is contemplated for the Park Steel Works in Pittsburgh. Even greater progress is reported from England. A number of large English steel works are making plans now to put up several large Heroult furnaces, three of which will have a capacity of 20 tons each.

Friedrich Krupp & Company, in Essen, are installing the Girod electric furnace for the production of the highest qualities of steel. It is understood that their contract calls for the installation of five furnaces each of 10 to 12 tons capacity, and that it is intended to operate them both with cold and molten charges.

A Scientific Abrasive Helpful.

Steel emery and crushed steel is, no doubt, known to some of our readers, but whether its use as it has developed in the last five years has been noticed by others than those who come in contact with it is doubtful.

Crushed steel and steel emery is a product made of crucible steel fractured into grains of various fineness and then tempered to the required hardness and toughness. These particles are very carefully graded, so that the grains are of the exact size.

This in itself is a remarkable development and took years to perfect.

Its effectiveness in abrading or grinding, particularly in valves that show wear or leakage in plants where shut-down or removal of parts means serious loss, is one of its many applications or uses. Many of the water works and practically all the railroad shops in the United States use it for valve grinding when two surfaces are to be faced together so as to form a steam or air-tight joint.

A very small quantity of steel emery is effective, and its grinding quality is so rapid that its value for work of this kind is easily appreciated.

These materials are used for all other applications of grinding and cutting where same is used loose or in bulk, not used with a binder. This is brought about from the fact that all the angles of the grains are sharp and distinctive and in use for grinding, cutting and sawing, when one edge becomes worn out it presents in its operation a new one, thereby avoiding at all times flat abrading or cutting sides.

It is used effectively in the making of slides and absolutely plane surfaces for scientific measurements and is used largely by the optical and lens makers.

The largest use for crushed steel is for the sawing and rubbing of all kinds of stone, marble, granite, etc., taking the place of sand at the very lowest cost of the latter.

This material has been manufactured for more than 20 years by the patentees and owners of the Pittsburgh Crushed Steel Company, Pittsburgh, Pa.

Filter-Tank Cyanide Process.

Among the many mechanical devices invented for the purpose of making ores more amenable to the cyanide process, the Holderman filter tank, of the Holderman Process Company, Salt Lake City, is worthy of some note. The basic idea of the Holderman process is to treat finely ground ore with cyanide solution, without previous separation of sand and slime, and to filter the gold-cyanide solution from the pulp without the use of filter presses or other expensive auxiliary apparatus. Rapidity of operation is effected by greatly increasing the leaching and filtering surface, and by the use of a simple agitator.

Each filter-tank is an oblong wooden or steel body about 10 ft. long, 6 ft. wide and 6 ft. deep. Fitting inside the tank and of about the same dimensions, is a large canvas filtering sack or basket. Space is maintained between the canvas and the walls of the tank by means of square slats nailed to the inside bottom and by V-shaped slats nailed to the sides and ends. The bottom of the tank slopes from back to front, having a pitch of 2 in. to the foot. Discharge gates are placed in the front wall, the bottoms of the openings being flush with the bottom of the tank. A drain pipe is set in one corner of the tank to carry off filtered solution. By this arrangement of the filtering sack the area of the filter medium is greatly increased.

This effect is still further augmented by the use of adjustable frame filters set at intervals across the tank. These occupy but little space and are made to communicate with the space between the large filter sack and the walls of the tank. The number of these filter leaves used in any tank depends on the treatment conditions and is increased or decreased according as greater or less leaching and filtering area is wanted. The frame of each leaf is of iron pipe enclosing a leaf of coarse matting, all being sewed over with the filter fabric.

Agitation may be used if found necessary and is accomplished by a simple device consisting of a revolving longitudinal shaft to which are fixed screw blades or paddles at intervals to correspond with the spaces between the filter frames.

For simple percolation three or more of the above-described tanks are arranged in stair-like succession, as shown in the accompanying illustration. The ore, crushed to pass through a 40-mesh, or finer, screen, is charged into the tank without separation of slimes from sands. If the charge be of dry ore the dissolving solution may be added by flooding the surface of the charge, in which case the fluid makes its way quickly through the exposed parts of the various filters until the liquid spaces between filter fabric and body of tank and within the frame filters are filled; or the solution may be introduced, to the same effect, through the solution-discharge pipe of the tank and be thus immediately communicated to every inch of filter fabric, providing an aggregate of percolating area co-extensive with that of the filtering area, plus the surface square measurement of the ore contained.

As every particle of ore is within a few inches of a fluid-filled surface, thorough percolation is certain and quick. Here is seen the effectiveness of side percolation as against that of percolation merely from above or below, for each moistened particle becomes suspended in the all-surrounding liquid, leaving the next free to the attack of the incoming flood, whereas in percolation from above or from below only, packing is certain and percolation out of the question when ore is slimy.

If the ore to be treated is crushed wet without cyanide, the excess of moisture may be quickly filtered off by turning the charge into the first tank, leaving the solution drain pipe open; or, what it better still, large square or oblong settling tanks fitted with the cross-frame filters may be used to receive the crushed product, in which case the recovered moisture may be pumped back to the crushing machinery or otherwise usefully disposed of, this arrangement proving of immense value where there is a scarcity of water. From such settling tanks the pulp drained thus of its surplus moisture may be sluiced down into the dissolving filter tank with the required strong cyanide solution.

Crushing with dissolvent solution is desirable, from many points of view, with certain ores. Where this method is adopted, the first tank may receive the pulp directly from the crushing machine or from tables, copper plates or other intermediary devices.

When sufficient time has elapsed for the dissolving of values contained in the ore under treatment, the drain pipe of the tank is opened and the gold-bearing solution flows in a continuous crystal stream to the zinc boxes. Supposing the tank to have been charged with equal weight of pulp and solution, 70 per cent to 84 per cent of the solution and necessarily an equal percentage of dissolved values will have been filtered off within from three to five hours, the exact percentage and the time within the stated limits being dependent upon the amount of slime in the charge. Thus, an all-slime pulp would give off 70 per cent to 75 per cent of the moisture within five hours, while an ore comparatively free from slime would give off 84 per cent or more within three hours.

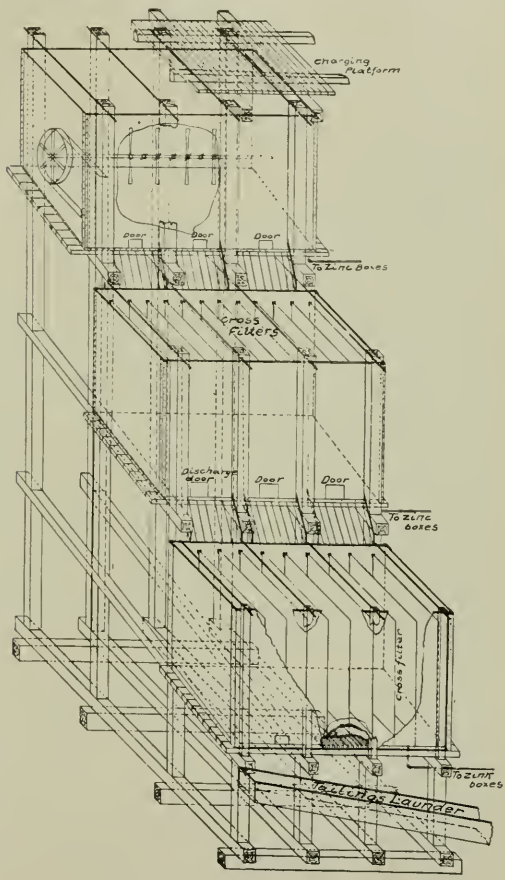
As soon as the desired percentage of moisture has been filtered off by the first tank, the front gates are opened, and by means of a hose supplied with weak solution or wash water, the partially treated pulp is sluiced into the succeeding tank. In this operation the merit of the inclined bottom of the tanks is seen, the discharge being completely effected with no more effort than is required to direct the hose. The discharging of the first tank effects the charging of the second, giving the pulp a thorough washing and aerating it in transit. This done, the first tank is left clean and ready to be recharged.

The time of the second tank is occupied in filtering, beginning with the inrush of sluiced matter from the first tank. Here may be seen to advantage one of the phenomenal features of the method, namely, this inrush of mud immediately transformed, as it were, and giving off or causing an outflow of a stream as clear as crystal, continuing so until 70 per cent to 84 per cent moisture is rendered through the filters and with it an equal amount of the dissolved values remaining from treatment in first tank.

The third tank, if used, bears the same relation that the second bears to the first, and is charged by the discharging and cleaning out of the second member, rendering in turn 70 per cent to 84 per cent of the dissolved values given to it. The pulp may then be finally disposed of either by sluicing through front discharge gates, where there is a sufficiency of water, or it may be dropped into cars through bottom discharge gates.

The per diem capacity of a set of three 10-ton tanks arranged and operated as described will depend in great part upon the time required to dissolve the values, and the time required to dissolve the values will depend mainly upon the fineness to which the ore is reduced. Having solved the mechanical difficulties of slime treatment, this method not only advocates fine grinding, but requires it, the additional cost of fine grinding being more than compensated for by higher extraction and less cost of treatment.

Given an all-slime ore, which without previous agitation with dissolvert solution, is charged into dissolving tanks as described, it will take from six to eight hours to dissolve the gold and filter off 70 per cent of the moisture; in other words, the dissolving tank, used as such in conjunction with its filtering function, will treat three charges per day, making one set of tanks equal to 30 tons daily capacity. But where the sole duty of the tanks is to separate dissolved values the daily capacity



FILTER TANK FOR CYANIDE PROCESS.

is nearly doubled, the tanks then being capable of treating five to six charges per day, or from 50 to 60 tons.

A pair of tanks, one agitating and the other non-agitating, each having a capacity of 10 tons of fine pulp or slime, can dispose of two charges in 24 hours. In a very slimy ore the entire operation can sometimes be carried out to the best advantage in a single agitator without using a supplementary tank, as the agitation would serve to keep the filter surfaces free from accumulation of slime. Considering the large filtering capacity of these tanks and their consequent ability to save by washing all that can be dissolved irrespective of slime contained, it is evident that high extraction is dependent on the dissolution of values, and to this end the ore may be ground as

fine as necessary. The Holderman system requires no slime separators or filter presses.

The Holderman Process Company has two experimental plants in Salt Lake City, one for small lots of 100 lb. per charge, or 300 lb. to 600 lb. per day, and the other for charges of 3000 lb., or 8 to 15 tons per day.

NOTES.

Gold in Country Rock.—The detection of minute traces of gold in country rock is almost a fruitless proceeding for the reason that the processes of assaying and the materials used are not of sufficient refinement. If one attempts to determine the gold by making many assays and parting all the buttons together, he concentrates the possible gold content of his litharge and thereby vitiates the result. If he attempts to make the determination on a single assay he can detect no gold.

Slag Bricks have been used rather extensively in the building operations of the Great Fitzroy copper mines, Mount Chalmers, Queensland. Offices, water tanks, retaining walls, and foundations have been made of them. The molds are made of two L-shaped cast-iron plates, of sufficient thickness to chill the slag without becoming overheated. Slag is cast from the slag pot and the molds are removed as soon as the slag has set. The bricks weigh about 80 lb. and have resisted a crushing stress of about 13 tons per square foot.

The **Bickford Firebrick Company**, having completed an addition to their plant, 300 ft. by 100 ft., will move their general office from Farmers' Bank Building, Pittsburgh, to the enlarged plant at Curwensville, Pa. The new plant will have a capacity of 100,000 brick per day and the increased output is to be available about July 15. The office moves May 1.

Sherardizing in the United States.—From the United States Sherardizing Company, of New Castle (the owners of the United States patents for Mr. Sherard Cowper Coles' "sherardizing" process of dry galvanizing), we learn that at the present time they are erecting sherardizing plants for the Westinghouse Electric & Manufacturing Company, of Pittsburgh; the Union Switch & Signal Company, at Swissvale; the John Finn Metal Works, of San Francisco, Cal.; and the American Three-Way Prism Company, of Chicago, Ill. The National Metal Molding Company, of Pittsburgh, have just installed a new sherardizing furnace that will increase their output of sherardized conduit pipe from 20 tons a day to 40 tons.

Pebbles from Denmark.—A carload of flint pebbles has been ordered from Denmark by the Rio Plata Mining Company for use in its mines in Chihuahua, Mexico. The pebbles are used for the regrinding of ore for the cyanide process. The reason for sending to Denmark for them is that the hardest flint pebbles in the world, superior to any that can be obtained on the North American continent, are to be found there. Most of the mines in the Arteaga mining district in Mexico use these, the Rio Plata Company alone, with an output of about 1,000,000 ounces of silver a year, requires a carload every five months.

The **Allis-Chalmers Company** has just closed the largest contract for mining and ore reduction machinery let since the panic, through its London office, with the Union Miniere du Haut Katanga, of Brussels, Belgium. This comprises every detail of the crushing, concentrating, and smelting equipment for a copper-producing plant of 900,000 tons yearly capacity, including power house and electrical equipment to be built near Lake Tanganyika in the Congo territory of Central Africa. This order, which covers one of the largest and most perfectly equipped mining plants in the world, was obtained in a wide competition because of the unique ability of an American company to supply all the machinery under a single contract.

The **Electric Controller and Manufacturing Company**, of Cleveland, Ohio, has recently issued a leaflet describing type G-2 controller, which has been designed primarily for 3½-hp. to 10-hp. motors. A few of the features claimed for this controller are accessibility, ruggedness, and long life. It has a cast grid resistance which will not burn out, and the heavy copper contacts are in a vertical plane, so that mill dust will not accumulate on them.

The **Denver Fire Clay Company**, Denver, Colo., has just issued the 1910 edition of its catalog of assayers' and chemists' supplies. This is by far the most elaborate and complete catalog this company has yet issued and contains in its 424 pages many illustrations and descriptions of new and improved forms of apparatus. In addition to the usual list of supplies for laboratories, assay offices, schools and colleges, there are the following new items: improved gas furnaces and burners, crushers and grinders, electrolytic assay apparatus, and revolving filter supports. Special laboratory furniture is a new feature of the catalog. The company has its main store and fireclay factory at Denver, Col., and a branch at Salt Lake City, Utah.

Edward Brown & Son, 311 Walnut Street, Philadelphia, manufacturers of indicating and recording instruments, have recently changed the firm name to *The Brown Instrument Company*. The officers remain exactly the same as before. In addition to the company's branch at Chicago, there has recently been opened an additional office in Pittsburg, in the Oliver Building, in charge of Mr. W. H. Gibb. This step has been necessary to give proper service to the company's growing business in that district.

Electric Steel in Mexico.—It is reported that the first "electric steel converter" (whatever that may mean) to be used in Mexico is to be installed in Mexico City at the Delicias rolling mills of Mr. Richard Honey. Preparations are also made to install several "modern furnaces" at the same plant. Bar iron and small iron rails comprise the output.

Oxy-Acetylene Welding.—It is announced that the Ohio Welding & Manufacturing Company, 282 West Sixth Street, Cincinnati, Ohio, is to act as a dealer for the Davis-Bournonville Company, of New York. They have installed a large demonstration plant, including not only the welding equipment, but an oxygen plant, and will carry out all kinds of repair work for which the oxy-acetylene process is suitable. This will include broken or cracked parts in cast iron, steel, aluminum, brass, copper, German silver, platinum, etc. Further parts cut too short can be lengthened, parts worn beyond use can have metal added and dressed down, etc. This company will also shortly open a demonstration and repair shop at 2121 East Second Street, Cleveland, Ohio.

Boston Belting Company, of Boston, Mass., announces that it has terminated the arrangements whereby the Jewell Belting Company, of Chicago, Ill., has been acting as its Western agent. The Boston Belting Company will open its own Western store at 177 Lake Street, Chicago, with Mr. M. S. Curwen in charge as manager of sales. The new store will carry an even more complete line of rubber belting, hose, packings, etc., than in the past, and will be able promptly to meet all orders and inquiries.

PERSONAL.

Mr. **R. L. Agassiz**, son of the late Alexander Agassiz, has been elected a director and vice-president of the Calumet & Hecla Mining Company.

Mr. **T. Walter Beam**, consulting engineer of Denver, was in New York early in April. He will visit the new mining camp of Porcupine, in Canada, in the interests of the International Smelting & Refining Company.

Mr. **D. W. Brunton**, consulting engineer of Denver, Colo., and president of the American Institute of Mining Engineers, met with a serious accident recently while on an automobile trip to inspect the Laramie tunnel in northern Colorado. While going down a steep grade the chauffeur lost control of

the machine and it plunged into the cañon below. One of the occupants of the car was killed, another suffered serious injuries and Mr. Brunton was badly injured. It is expected, however, that he will soon be able to resume his duties.

Mr. **H. C. Eastman**, of Denver, Colo., has been in New York for several weeks past.

Mr. **W. A. Hendryx**, manager of the Hendryx Cyanide Machinery Company, of Denver, Col., was in the East during April on a business trip to New York and Philadelphia.

Mr. **E. A. Le Sueur**, of Ottawa, Canada, was in New York last month.

Mr. **James McNaughton**, general manager of the Calumet & Hecla properties, who has been spending some time in Florida, has returned to Michigan.

Mr. **R. B. Martin**, formerly of Salt Lake City, has gone to Boston, where he will be in the employ of the American Chemical Company.

Mr. **C. W. Merrill**, metallurgical engineer of San Francisco, recently returned to the States after a two weeks' stay in Santa Barbara, Mexico.

Mr. **Louis D. Mills**, of San Francisco, is now in Mexico City, where he has charge of the Mexican territory in the interests of C. W. Merrill.

Mr. **Quincy A. Shaw** has been elected president of the Calumet & Hecla Mining Company, to succeed the late Alexander Agassiz.

Mr. **D. Sheedy**, of Denver, vice-president of the International Smelting & Refining Company, was in New York early in April.

OBITUARY.

The death of Mr. **Charles Loeb** occurred in New York City, Sunday, March 20, 1910. Mr. Loeb was president of the Zucker & Levett & Loeb Company, New York, and had long been intimately identified with the electroplating industry.

Dr. **Franklin R. Carpenter**, mining engineer and metallurgist, of Denver, Col., died on April 1, at the Auditorium Hotel, Chicago, where he was stricken with paralysis of the heart. Dr. Carpenter was 61 years of age and had spent the greater part of his life in the West, where he was successively identified with the educational, mining, and smelting interests of Colorado and South Dakota. In the Black Hills region of South Dakota he was well known on account of his early connection with the State School of Mines, and his later work in pyritic smelting at Deadwood. About 10 years ago he began operating pyritic plants in Colorado, the most notable of which were those at Golden and Florence. Dr. Carpenter was a member of many scientific bodies and a contributor to scientific literature on mining, geology and metallurgy.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matters and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

Electric Furnaces (*Continued*).

660,043, Oct. 16, 1900, Wilhelm Borchers, of Aachen, Germany.

Combined arc and resistance furnaces. The charge is heated by its own resistance; by a separate resistor; by independent arcs; or by using it as one or both poles of an arc. In one furnace the charge, for example, one for the production of calcium carbide, is fed from a hopper over an adjustably inclined hearth and is gradually heated by resistance, thence passing beneath an arc whereby it is fused and discharging into a receptacle beneath the end of the hearth. A tapered electrode serving as a resistor extends into the charge on the hearth, and the other vertical electrode depends over the end of the hearth. In a similar modified furnace, the hearth has a trough at its lower end to receive the molten product. In a second modification, the furnace chamber is vertical, and horizontal carbon electrodes are embedded in its side walls, making contact with the charge. Beneath the open lower end of the chamber is a rotating disk, serving as a hearth and cooled beneath by a spray of water. Electrodes project over the hearth at its sides and the heated material discharges over the edge of the hearth onto an inclined chute. In the third modification, a mixture of the charge and a binding material is continuously formed into a slab or bar which is forced over an adjustably inclined carbon hearth or contact block and beneath an arc, the molten product falling into a receptacle. The charge acts as a resistor and is supplied with increasing current as it approaches the arc. The furnace walls may be water-cooled and the electrodes may pass through water-cooled stuffing boxes.

662,537, Nov. 27, 1900, Hugo Koller, of Nuremberg, Germany.

The smelting chamber has vertical current-feed electrodes at its top and bottom, between which is a series of separate superposed carbon hoppers, serving as intermediate (bipolar) electrodes. Each hopper is supported by carbon blocks of fireproof insulating blocks. The hoppers are so placed that "several spheres of reaction arranged in series, but communicating one with the other, will be formed."

664,034, Dec. 18, 1900, Franz Heinrich Becker, of Weveling-hoven, Germany.

Arc type, for melting and refining glass. The charge descends from a hopper over an inclined undulating hearth, over which a series of arcs are sprung. It thence falls into one of several communicating compartments in a receiving and refining pot, heated by arcs from electrodes passing through the walls of the chamber. Continuous, alternating or polyphase current may be used.

665,704, Jan. 8, 1901, George S. Strong, of New York, N. Y.

Arc type. Improvement on furnace of patent No. 587,343, and especially designed for the manufacture of calcium carbide. The electrodes are constituted of the charge-mixture, a plastic mass, shaped and fed through opposed guides and conductors of hard carbon, held in water-cooled sockets. The mixture on its way to the arc zone passes through muffles heated externally by a mixture of oil or gas and air. The feeding is effected by rollers and screws, driven by electric motors, the speed of which is automatically controlled by the rate of consumption of the electrodes.

671,008, April 2, 1901, John Zimmermann and Isidore Sol Prenner, of Chicago, Ill.

Arc type, for producing calcium carbide (by the process of patent No. 675,646). The charge is a pasty mixture of lime, carbon and a binder, such as borax, graphite and iron or molasses, fed downward from a hopper and thence upward by a screw through a vertical tube. It thence discharges into an arc sprung between the ends of horizontal carbon electrodes, and the carbide falls downward into a receptacle below, two receptacles being filled alternately. The plastic mixture is preheated in the feed tube.

NEW BOOKS.

Van Nostrand's chemical annual. Edited by J. C. Olsen. Second edition revised and enlarged, 1909. 592 pages. Price, $2.50 net. New York: D. Van Nostrand Co.

Sexton, A. Humboldt. Fuel and Refractory Materials. New and revised edition. 364 pages, illustrated. Price, $2.50 net. New York: D. Van Nostrand Company.

Armstrong, E. Frankland. The simple carbohydrates and the glucosides. 121 pages. Price, $1.20 net. New York: Longmans, Green & Company.

Ripper, W. Heat engines (being a new edition of "Steam"). 332 pages, diagrams. Price, 90 cents net. New York: Longmans, Green & Company.

Maxim, Ja. L. The apprentices' course in experimental physics and mechanics for preliminary technical students and secondary school pupils; with numerous diagrams and recent examination questions. 126 pages. Price, 50 cents. New York: Longmans, Green & Company.

Warren, W. H. Engineering construction in steel and timber. Second edition. 487 pages, diagrams. Price, $5 net. New York: Longmans, Green & Company.

BOOK REVIEWS.

The Romance of Modern Chemistry. By **James C. Philip,** D. Sc., Ph. D. 12mo., 348 pages. Good index. 29 illustrations, 15 diagrams. Price, $1.50. Philadelphia: J. B. Lippincott Company.

The author is assistant professor of chemistry at the Imperial College of Science and Technology, London. He is to be heartily congratulated on his work. He presents the whole field of chemistry in a systematic, logical, highly accurate and fascinating manner. His facts are unimpeachable, theory carefully introduced, observations and conclusions keen and honest. The book is so simply written that almost any person who can read could understand and enjoy it: on the other hand, it is so accurate and comprehensive that every chemist would find pleasure and profit in it. It is truly an interesting book, and is most warmly recommended to our readers.

* * *

The Romance of Modern Manufacture. By **Charles R. Gibson.** 12mo., 320 pages. 28 illustrations, 16 diagrams. Price, $1.50. Philadelphia: J. B. Lippincott Company.

The book lives up to its name; sugar-coated phrases full of the marvelous; very readable and reasonably accurate. To us who know more about metallurgy than we do of cotton spinning, for instance, the latter seems more accurately described than the former. The book may be recommended as a good one for beginning a general scientific education or passing profitably some leisure hours; it should be especially attractive to boys.

* * *

The Copper Handbook. Vol. IX (1909). Compiled and published by **Horace J. Stevens,** Houghton, Mich. Octavo, cloth, pp. 1628. Price, $5.

The ninth annual volume of this work follows the same lines as did the preceding volumes, and brings statistics relative to copper and copper mines as nearly down to date as possible. Comparatively little new matter has been added to this volume, and most of the chapters remain as before, owing to the impossibility of properly revising the entire work in the short time available. The publisher plans to improve the conditions under which the work is carried on so that the entire volume will receive the attention it needs. The important chapters of the book have been thoroughly revised. These are the chapters devoted to the copper mines of the world, and the statistics of copper. The former lists and describes 7551 copper mines and mining companies, and the statistics cover every phase of the copper industry.

* * *

Mineral Resources of the United States. 1908. Vol. 1, Metallic Products (816 pages); Vol 2, Non-Metallic Products (869 pages). Washington, D. C.: U. S. Geological Survey.

These volumes constitute the twenty-fifth report on the mineral resources of the United States and comprise a census of the production, imports, exports, and other features of the metallic and non-metallic products of the country. As usual, the publication of the volumes has been anticipated by the advance publication of pamphlets comprising the various chapters of the volumes, so that the information contained is given to the public without waiting for the completion of the entire work.

The most notable general feature of the mineral production of the country for 1908 was the decline in value as compared with 1907. In 1905 and 1906 the total value of the domestic mineral production exceeded $1,500,000,000 and in 1907 the still greater sum of $2,000,000,000. In 1908, however, the effect of the industrial depression of the latter part of 1907 was reflected in the decreased value of total mineral production to about $1,596,000,000, or less than that for either 1905, 1906 or 1907. Iron, steel, manganese and silver showed a marked decline in production. Gold and copper increased, the output of the latter being the largest in the history of the industry. Lead recovered from the depression of 1907 more quickly than spelter, the latter experiencing a very hard year. The minor metals, as a rule, experienced a decrease in production.

Although the coal mining industry experienced the effects of the depression of 1907, nevertheless the production of coal for 1908 exceeded that of any year in our history except that of 1907. Petroleum experienced severe industrial reverses in 1908 due to floods and fires, but the output exceeded that of 1907. In the cement industry the production showed an increase of 1.3 per cent in quantity and a decrease of 20 per cent in value. The minor non-metallic mineral industries showed in a majority of cases decreases under the production for 1907.

* * *

A Study of Electrothermal and Electrolytic Industries. By **Edgar A. Ashcroft** (Statistics by **I. J. Moltkehansen**). Part I: Introductory. 8vo., 130 pages. 37 illustrations. Price, $2 net. New York: McGraw-Hill Book Company, 1909.

The whole work will be in three parts. Part I deals with the general chemical, physical, mathematical and engineering principles involved. Parts II and III will deal with the specific operations of electrochemistry and electrometallurgy in alphabetical order.

The present volume begins with an "Appreciation" concerning the historical side of the subject, which we fear will fail of being "appreciated" by the average American reader; it is too sketchy to be really instructive, and too philosophical to be *apropos:* his Pegasus flies too high. The chapter on the theory of electrochemistry is pleasant reading, accurate but not comprehensive. (On p. 12 Boyle's gas law is inverted, and on p. 13, 0.222 is printed for 0.0222; we regret to see decimals given without 0 before the decimal point.) The chapter on "Calculations" is lucid and accurate, but brief. The electrochemical equivalents of argon, helium, krypton, neon and xenon exhibit an amusing electrochemical *impasse*, with their valencies taken as zero.

The remaining two-thirds of the book contains very incisive views and practical observations upon the technical and commercial considerations involved in practical electrochemistry. The views expressed are in many cases far-sighted, and show a deep insight into and appreciation of the fundamental economics of the subject. The remarks upon cheap water-powers, future development of large industries, and their effects upon commercial geography and national developments are very well worth reading. The author states emphatically that "a great and important change in the world's industrial conditions is impending"—a change to be wrought by practical electrochemistry.

Finally, the philosophical author of "The World's Desires" (a very readable book, by the way) shows his appreciation of the liberal conditions for development in this country by the sentence: "No such restrictions exist in America, the land of freedom." Therefore, let the eagle (justifiably) scream, and may electrochemical industries flock to "the asylum for the oppressed."

Metallurgical and Chemical Engineering

A Consolidation of Electrochemical and Metallurgical Industry and Iron and Steel Magazine

Vol. VIII. New York, June, 1910 No. 6

PUBLISHED MONTHLY BY THE

Electrochemical Publishing Company

239 West 39th Street, New York.

Pittsburgh Office, 201 Oliver Building.
Western Office, 1510 Court Place, Denver, Colo.
European Office, Hastings House, Norfolk St., Strand, London, Eng.

J. W. Richards, Ph.D., President.
Curtis E. Whittlesey, Treasurer. E. F. Roeber, Secretary.
Telephone Call: 4700 Bryant.

E. F. Roeber, Ph.D., Editor.
H. C. Parmelee, Western Editor.

Yearly subscription price for United States, Mexico and United States dependencies, $2.00; for all other countries, $2.50 (European exchange, 10 shillings, 10 marks, 12 50 francs).

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Entered as Second-Class Matter at the Post Office at New York, N. Y., under the Act of Congress, March 3, 1879

New York, June, 1910.

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Pittsburgh and Electrochemistry.

In view of the very full report of the proceedings of the Pittsburgh meeting of the American Electrochemical Society, which will be found elsewhere in this issue, it will be sufficient here to emphasize the most remarkable feature of this remarkable convention. This was the exceedingly hearty and enthusiastic welcome which Pittsburgh—all industrial Pittsburgh—extended to her electrochemical visitors. The Society was royally entertained through the four days. The courteous welcome extended by the leading industrial and public men of the city was reflected with friendly sincerity by the daily press. But Pittsburgh's hospitality was shown most splendidly and liberally by the wide-open doors of all her important industrial establishments. The series of visits and excursions was certainly delightful and will long be remembered by all those who had the good fortune to attend them. The most remarkable point, however, is that all this would have been simply impossible a few years ago, since Pittsburgh could impossibly then have shown so much interest in electrochemistry. Now the commercial success of electric steel refining has changed everything. It is another proof that the American Electrochemical Society was founded at the right time and that its formation was a necessity. The Society has always done the right thing at the psychological moment in its career, and at the right moment the Society went to Pittsburgh.

* * *

Pittsburgh is now taking an interest in electric steel refining. There is no doubt about this fact. Nor can there be any doubt that when Pittsburgh undertakes to do something, it will be done in the most effective and successful manner. As we have said before, it requires no particular flight of fancy to think of Pittsburgh as a coming electrochemical center, as a center of electric-furnace activity for steel refining. And the commercial beauty of this coming development is that it must be a benefit all around—a benefit to the consumer by giving him better or cheaper steel, and a benefit to the iron and steel industry which gets a new important tool in the electric furnace, that does not impair in the least, but increases the usefulness of the existing installations. The electric furnace will not put anybody out of business. This is a fact that ought to be emphasized, as such is not often the case in new commercial developments. This evolution, being logical and natural, would have come, of course, whether the American Electrochemical Society had held a meeting in Pittsburgh or not. But that the past meeting will tend to accelerate this evolution—the Society acting as a catalytic agent—there can be no doubt; and in this sense it may be said, as was remarked by one of the speakers at the meeting, that future generations will probably point to the Pittsburgh meeting as the dawning of the new era of electric steel refining in this country.

The Duplex Process.

The duplex process, in form of the combination of the acid Bessemer converter and basic open-hearth, is apparently now

becoming fashionable. Aside from commercial considerations which we need not discuss at present, its metallurgical rationale is twofold—the increased output over the straight open-hearth process and the simplification of the conditions with which the composition of the pig iron has to comply, rendering possible a greater leeway in blast-furnace working. Granting these advantages, we are forced to ask: Is not the combination of converter and open-hearth already out of date? Could not all the advantages, and others besides, be obtained from a combination of converter and electric furnace or from a combination of open-hearth and electric furnace? Is not the combination of an oxidizing furnace with a furnace of neutral or reducing atmosphere inherently superior to a combination of two oxidizing furnaces? Is the combination of converter and electric furnace not cheaper than the combination of converter and open hearth? These are pertinent questions which require an answer. We have formulated them here to elicit thought and discussion. For the subject is important and of live interest.

A New Process for the Fixation of Atmospheric Nitrogen.

Atmospheric air is a mechanical mixture (as distinguished from a chemical compound) of oxygen and nitrogen gases—that is, the nitrogen in the air is in elementary form, mixed, but not combined with other elements. The importance of the problem of the "fixation" of atmospheric nitrogen rests in the fact that elementary nitrogen, as contained in the air, is inert and practically useless, while nitrogen compounds are of great commercial and industrial value, especially in the fertilizer industry. The object of the fixation of atmospheric nitrogen is simply to change the elementary nitrogen of the air into a useful nitrogen compound. Since there are various useful nitrogen compounds there are various fixation processes possible. The simplest in idea is to take the oxygen and nitrogen in the air, as it is, and to try and combine as large a portion of them as possible directly into nitrogen oxides (to "burn the atmospheric nitrogen") and then work them up into nitric acid or nitrates. This can be done by subjecting atmospheric air to a very high temperature by means of electric arc discharges, as is done with commercial success in Norway in the process of Birkeland & Eyde and in the Schoenherr process of the Badische Company. The disadvantages of this method are twofold: The first product of the reaction is a dilute mixture of nitrogen oxides in atmospheric air and its transformation into nitric acid is complicated and not inexpensive; and secondly, the electric power consumption is high and the process is commercially practical only in countries where electric power is exceptionally cheap and where there is little incentive to use the available cheap power for other purposes. Norway is the classical example of such countries. Another useful nitrogen compound is calcium cyanamide, which may be used either directly as fertilizer or as starting material for the manufacture of other chemicals, like cyanide. In the manufacture of calcium cyanamide the air cannot be directly used, but it is first necessary to separate the nitrogen from the oxygen in the air (for instance, by fractional distillation of liquid air); the nitrogen is then made to react with calcium carbide at an elevated temperature, yielding directly cyanamide. The power consumption is not so high as in the nitric acid process mentioned above. Nevertheless, the production of calcium cyanamide, being based on calcium carbide manufacture, also requires comparatively cheap power.

* * *

A third process has just been heard of from Europe. Its object is the synthesis of ammonia from nitrogen and hydrogen. There is a considerable margin between the selling price of ammonia and the cost of making nitrogen gas and hydrogen gas on a large scale. But it has always been thought that such a synthetic process was impractical, because the nitrogen and hydrogen gases would be too slow to react. This prevailing idea has now been disproved by Prof. Fritz Haber, of the Institute of Technology, of Karlsruhe, who jointly with Dr. R. Le Rossignol has developed an apparatus for producing in the laboratory 90 grams of liquid ammonia per hour; this apparatus has been in continuous operation for a sufficiently long time to prove its practicability. The essential feature of the process is the use of a high pressure—something like 200 atmospheres—and the use of either osmium or uranium as catalytic agent. It seems that no high temperature is required or desirable, since the inventors mention in one of their experiments a temperature of 500° C. We hope to give further details in the near future. The use of such high pressures as 200 atmospheres would be a novelty in large-scale chemical operations, but seems hardly a prohibitive drawback. On the other hand, the process does not seem to require particularly cheap power. The development of the process decidedly deserves watching. It has already passed out of the laboratory stage, having been taken up by the Badische Company.

The Raw Materials for Basic Open-Hearth Steel Manufacture.

The statistics of basic open-hearth steel manufacture in the United States in 1909 have been awaited with unusual interest, for the pig-iron statistics, made public Feb. 1, showed a large increase in basic pig-iron production over the best previous record and the question was then presented: Did the production of basic open-hearth steel increase in corresponding ratio, or was a portion of the increased pig-iron production due to a relative diminution in the supply of scrap? Some statistics merely confirm views derived from observation of current events or from a comparison of previous statistics; others furnish direct information as to what is occurring. Of the former class are the statistics of Bessemer steel production, for given the Bessemer pig-iron statistics for 1909 it was possible by comparison with previous achievements to estimate the production of Bessemer steel, and the statistics subsequently compiled showed such estimates to be in error by but a fraction of 1 per cent. Of the latter class are the open-hearth steel statistics. The basic pig-iron statistics furnish no guide to steel production, for the percentage of pig iron to steel has varied widely. The open-hearth statistics for 1909 have been made public the past week by the American Iron and Steel Association. In 1896 for the first time separate statistics were compiled of basic pig iron and basic open-hearth steel production. In the table below we give the statistics from that year to the present, and compute the percentage of pig iron to steel. Both commodities are expressed in gross tons of 2240 tons. A very limited tonnage of charcoal pig iron, occasionally used for basic open-hearth steel

manufacture, is excluded. The steel statistics refer to ingots and castings.

	Basic steel.	Basic pig iron.	Percentage, pig iron to steel.
1896	776,256	336,403	43.3
1897	1,056,043	556,291	52.7
1898	1,569,412	785,444	50.1
1899	2,080,426	985,033	47.4
1900	2,545,091	1,072,376	42.1
1901	3,618,993	1,448,850	40.0
1902	4,496,533	2,038,590	45.2
1903	4,734,913	2,040,726	43.0
1904	5,106,367	2,483,104	48.6
1905	7,815,728	4,105,179	52.5
1906	9,658,760	5,018,674	52.0
1907	10,279,315	5,375,219	52.3
1908	7,140,425	4,010,144	56.2
1909	13,417,472	8,250,225	61.5

* * *

One must carefully avoid the inference that the percentages shown above are in the least sense representative of average practice. There is no such thing as average practice. Even in the straight pig and scrap process in the early years there were variations in the percentage at different works, and at the same works at different times, according to market conditions affecting the supply and price of scrap. Then there were even at the start variations between works, one being attached to a Bessemer plant and, therefore, readily supplied with new Bessemer scrap, while another was not so attached. Even such a purely commercial transaction as the formation of the United States Steel Corporation affected the consumption of scrap, for some of the Bessemer works combined had hitherto used their scrap to cool the heat in the converter, but thereafter were made to ship this scrap to allied open-hearth plants. Again, the rise of the Monell and Talbot processes, and later the duplex process, introduced the manufacture of large tonnages of basic open-hearth steel in the manufacture of which little or no scrap was used. Finally, minor variations are introduced by the fact that basic pig iron is used, in a limited way, for other purposes than the production of basic open-hearth steel, and the stocks of basic pig iron at the beginning and end of a year are not identical. Thus the percentages shown in the above table are merely composites. They show nothing as to the practice at a given class of plants, but they do show, in a measure, relations between the classes. It is distinctly the case that the vogue of the different processes of open-hearth steel manufacture depends very largely upon the supply of scrap, and as the increment through the use of ore is small in any case, the relation between pig iron and steel furnishes a valuable index to the necessity of using the various non-scrap processes.

* * *

The figures for 1896 and 1897 show a wide variation, increasing from 43.3 to 52.7 per cent, and at this late date do not merit discussion. In general, we may observe that in the late nineties the proportion of pig iron to steel was about 50 per cent, this decreasing to 40 per cent in 1901. We may surmise that the chief factor was that in the late nineties Alabama pig iron was in poor demand, and found its way to Northern steel works, even at Pittsburgh, whereas the more prosperous times which followed, making the choice between Northern pig iron and scrap, reduced the proportion of pig iron used. From 1901, with 40 per cent, there has been a large and almost steady increase in the pig-iron proportion, to 61.5 per cent last year. The most impressive increase was from 52.3 per cent in 1907 to 61.5 per cent in 1909, since the off-year 1908 is naturally not to be taken so seriously as years of full employment. This recent large increase in the percentage of pig iron to steel represents chiefly the growth of the special non-scrap processes. It is probable that the straight pig and scrap plants use substantially as much scrap as they ever did, but whereas they once occupied the entire field, they now occupy but a portion of it. Again, these statistics of production of ingots and castings, while generally regarded as "the steel statistics" because there are no other, are not really statistics of production, but are rather merely the records of the ingot scales. The Bessemer converter removes the impurities of pig iron, but when the ingot is converted into finished product, nearly all the waste goes, not back to the Bessemer converter, but to the basic open-hearth furnace, and in due course the material is weighed over the ingot scale at the basic open-hearth furnace. The Bessemer process weighs only the material it has refined from pig iron, but the open-hearth process weighs the material it has refined from pig iron, and also most of the waste material which the Bessemer process has refined. Thus, in a sense, it obtains undue credit in the statistics.

* * *

It is really of minor consequence whether the decreased use of scrap is at all plants, or is merely a pulling down of the general average by the rise of processes which do not use scrap The point is, that taking the industry as a whole, it uses less scrap and more pig iron, in proportion to its total output, the commercial conditions modifying the process, rather than the reverse. The market price of scrap, relative to the market price of pig iron, has increased very materially in the past two years, showing which is the governing element, for otherwise the use of non-scrap processes would have depreciated the market value of scrap relative to pig iron. There are three chief influences decreasing the supply of scrap relative to the total production of the basic open-hearth steel process: (1) The increase in basic open-hearth steel manufacture has been much more rapid than the general expansion in the iron and steel industry, the former consuming scrap, the latter involving the production of it, whether new scrap or old scrap; (2) the Bessemer steel process, which up to 1907 was growing less rapidly than the open-hearth process, is now actually decadent, and furnishes less new Bessemer scrap; (3) the trend of late in finished steel products has been that of rapid increase in the demand for the light lines, such as sheets, tin plates, wire, pipe and merchant mill products, including hoops, bands and merchant bars, with a smaller increase in demand for the heavy materials, plates, shapes and rails. In the manufacture of these light lines scrap is produced, it is true, but when put into consumption they do not replace so much old material as, on the whole, do rails, plates and shapes. Furthermore, they do not return as old material in as large tonnage as do the heavy sections. Usually there is little left of wire, sheets and pipe after they have served their period of usefulness in their original form. Evidently the basic open-hearth steel industry of the future must depend chiefly upon the refining of pig iron, perhaps with the use of ore, and with such scrap as arises in the rolling of the ingots it produces.

American Chemical Society.

As already noticed in these columns, the forty-second meeting of the American Chemical Society will be held in San Francisco from July 12 to 15.

The Atchison, Topeka and Santa Fé Railway system has been designated as the official route. The official train will be run from Chicago to San Francisco via Colorado Springs, Grand Canyon of Arizona, Redlands, Riverside, Los Angeles and Santa Barbara, stopping at Adamana for a trip to the petrified forest, and at Lang for a visit to the borax mines. This train will leave Chicago on July 4 and arrive at San Francisco early in the morning of July 12.

The plans of the local committee at San Francisco indicate that the entertainment feature of the meeting will be one of the greatest in the history of the society.

Dr. Wilder D. Bancroft is the president of the American Chemical Society, and Dr. Charles L. Parsons, Durham, N. H., is secretary.

American Institute of Chemical Engineers.

The summer meeting of the American Institute of Chemical Engineers will be held at Niagara Falls from June 22 to 25. All the sessions will be held at the Clifton Hotel, Niagara Falls, Canada.

On Wednesday, June 22, the meeting will be opened with a business session, at which the reports of officers, council and committees will be presented. The program of papers is as follows:

Wednesday morning, June 22, 11 a. m.—"Changes in Industrial Chemistry Caused by Electricity," by Edward R. Taylor. "Notes on the corrosion of iron and steel and its prevention," G. W. Thompson. "Vacuum Distilling Apparatus," Philip B. Sadtler.

Wednesday evening, June 22, 7.30.—Address of the president, Dr Charles F. McKenna, "The Study of Materials as an Element in a Course of Chemical Engineering." Report of the committee on chemical engineering education, F. W. Frerichs. "A New Product for Use in the Arts," E. G. Wiechmann.

Friday morning session, 9.30.—"Chemical Industries of Canada," J. A. DeCew. "The Manufacture and Industrial Application of Ozone," Oscar Linder. "Problems in Chemical Industry," J. T. Baker. "Arrangement of Filter Presses for Bleaching Oils with Fuller's Earth," David Wesson. "Commercial Manipulation of Refractory Elements for Incandescent Lamp Purposes," Ralph E. Myers.

Friday evening session, 7.30.—"Underground Waters for Manufacturing Purposes," W. M. Booth. "Loss in Coal Due to Storage," A. Bement. "Nitric and Mixed Acids," Schuyler Frazier. "Plant Design," W. M. Grosvenor.

A great many excursions and visits have been arranged as follows: For Wednesday afternoon a visit to the Ontario Power Company or to the International Paper Company (paper and pulp mill). For Thursday, Lackawanna Steel Company, H. W. Dopp Company plant, Linde Company liquid-air plant, Buffalo Foundry & Machine Company, Larkin Soap Company. For Friday afternoon, International Acheson Graphite Company and Hooker Electrochemical Company. For Saturday, Wickwire Steel Company, Niagara Falls Power Company and Carborundum Works. On Thursday evening a subscription dinner will be held at the Hotel Clifton.

Dr. Charles F. McKenna is the president of the institute, Dr. John C. Olsen, Polytechnic Institute, Brooklyn, is the secretary.

American Foundrymen's Association.

As mentioned before in these columns, the convention of the Allied Foundry Associations will be held in Detroit, Mich., from June 6 to 10, 1910. Very complete arrangements are being made for the entertainment of the visitors, and the usual great success of the conventions of these associations seems assured.

Among the papers to be presented before the American Foundrymen's Association are the following:

"Foundry Efficiency," by Benjamin D. Fuller; "The Personal Equation in Accidents," by Thomas D. West; "Shockless Jarring Machine," by Wilfred Lewis; "Chemical Standards for Iron Castings"; report of special committee, by Prof. John J. Porter, chairman; "Autogenous Welding of Castings," by Steelman Stephenson; "The Electric Furnace," by H. M. Lane; "Overhead Tramrail Systems for the Foundry," by A. W. Moyer; "The Permanent Mold," by Edgar A. Custer; report of the committee on industrial education, by P. Krenzpointner; proposed foundry coke specifications, by the secretary, Dr. R. Moldenke.

Among the papers to be presented before the American Brass Founders' Association are the following: "Cost and Cost System Applied," by C. R. Stevenson; "Analysis of Lead in Brass Alloys," by C. P. Karr; "Co-operative Course in Metallurgy," by Prof. J. J. Porter; "Electric Furnaces for Melting Non-Ferrous Alloys," by A. L. Marsh; "Flues in the Brass Foundry," by Erwin S. Sperry; "Magnesium for Deoxidizing Aluminium Alloys," by H. M. Lane; "Electric Power Required to Melt Brass, Bronze, Etc.," by Prof. J. W. Richards; Papers by Dr. F. T. F. Stephenson and Mr Jesse L. Jones are also promised.

Bakelite in Germany.

According to a recent issue of the *Chemiker Zeitung* "a new company has been formed in Germany under the auspices of the widely known Rutgers Works to exploit the bakelite patents of Dr. Leo H. Baekeland. The Rutgers Works are naturally deeply interested in the subject as the largest producers of carbolic acid and cresols in continental Europe, since the manufacture of bakelite will open a new and wide field of application for their products. For the start the existing installations are sufficient to manufacture bakelite, so that the new bakelite company will begin to do business shortly."

Dr. Backeland may be congratulated on having succeeded in placing the latest child of his inventive genius into such good hands in Germany.

News from Niagara Falls.

The **National Carbon Company**, the largest manufacturer of carbon products in the world, has for some time realized the importance of being in closer touch with the great electrochemical industries located at Niagara Falls, inasmuch as they nearly all are users more or less of its products. The company has, therefore, decided to locate a large plant at the Falls and equip it with new and improved machinery especially designed to meet the widest range of electrometallurgical and electrochemical requirements.

For several years a large force of experts has been studying the difficult problems of carbon electrode manufacture and the time has now arrived to put into operation the results of their investigations. Although the company has a large plant at Clarksburg, W. Va., devoted exclusively to this work, the constantly increasing use of carbon makes apparent the necessity for still further enlargement of its facilities and Niagara Falls, the very center of electrochemical industries in the United States, seems to be the logical location for the greatest benefit both to the users of its products and to the company itself.

A desirable, well-located piece of property, comprising 13 acres, has been purchased, plans are being prepared and the work of erecting the buildings will soon commence. As now planned it will require an expenditure of nearly a half million dollars for buildings and machinery, and there will still be room to double the capacity as requirements demand. It is expected the plant will be ready for operation early next year

The **Norton Company**, of Worcester, Mass., and Niagara Falls, N. Y., are about to erect a carborundum plant at Chippawa, Ontario (adjacent to Niagara, Ontario). One thousand horse-power will be employed. The plant will be located in a reinforced concrete building 120 ft. x 130 ft.

The **United States Heating & Lighting Company** are also to erect a large plant at Niagara Falls to take over the work of the National Battery Company, of Buffalo, together with other work.

"Gredag" is the trade name adopted by the **International Acheson Graphite Company** for its graphite grease in order that it may be distinguished from the inferior grease products which contain impure natural graphite. The graphite blended with a high-grade grease in gredag is the electric furnace product of the Acheson Works. As it is soft and unctuous, this graphite is a perfect lubricant in itself, but in gredag are combined all the merits of both graphite and grease, the combination making a lubricant of high quality and value.

While going to press we learn of a destructive fire on the evening of May 26 at the works of the **Hooker Electrochemical Company**. The frame buildings, including the cell room, have been destroyed. It will take four months to rebuild them. The damage is estimated at $400,000.

A Pathological Museum for Metals and Alloys.

The council of the (British) Institute of Metals announces that there has been established at the offices of the Institute, in London, a pathological museum for specimens of metals and alloys, the first contributions to the museum having been received from the president, Sir Gerard A. Muntz, Bart.

This museum, which is the only one of its kind, ought to be of great service to all interested in the metallurgy of the non-ferrous metals, as it is intended that it shall contain specimens showing the various ways in which such metals as copper, brass, aluminium, etc., can fail either as a result of faulty manufacture or of improper usage.

Electric Iron Ore Reduction in Sweden.

A contract has been placed by the Electrotemiska Metall Aktiebolaget for electrical machinery of 10,000 hp for electric shaft furnaces for the production of pig iron from ore at Trollhatten, Sweden. This is in addition to the 2500-hp furnace which is being erected at that place by the Jernkontoret.

The Iron and Steel Market.

May has seen a continuance of the curtailment in pig iron production which began in the latter part of April. Production at the beginning of April was at the rate of about 31,250,000 tons a year, there being a decrease of about 2,000,000 tons in the rate during April, and nearly as large a decrease in May. Most of the May decrease occurred in the early part of the month, and the rate of output is substantially stationary at present.

While a decrease of nearly 4,000,000 tons in the rate of pig iron production in two months or less seems spectacular, it is only about 12 per cent. In about two months at the close of 1907 there was a drop of 50 per cent.

Production of steel ingots and of finished steel products has decreased less than production of pig iron, although there have been some important restrictions at individual steel plants. It is probable that pig iron was being made to excess in February and March.

Business in finished steel products was materially larger in May than in April, and the same was true of pig iron, but in neither quarter was there active buying and the total bookings fell short of production by from 25 per cent to 50 per cent in the different lines. The trend of prices has been distinctly downward in pig iron, and such trend as finished steel prices have shown have been downward, but the movement has been relatively inconsequential.

In many quarters predictions are made of an upturn in demand and prices in the pig-iron market, based on the assumption that actual cost has been reached in the case of furnaces making pig iron from purchased Lake Superior ore. For the iron and steel industry as a whole, however, a very dull summer is looked for.

The present condition of the iron and steel market is the natural product of well-known factors. The actual ultimate iron and steel requirements of the country increase with great regularity, producing a doubling about once in 10 years. The filling of the requirements, which are more or less flexible at any given time, depends partly upon general conditions, the ease with which loans can be placed, the business and political outlook, etc. These introduce variations in the rate of expressing the requirements. The actual buying is subject to still greater variation, on account of the varying attitude of buyers to the market. If prices promise to advance, they buy more than the mere expression of their requirements would call for; if prices promise to decline, they buy less.

While in the long run productive capacity grows as demand grows, therefore, doubling about once in 10 years, the planning for new erection is done chiefly when the market is strong and active, and as the average project requires from one to two years from original conception to actual production, each period of intense market activity leaves a trail of new capacity. From late in 1904 to late in 1907 there was a strong market, with all productive capacity fully engaged, constituting the longest period of pressure in this business generation. Naturally, the crop of new erections was the largest. Under full pressure in 1907 the country was barely able to reach a rate of pig iron production of 28,000,000 tons a year; the present capacity is between 32,000,000 tons and 33,000,000 tons, and with all present projects completed 34,000,000 tons will be closely approximated.

There has been the normal increase in requirements, since 1907, but that has not been sufficient to employ all of the additional capacity. In 1908 the filling of requirements was held in abeyance as far as possible on account of the artificial maintenance of prices. The break in prices in February, 1909, soon brought buyers into the market with their postponed requirements, and as prices were advancing there was forward buying, price advances and buying interacting on each other to increase both. On the appearance of winter buying decreased, and naturally prices stopped advancing. From a condition in which buying was in excess of actual consumption the market passed to a condition in which buying was less than actual consumption. The actual consumption, there is excellent reason for believing, has been greater this year than last, or, to be more specific, the rate of actual consumption or putting into use of iron and steel in the first five months of this year has been greater than in the last five months of last year. Market conditions, however, have been precisely the reverse, for a strong advancing tendency has given way to a slight declining tendency.

The present situation is that of buyers of coke and pig iron taking hold in a limited way, simply from the belief that in these commodities prices have approximated the bottom, while in finished steel products they are holding off as much as possible, on the assumption that since there has been no noteworthy decline in prices, one is yet to come.

The ease with which demand has grown up to productive capacity, after previous dull periods, has created disappointment at the failure thus far. An adequate explanation is that the increase in capacity has been greater, really much greater, than at previous times because the period of sustained activity which preceded the recent lull was the longest. It will require longer than formerly for demand to grow up to capacity. Indeed, it may, and probably will not, do so for many years, simply because capacity continues to increase even though

present capacity is not under pressure. Apart from the new erection which is based upon existing capacity being strained, there is another class of new erection, of a distinctly competitive character. This class of new erection has been strongly exemplified lately. Large steel interests, finding their profits per ton diminished, have sought new lines of production. One large steel interest has gone into pipe, another is going into wire, and another is going into tin plate and wire. Lake Superior ore interests, finding it difficult to sell ore at the high prices now ruling, are building many merchant furnaces simply to act as channels for the disposal of ore.

Thus it may be several years before existing capacity is strained as it was in the greater part of 1899 and the forepart of 1900, in the latter part of 1901, extending through 1902 into 1903, and from late in 1904 to late in 1907. This does not reflect upon the vitality or growth of demand, but rather upon the freedom in new erection which a series of high-pressure periods has engendered. The industry will likely not find its capacity really strained until a general industrial depression, some time in the future, puts an effective quietus upon new erection and affords time for the country to grow up fully to the capacity. Afterward, as in former cases, actual requirements will probably exceed capacity, and being suddenly expressed will create another boom, such as experienced in 1879 and 1899. Our nomenclature does not regard as "booms" the periods of strong and steady pressure experienced in 1902 and 1905-6-7.

Pig Iron.

Pig iron prices continued to decline in the forepart of May, from 50 cents to $1.50 per ton in different districts and for different grades. In the latter part of May the decline seems to have been arrested, and there has been much larger buying, although there is no regular buying movement, and at the close of the month the rate of buying remains less than either production or actual consumption. Usually, whether in a downward or upward movement, Southern iron finds a level first and it may be doing so in this case. At any rate, our last report noted a decline to $12, Birmingham, and the observation was made: "At this figure there is the first definite resistance to decline." A month later we may note that pig iron in the valleys has declined 75 cents for foundry, $1 for basic and $1.50 for Bessemer, whereas Southern iron is actually selling for second half delivery at the former price of $12, the only recession being that prompt iron can be picked up at $11.75, and possibly for iron slightly off the standard at $11.50. Sales of 5000 tons and 3000 tons of Bessemer have been made at $16, valley, and of 15,000 tons to 20,000 tons of basic at about $15, valley. We quote valley prices, 90 cents higher delivered Pittsburgh, as follows: Bessemer, $16; basic, $15; No. 2 foundry, $15 to $15.25; malleable, $15.25 to $15.50; forge, $14.50 to $15. Where a range is given the lower price is for moderate-sized lots for early shipment, the higher figure being for small lots or larger lots for extended delivery.

Steel.

There has been a general declining tendency, due to light buying and the continued decline in pig iron, but, noteworthy in the general conditions, open-hearth steel has remained relatively scarce and has declined but slightly. Roundly speaking, Bessemer steel has declined $1.50 in the month and open-hearth 50 cents, prices being quotable, at Pittsburgh, as follows: Bessemer billets, $25.50 to $26; sheet bars, $26.30 to $27; open-hearth billets, $28 to $28.50; sheet bars, $29 to $29.50, rods, $31 to $32.

Finished Products.

Plate prices have yielded fully $1 a ton, and against $1.55 for plates and $1.50 for structural shapes at last report we can report both at $1.50, with the probability that these prices would be shaded $1 a ton on attractive business. Black sheet products are being shaded by a few mills $2 a ton, and galvanized sheet products by $3 a ton. Merchant steel pipe is unchanged, the official price being 78 off on full weight, the actual inside price remaining 80 and 5 off for "catch weight"; wrought-iron pipe has declined one point, to an inside price of 75 and 5 off, for full weight, there being little "catch weight" iron pipe made. Steel bars have continued strong, and the agricultural implement interest, which hitherto have obtained concessions from the going market of $2 to $3 a ton, on their season contracts, have placed about 200,000 tons for second half and for the full twelvemonth at the full price of $1.45, Pittsburgh. The refusal to give a concession to this class of buyers is generally interpreted as indicating market strength. An interpretation which is worthy of consideration, however, is along the line that in former years the mills had a definite price understanding and could limit a concession to the class of buyers for whom intended, whereas in the present open-market conditions a concession given to one class might develop into a general break, so that mills have been chary of departing at all from the straight line. Wire nails and plain wire are being shaded $1 a ton.

Regular prices, f.o.b. Pittsburgh, are as follows:

Standard steel rails, $28 for Bessemer; $30 for open-hearth, f.o.b. mill, except Colorado.

Plates, $1.50 for tank quality, ¼ in. and heavier.

Shapes, $1.50 for I-beams and channels, 15 in. and under, zees and angles 2 x 3 and larger.

Steel bars, $1.45, base.

Iron bars, $1.55, Pittsburgh; $1.50 to $1.55, Philadelphia; $1.50, Chicago.

Wire nails, $1.85, base; plain wire, $1.65, base; galvanized barb wire, $2.15.

Black sheets, 28 gage, $2.40; galvanized, $3.50; blue annealed, 10 gage, $1.75 to $1.90; painted corrugated roofing, $1.70; galvanized, $3.00 per square.

Tin plates, $3.60 for 100-lb. cokes.

The Western Metallurgical Field.

Deficiency of Present Processes.

The unnecessary status of mining in Colorado and other parts of the West was charged directly to the inefficiency of metallurgy in a paper presented at the May meeting of the Western Association of Technical Chemists and Metallurgists by Mr. A. W. Warwick.

He arraigned metallurgists in the following manner: "Metallurgical work has not kept pace with the operations of the miner. The miners have developed tonnages of ore that must be seen to be appreciated, but which cannot be moved for the simple reason that the metallurgical treatment fails to pay a profit to the miner. Not because the ores lack value, but because the science of metallurgy has been in a stationary condition for years. Metallurgical research work has almost ceased in this Western country. Great deposits of ore and inconceivable tonnage of tailings are unworked, not because of their lack of value, but because of our lack of metallurgical skill. The time has come when these problems must be attacked in all seriousness and they must be solved if the mining industry and, of course that means also the metallurgical industry, is to maintain its prosperity." The statements are, in the main, correct and the conclusions logical. Perhaps not enough credit is given to inventors who are quietly seeking a solution of some of our problems, but on the other hand the problems have multiplied faster than they have been solved, so that at the present time the situation looks unfavorable.

It is true that the easily mined and easily treated oxidized surface ores have largely been treated by the smelting process and the smelting centers have constantly shifted to new fields as fast as the older ones began to yield ores not easily smelted. Instead of improving the smelting process to make it possible to treat economically the more refractory ores, and extract all of their metallic values, the smelter has avoided the problem and moved to more inviting fields. What may be called "by-

product metallurgy" has not been practiced as it must be if the smelting industry is to maintain its footing in the West.

The possibilities of electric-furnace methods of ore treatment were considered by Mr. Warwick as offering some hope for the future of lead and zinc reduction. In matte smelting, also, the electric current should find a possible application. The principal criticism of electric-furnace methods is that the plants are usually of small capacity and the capital invested for the tonnage of ore treated is large. Furthermore, Mr. Warwick believes that, as a rule, metallurgists know very little about electrometallurgical processes, and electrochemists know very little about metallurgy.

In electrolytic methods, however, Mr. Warwick finds hope for the future and even ventures a prophecy that "the problem of treating complex sulphide ores will be most satisfactorily solved by chlorine and electrolysis." While many details of the chlorination of sulphide ores still have to be worked out, it is believed they will be solved satisfactorily and in the Malm process it is hoped a long step in advance will be made toward bringing metallurgy up to the requirements of mining.

The Smelter Situation.

The final determination of the principal stockholders of the Modern Smelting & Refining Company to close the company's matte smelter near Denver has occasioned no surprise among those conversant with the situation. The smelter has led a precarious existence ever since it was blown in some months ago and the impossibility of getting ample supplies of ore has led to the final decision to close. Smelting ores in Colorado are a very scarce commodity, and a plant relying wholly on copper as a base for smelting must, under present conditions, have small chances of success. When matte has to be resmelted several times in order to bring it up to shipping grade and also to provide copper for the collection of the gold and silver in successive charges, cost of production must be excessive. It is reported that the present venture has cost its chief supporter some hundred thousand dollars.

In the meantime preparations are going forward for the blowing-in of the matte smelter of the North American Smelter & Mines Company at Golden, near Denver, and of another similar plant, near Alma, Col. It is reported that the Golden smelter has purchased the stock of ore and other supplies now on hand at the Modern plant.

The four smelting plants of the American Smelting & Refining Company, at Denver, Pueblo, Leadville and Durango are running at about half capacity and there is little expectation of any change in these conditions.

In view of what has been said it will be apparent that the constantly recurring reports to the effect that the International Smelting & Refining Company will build a plant in Colorado are nothing but rumors, wholly without foundation. In fact, it can be almost positively stated that such a plant will not be built.

In Utah smelting operations are far more active and the volume of business done much greater than in Colorado. The International Company is rapidly completing its plant for copper smelting at Tooele, Utah. It is expected that ores will be received about June 1 and that the plant will be blown in as soon thereafter as suitable supplies are on hand or are coming in steadily. Reverberatory furnaces will be used exclusively at first, although blast furnaces also may be installed later. In the design of the Tooele plant the International Company has had the benefit of experience gained in the best practice in copper smelting in this country and the new smelter should be successful in every way.

The plant of the American Smelting & Refining Company at Murray, Utah (lead), of the U. S. Smelting, Refining & Mining Company at Midvale, Utah (copper), and of the Garfield Smelting Company at Garfield, Utah (copper), are all in operation.

Agriculture and Metallurgy.

The disposition of smelter fumes is quite as live a problem as it ever was, and in Utah and Montana especially the companies give it considerable attention. The International Company, profiting by the experience of its competitors, is anticipating its day of trouble by placing experts in the field at its new Tooele plant to study conditions in vegetable and animal life before the smelter is blown in. Hitherto many of the litigants against smelting companies have broadly charged all forms of animal and plant disease to the presence of noxious fumes from the smelters. Hence the protective measures of the International Company.

The United States Company at Midvale is removing the SO_3 from its fumes by means of ZnO and by powdered CaO injected into the flues. Although the use of the former was at first more extensive and regarded as preferable, it is understood that at present quick lime is more generally used.

Although the Murray plant of the American Smelting & Refining Company is operating under an agreement to reimburse the farmers for smoke damage according to the awards of a board of arbitration, there is considerable dissatisfaction reported among the farmers and it is stated that the Murray plant will be forced to install efficient fume condensing devices to remove the SO_3 gas.

The suit of the United States Government against the Anaconda Company in Montana to force that company to install fume-condensing equipment is attracting considerable attention. The impression prevails in some quarters that the Government has undertaken a difficult work. Operations at Anaconda are on a scale not duplicated elsewhere in the country, and furthermore the Anaconda Company has made what is probably the most extensive and elaborate scientific investigation into the fume question that has ever been undertaken, and there can be no doubt that if there were a feasible commercial method of handling the fumes at Anaconda the company would gladly install it.

Mills.

Negotiations for the sale of the Golden Cycle properties, comprising mines at Cripple Creek, Col., and cyanide mills at Colorado Springs, will hardly be brought to a successful conclusion, in the opinion of the president of the company. An option, expiring June 1, was given to an English syndicate to purchase the Golden Cycle properties, but it has not been exercised. The Golden Cycle mill is probably the most successful cyanide mill in Colorado. It treats Cripple Creek telluride ores from its own mines as well as from others in the same district.

Since the absorption of the Boston Consolidated Company (Utah) by the Utah Copper Company, it is reported that the system of ore grinding used in the Boston Consolidated mill will be altered to conform to that of the Utah Copper mill. At the time these two plants were created, they attracted considerable attention by reason of the fact that such widely different systems of grinding were installed to prepare the same ore for subsequent concentration. The Utah Copper Company introduced graded crushing with rolls and chilian mills to accomplish the same purpose that the Boston Consolidated Company accomplished with Nissen stamps. The tonnage treated by the former system was twice that of the latter, but the percentage recovery of the latter was probably greater than that of the former.

Now that the Utah Copper Company's management controls both mills, it is understood that changes will be made as stated before, so that tonnage will be greatly increased. Some metallurgists regard it as unfortunate that what appears to be purely a commercial consideration should dictate the dismantling of a system which is giving good results from a metallurgical point of view. It has the appearance of sacrificing quality to quantity; of abandoning good metallurgy for immediate financial considerations. If it is true that the Boston Consolidated system effects a better recovery of copper, it would seem to be good metallurgy to continue to use it, but perhaps the demands of business do not always take small refinements into consideration.

Iron and Steel Institute Meeting.

The forty-first annual meeting of the Iron and Steel Institute was held in London on May 4 and 5.

The president of the Institute for the last three years, Sir Hugh Bell, occupied the chair during the first part of the proceedings. From the report of the Council it may be mentioned that the following gentlemen, representing various nationalities, have been elected honorary vice-presidents: Mr. John Fritz (United States), Dr. F. W. Lürman (Germany), Baron Fernard d'Huart (France), Mr. E. J. Ljungberg (Sweden), and Mr. William Kestronek (Austria).

Dr. Lurman, of Berlin, announced that Sir Hugh Bell had been elected an honorary member of the Verein Deutscher Eisenhüttenleute and presented him with the diploma.

The new president, the Duke of Devonshire, then took the chair. He is the grandson of the seventh Duke of Devonshire who was the first president of the Iron and Steel Institute in 1869. His first official duty was to present the Bessemer medal to Mr. E. H. Saniter.

The Duke of Devonshire then read his presidential address. He reviewed the development of the iron and steel industry since 1869. At that time the four great industrial nations of the world—Great Britain, Germany, the United States, and France—started almost equal, so far as their respective populations were concerned, on the great race for economic superiority. In that year the position of Great Britain in the production of coal, iron and steel was, however, one of absolute supremacy, but that enviable position of superiority was not one which that country could hope to maintain forever.

From 1869 to 1908 the increase of coal production has amounted to 144 per cent in the United Kingdom, 334 per cent in Germany, 1120 per cent in the United States, 170 per cent in France, and 85 per cent in Belgium. In the same time the increase in iron ore production has been 30 per cent in the United Kingdom, 500 per cent in Germany, 550 per cent in the United States, and 300 per cent in France, while the increase in pig iron production has been 66 per cent in the United Kingdom, 740 per cent in Germany, 830 per cent in the United States, and 147 per cent in France, and the increase in steel production 54 per cent in the United Kingdom, 400 per cent in Germany, 233 per cent in the United States, and 285 per cent in France.

The absolute figures of production in 1908 in tons are given as follows:

	United Kingdom.	Germany.	United States.	France.
Coal production.....	261,529,000	148,537,000	415,843,000	36,874,000
Iron ore production....	15,031,000	24,225,000	34,202,000	10,000,000
Pig iron production....	9,057,000	11,805,000	15,936,000	3,412,000
Steel production.....	5,380,000	11,186,000	14,120,000	2,713,000

England can only regain the position of supremacy occupied in 1869 by reckoning in with the United Kingdom the great dominions overseas. "When we compare the production of Great Britain alone with that of Germany or the United States we must necessarily fall short; but when we compare the British Empire and the volume of its trade and manufactures with those of any other competing countries in the world, we need have no fear of the comparison."

The president then took up the social side of the problem, and inquired how far the working classes of Great Britain have benefited by the great changes in industrial and social conditions since 1869. Conditions of work are now safer and more sanitary. Wages are better. Working hours are lighter. Housing is better. A host of improvements in traffic, lighting, education and public assistance have made for the comfort, health and enlightenment of the people. Further, there are free education, old age pensions, municipal tram services, housing schemes, public libraries, etc. "We may safely say that the working classes of this country at the present day are, on the average, 100 per cent better off than they were when this Institute was founded in 1869."

In the conclusion of his address the president emphasized that to be of service, scientific knowledge must be combined with practical experience. "There can be, no doubt, recalled by most of you instances of men who have been highly educated in science who in practice were absolute failures—exactly from what cause it is not easy to determine; but it undoubtedly is the case that some men highly trained in science—having left the practical side probably until too late—have upon entering the works failed to properly grasp the situation and apply themselves to the work. Hence the reason that many others who may have been indifferently educated have, by hard plodding and determination, succeeded in occupying the chief position in the country.

"Undoubtedly it is by a combination of theory and practice alone that success can be achieved, and it is to an institute of this nature, with a cosmopolitan membership such as this institute enjoys, that we must look in the future, as we have done in the past, for their guidance and information, without which no industry can hope permanently to prosper."

Mr. S. Selby Bigge, of Newcastle-on-Tyne, then presented a paper on the "development in the production of electric power; its application and bearing upon the iron and steel industries." Mr. E. G. Schemer and Dr. R. Drawe, of Saarbrücken, presented a paper on the "economy and design of modern reversing rolling-mill steam engines." Both papers were discussed together.

A paper by Prof. W. Borchers, of Aachen, Germany, on "the Girod electric furnace" was read at the session of May 5 and discussed at some length. Borchers considers the Girod furnace to be the simplest electric furnace, but this was questioned by those who participated in the discussion.

A paper by Dr. J. O. Arnold and Mr. A. A. Reed on "the chemical and mechanical relations of iron, manganese and carbon" was next read and discussed.

This was followed by a paper of Mr. Edward G. Herbert, of Manchester, on "the cutting properties of tool steels," a paper by Mr. Andrew McWilliam and Mr. Ernest J. Barnes on "some physical properties of 2 per cent chromium steels," and a paper by Mr. Harold Moore on "the A2 point in chromium steel." The latter two papers were discussed together.

The papers by Mr. Sydney A. Gray on case-hardening; by Mr. Adolph Kroll, Jr., on the crystallography of the iron-carbon system; by Mr. C. A. M. Smith on the elastic breakdown of certain steels; by Mr. D. M. Levey on the constitution of cast irons and carbon steels, and by Mr. G. Tagayeff on the homogeneity of metals were read by title.

Abstracts of all these papers together with accounts of the discussions will be given in due course in the Synopsis in this journal.

CORRESPONDENCE.

Temperatures in Electric Steel Refining.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—The letter by "M. N.," on page 228 of your April issue, shows an increasing interest in electric-steel melting, even on the part of those engineers who until recently understood little what the electric steel furnace can do and is doing. It must, therefore, be appreciated from this point of view. Its material contents, however, should be modified. Whilst the writer suggests that the reducing atmosphere is the main feature of the electric furnace, it is a well-known fact among all electric steel melters that the high temperature has at least the same important bearing, if not a higher one, upon the refining operation.

I have been present at the same heats referred to in the letter and can say first that the temperature measurements were of only a relative value, that they gave the difference between temperatures observed, but hardly the absolute values of temperature. The Bureau of Standards in Washington writes about the Mesure-Nouel pyrometer used by Professor Richards: "It can hardly be termed a reliable pyrometer. Differ-

ent observers will differ from one another by over 100°, and even the same observer is liable to differ considerably on different days and under different conditions of illumination. It is a little better than the unaided eye." Thus Professor Richards, who was very familiar with his instrument, could tell about the differences in temperature which he observed.

Secondly, when three extremely hot spots, as they are formed by the electric arcs, are distributed over a surface, the temperature of the latter cannot be homogeneous and it is not likely that during the short moment the door of the furnace is open the pyrometer was directed just upon the hottest places, as the arc itself would affect the observation.

Concerning the effect of temperature I refer to a former article in your journal (*Electrochemical and Metallurgical Industry*, 1909, Vol. VII, p. 115) where I indicated in rough lines, both the physical and the chemical effects of high temperatures on the slag as well as on the metal. It should always be kept in mind that the refining of metal underneath a slag is a reaction which mainly occurs along the boundary surface between slag and metal, i.e., what physical chemists call a "heterogeneous reaction." The velocity of it depends on the speed with which the reacting materials are carried from both sides to this boundary surface. Diffusion and mechanical movement contribute to increase this speed and they are themselves increased by heat.

Another class of reaction is going on in both phases (slag and metal as two liquids, not mixed with one another, are each termed a separate "phase"), as, for instance, the reaction of dissolved carbon or carbides on dissolved oxygen or oxides in the metallic phase or the transformation of manganese sulphide into calcium sulphide in the slag phase. These so-called "homogeneous reactions" depend directly upon the temperature. At lower temperatures their velocity is doubled by an increase of 10 centigrades, and at higher temperatures the effect which has not yet been measured, seems to be still more pronounced. I have also shown in the paper mentioned above how the local distribution of temperature in the bath causes the metal to move and need not refer to it again at this place.

To sum up, the high temperature produced right above the metal and the slag yields results that have not been obtained by heating with fuel. And these results are so conspicuous that some people have even supposed the working of some mysterious not yet discovered forces. But there is nothing of that kind. And as the reacting elements are in the main the same as in fuel furnaces aside from the fact that their proportion can be altered so that slag of higher basicity can be employed, the temperature remains as the most important factor for explaining the advantage of the electric furnace, as under its influence the elements can combine and dissociate in a different and quicker way than known before in steel refining.

Chicago, Ill. RICHARD AMBERG.

* * *

To the Editor of Metallurgical and Chemical Engineering:

SIR:—I am glad that my brief remarks have elicited Dr. Amberg's interesting letter.

What I endeavored to emphasize was that the *average* temperature of the steel is not very different in the electric furnace and in ordinary oxidizing furnaces. This is proven by the temperature measurements referred to, and to prove this, the relative or comparative temperature indications of the Mesure-Nouel pyrometer are sufficient. It seems to me worth while to point out this fact, as I know there exists a popular erroneous idea that the *average* temperature in electric steel furnaces is so much higher than in ordinary furnaces.

Dr. Amberg is, of course, correct in emphasizing the great importance of the temperature maxima in the slag, just below the three arcs. I agree with everything he says about the effect of these temperature maxima. But I would emphasize more strongly than he does the possibilities of certain purifying reactions in the electric furnace, due to its neutral or reducing atmosphere, which could not occur in the oxidizing atmosphere of the open hearth.

In other words, while Dr. Amberg is right to point out the possibility of using a slag of higher basicity, because it can be kept liquid at the higher temperature under the arcs, yet in this respect electric refining is only quantitatively different from refining in the open hearth, and this does not exhaust the possibilities of the electric steel furnace as in it the reactions can be made qualitatively different from the reactions in the open hearth. I refer especially to the removal of sulphur as calcium sulphide in the electric furnace (which reaction would be impossible in an oxidizing atmosphere, since the calcium sulphide would be changed to sulphate, which would react with iron, and the iron sulphate would pass back into the metal bath) and to the removal of phosphorus as phosphide (which would also be impossible in an oxidizing atmosphere).

Of course, these reactions are strongly accelerated by high temperature. That a high temperature is required is indicated by the fact that the formation of calcium carbide seems to play a part in the removal of the sulphur as calcium sulphide, and we know that the formation temperature of calcium carbide is quite high. It is, of course, of fundamental importance that the zone of maximum temperature is produced just where it is needed—in the slag below the three arcs.

Chicago, Ill. M. N.

* * *

Estimation of Water in Petroleum.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—In your May number, on page 283, under the heading "Estimation of Water in Petroleum," a method is mentioned that the writer developed nine years ago in Melbourne, Australia, and described in an article to the Victorian Society of Chemical Industry.

Subsequently, at the request of Prof. Orme Masson, of the Melbourne University, and Mr. Hake, chief inspector of explosives, such an apparatus was installed by me in the Melbourne University for general use and in the Government Laboratory for the estimation of moisture in cordite, etc., and the chemists there were shown its use.

I did a very great deal of work and found the apparatus very useful for all sorts of substances, providing a very positive way of estimating moisture.

An extract from a letter but recently received from Professor Masson may be of interest.

"It may interest you to hear that the use of calcium carbide for determining moisture, which you were the first to describe and which was applied by Dupré four years later to the analysis of cordite, has been much in evidence here lately. There has been a rather important conference between representatives of the Pastoralists and the Shearers on the question of shearing wet sheep, and it becomes necessary to make an extended series of observations of the amounts of moisture in wool.

"The method adopted was essentially yours, though modified in detail, and (as chairman of the Scientific Advisory Committee) I know it has proved of great use. One of my students has also been using the method in a different research, which will shortly be published in one of the English journals, and he will, of course, refer to you."

As I gave the method out free to all, without let or hindrance, the principle of "Honor to whom honor is due" should at least give me the credit by the use of my name whenever the work done by me proves valuable to the user.

Clarksburg, W. Va. HAROLD A. DANNE.

* * *

Blast Furnace Operation.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—In Mr. E. B. Cook's excellent article on "Causes of Non-Uniformity of Blast-Furnace Operation" in your May issue is a statement that I wish to challenge. He says, "as a proof of dry blast's help in preventing this trouble, *it is noticed that*

the dry-blast furnaces last year surpassed all other furnaces in average work." But I have yet to find the record of a furnace that has been doing better work than our East furnace during the past nine months. For the six months ending March 31, 1910, the average coke per ton of pig (2268 lb.) was only 1977 lb. This figure is based on railroad weights of coke delivered into the coke chutes, and on the tons of merchantable iron made.

The furnace has been on basic iron for the past nine months and has been in blast since July 15, 1908. There has been no serious trouble during this blast except the breaking of the bell rod and the losing of the main bell in June, 1909, when the blast was off for about 48 hours.

The furnace is 75 x 18 x 12 and has eight 5½-in. tuyeres. The blast temperature is carried evenly by blowing through two stoves, and the average temperature is about 1100°.

The average silicon in the pig iron for the six months was 0.75 per cent, and the average sulphur 0.029 per cent. The theoretical yield of the mixture was 55.06 per cent (based on 94 per cent iron in the pig iron) and the actual yield was 54.04 per cent, making the loss in yield only 1.02 per cent. This low loss for flue dust and all other wastes is very low for the 70 per cent of Mesabi ores in the mixture, and shows that the furnace worked regularly. Of course, there were slips, but on the whole the furnace ran smoothly.

The average limestone per ton of pig was 878 lb. The total tonnage of pig (merchantable basic) for the six months was 49,420, or 271½ tons per day. The very severe winter weather in January and February cut down the tonnage and also ran up the fuel consumption to 2029 lb. and 2054 lb., respectively.

The best month's work was 9000 tons, made in December, on 1906 lb. of coke per ton of pig. The loss in yield was only 0.30 per cent, with a 55.34 per cent theoretical mixture.

Have the dry-blast furnaces done better than this?

Mr. Cook's suggestion about blast-furnace managers meeting together to discuss furnace practice is a good one, and it is likely that a practical plan for such meetings of the managers and superintendents of merchant furnaces would be readily adopted. Lately the superintendents of many Ohio furnaces have been exchanging visits with mutual benefit.

The Columbus Iron & Steel Company. R. H. Sweetser,
Columbus, Ohio.

* * *

To the Editor of Metallurgical and Chemical Engineering:

Sir:—The writer was much interested in reading the article in your May issue by Edward B. Cook on the causes of non-uniformity of blast-furnace operation.

In this picture of a blast-furnace man's troubles, much stress is laid on furnace lines and accumulations on the bosh, but the author does not mention one of the most important causes of bosh accumulations, and that is the construction of the furnace.

To make this point clear, it may be stated that there are two methods in general use by blast-furnace engineers for protecting the walls of a blast furnace. This protection is especially necessary in that part of the furnace containing the melting zone—that is, from the tuyeres to the top of the bosh, and it is in this part of the furnace that Mr. Cook states most of the trouble of "building up" occurs.

The two systems of wall protection may be described as the internal and external system.

Internal Cooling.—In the internal method, water-cooled plates of cast iron with enclosed wrought-iron pipes, or hollow cast bronze or copper plates through which water circulates, are laid in the brick walls of the furnace at intervals of from 12 in. to 30 in., depending upon the degree of cooling thought necessary. One practice is to lay the plates flush with the inside of the wall; another to set them back to give a cover of brickwork 4½ in. thick, which is often increased to 13½ in. where less cooling effect is desired.

The purpose of this cooling is to hold the furnace to its original shape, by maintaining the thickness of the walls. The results with this system, as far as protection to the walls is concerned, is very satisfactory and the thickness is maintained very well where these water-cooled rings are placed. In between these plates, however, the brickwork burns away, forming recesses where the building up takes place, referred to by Mr. Cook, and the cooling plates form shelves that assist the accumulation by furnishing a support.

When a furnace is blown out after being operated with a bosh wall protected by the internal cooling system, these plates may be seen in most instances entirely bare of any brickwork covering and like a series of annular shelves protruding from the wall. That these plates are frequently bare of any brickwork, the writer has also ascertained by cutting into the bosh wall of a furnace that was temporarily shut down for the purpose of replacing some of these cooling plates that had been leaking.

External Cooling.—With the external method of cooling, a much thinner brick wall is built for the furnace wall at the start, and this is protected by a plate-steel jacket on the outside, cooled with a water spray, the drippings falling into a water-trough at the base of the steel jacket. The cooling is, therefore, practically uniform over the surface of this jacket and hence over the wall. There is, therefore, no projections formed on the inside of the furnace to act as shelves or pockets for holding material to be discharged at irregular intervals with disastrous results to the temperature of the furnace hearth and to the furnace working.

Cooling Above the Melting Zone.—This method of external cooling with thin walls has been in use in Germany for a number of years, for that part of the furnace called the "inwall"—that is, from the top of the bosh to the tunnel head.

During the past year two furnaces have been built in this country with thin inwalls and external cooling and so far the results have been very satisfactory. The most common form of cooling for the furnace wall above the melting zone, however, is the internal system, where any cooling is used or thought necessary. The external cooling with thin walls is, therefore, experimental when used in the upper part of the furnace wall, but it is the older practice for the protection of the bosh wall of the furnace below the bosh line.

How to Build a Furnace.—The statement made by Mr. Cook that "No one knows how to build a furnace right" needs some qualification. He cannot mean that no successful furnaces are built, for if that were true, how could any money be made in the iron business?

The condition of furnace construction to-day is the result of very rapid development and improvement in the past 25 years in the direction of increasing the output per stack and reducing the labor costs of operation, but, with that development, new problems of construction have arisen.

These problems vary with local conditions—that is, the ores, fuels and fluxes in use; hence, a construction that may be right for one locality may not give the best results in another. In the same way the management of a furnace must be adapted to local conditions, to the furnace actually in use, the kind of labor available, and the materials on hand for use.

What we can join with Mr. Cook in stating is that the ideal blast furnace has not been built, but it is the goal toward which all blast-furnace engineers and managers are striving and the writer believes much progress has been made toward that end.

David Baker.

Ladd & Baker, Inc., Philadelphia, Pa.

Concrete flues have not proved successful when used in smelters for conducting roaster fumes to the stack. Disintegration of the cement coating first occurs, with subsequent sloughing off in large cakes. This ultimately exposes the iron reinforcement, which rapidly weakens under the attack of the acid gases, and the final collapse is only a question of time.

A New Resistor Furnace.

BY F. A. J. FITZGERALD.

There are certain obvious objections to the use of an arc furnace where it is desired to heat the material charged therein by means of radiation. This has led to the development of a furnace in which the charge in the crucible is heated by radiation from resistance, which in some forms may be described as the sealing of the reaction or melting chamber.[1]

During the last few months some interesting experiments have been carried on with a 150-kw furnace of this type at the plant of the Hohenlohe Werke, in Upper Silesia, Germany. The furnace was used for certain experiments in the production of zinc. Fig. 1 is from a photograph which gives a general view of the furnace while actually working. Fig. 2 is a top view of the furnace after the current has been cut off and the cover removed.

Some careful determinations of the heat losses were made, and from these a good notion of the efficiency of the furnace can be obtained. In one experiment lasting several days a bath of molten pig iron was held at a constant temperature of 1250° C. to 1260° C., and it was found that the rate of generation of energy necessary for this was 33 kw. If, then, the furnace is worked at 150 kw and a temperature of 1250° C. maintained, its efficiency would be 78 per cent.

In another experiment, lasting 12 hours, the temperature of the bath was maintained at 1400° C. to 1450° C. with a rate of generation of energy amounting to 42 kw. Therefore, working at full load and at this temperature the efficiency would be 74 per cent.

This last determination is not so accurate as the first for the furnace had not reached a steady condition. This was found out by taking the temperature of the walls, which showed that during 12 hours the wall temperature had increased more than 100° C. and was still rising. If the experiment had been carried on until the temperature of the walls was constant the rate of generating energy would have been less than that actually determined, and consequently the efficiency would have been greater than 74 per cent.

In certain forms of electric furnaces it is possible nearly to eliminate the loss of electrically generated heat by surrounding the furnace with an envelope of burning gas. In a modification of the type of furnace under consideration this is accomplished by building the furnace with double walls, bottom and cover, between the inner and outer parts burning fuel

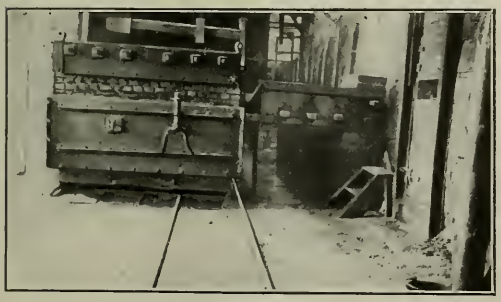


FIG. 1.—ELECTRIC ZINC FURNACE.

which keeps the temperature there the same, or nearly the same, as that in the working chamber.

In this way there is no leakage of heat from the working chamber to the surroundings, consequently all the electrically generated heat is employed in doing useful work. Extended experiments embodying this principle, and made on furnaces of large size, have shown that it introduces very great economies.

The useful life of a resistance furnace of this type is found to be of satisfactory duration. Although the longest continuous run of the furnace shown in the illustration was only 21 days, this was because it was necessary to shut down in order to make changes in construction which had nothing to do with the construction of the furnace itself.

Other furnaces of a similar type have been run continuously for nearly two months, and there is no reason why such a furnace could not run continuously for several months. Again,

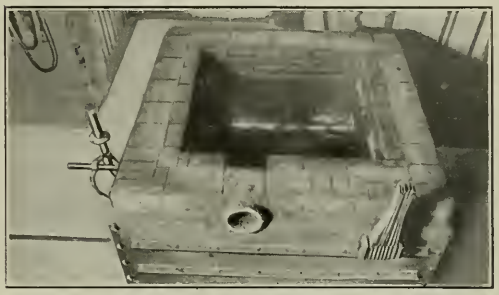


FIG. 2.—FURNACE OPEN.

it has been found possible to run these furnaces and let them cool down repeatedly without having to make any repairs.

While the furnace described was specially designed for zinc smelting, experiments have shown that it also has considerable value in several other metallurgical operations.

FitzGerald and Bennie Laboratories,
Niagara Falls, N. Y.

Blast and Down Draft Sintering.

BY REDICK R. MOORE.

The use of the general term "Bessemer roasting," as proposed in METALLURGICAL AND CHEMICAL ENGINEERING of May, 1910 (page 222), for the various processes using pressure or partial vacuum to force a blast of air through a mass of sulphides for their desulphurization and agglomeration seems to me to be unwarranted.

In the first place, the use of any man's name for the distinction of a process which he did not invent and with which he had nothing whatever to do is to be deprecated.

In addition, Sir Henry Bessemer was not the original inventor of the process of converting pig iron into steel, which is frequently referred to as the Bessemer process, but only of some of the apparatus used for the operation of the process. As was proven before the U. S. Patent Office in interference proceedings between Bessemer and Wm. Kelly, of Kentucky, in 1857, Kelly had operated the process eight years before Bessemer conceived it, and priority of patent was awarded to Kelly.

The recarburation of the converted metal by Spiegeleisen was invented and patented by Robt. Mushet, and it was this improvement that made the process practicable and valuable.

Mushet in writing of the "so-called Bessemer process" says "its true name is the Martin process."—(London *Engineering*, May 10, 1861).

The earliest reference that I have been able to find of the somewhat analogous process of copper matte converting is in the *American Journal of Mining* of Sept. 1, 1866, in which it is stated that Dr. F. Leclere, of Paris, patented blowing copper to metal by "the injection of air into the metallic bath . . . obtained by melting raw ores."

[1] See review of Patents by John Thomson and F. A. J. FitzGerald, Metallurgical & Chemical Engineering, May, 1910, p. 289.

The first actual operation of the matte converting operation appears to have been done by John Hollway in England in 1878 (*Jour. Soc. of Arts*, Feb. 14, 1879), and it was finally brought to a successful issue by the skill of David and Manhès in 1883.

The processes of desulphurization and agglomeration of fine sulphide ores under discussion are entirely dissimilar to the converting processes in two very essential conditions. Firstly, converting presupposes that the material operated upon is in a molten condition and, secondly, that it is finished or converted to a finished metallic molten product, whereas in the processes we are considering it is essential that the ores or products treated should be in a comparatively finely divided condition at the start and that they shall not be fused in the operation, as otherwise the process is ineffective, as the light blast used will not penetrate a fused mass and the desulphurization will be incomplete.

The fact that the operation is carried on by the fuel contained in the ore itself is beside the point. Other processes which no one would think of calling converting are carried on in the same manner without any extraneous fuel; for example, pyritic smelting, pyrite burning in shelf roasters and copper-iron sulphide ore roasting or calcining in furnaces of the McDougall or Wedge types.

Pyrite smelting is much more analogous to copper matte converting in that a blast of air under pressure is oxidizing a falling shower of molten matte in the furnace stack. Pyrite burning or McDougall calcining is performed by the passing of air over the beds or layers of ore upon the shelves or floors and through the stream of fine ore falling from one shelf or floor to another in the furnace without any extraneous heat, once the operation is started and as long as the ore is supplied to and caused to pass through the hearths—with proper precautions.

The distinctive features of these processes are the character of the products. In roasting the product is understood to be pulverulent and when fuse boxes or their equivalents or strong finishing heat is used the changed products are called "fused ore," or "sintered roast," according to the condition of the products. In pyritic smelting the products are matte and slag, although practically the same desulphurization as in roasting has been attained. In converting the product is understood to be a refined metallic product, although it may require further refining under certain conditions and the product is always molten. Although the operation may be stopped before a finished product is obtained when some limiting term is employed, such as "converted to white metal," "gas copper finish," etc.

In the Heberlein and other similar processes the distinctive features are the light pressure of the blast used and the condition of the product, which, while not smelted, has been brought to the condition of incipient fusion, and has a honeycombed structure which we call sintered, by the action of a blast of air under pressure through the mass, some portion of which must have been previously brought to a temperature at which the ore would ignite under the conditions maintained.

For this class of processes, it seems to me that the name blast sintering would be appropriate and fully express their character and distinctive action and product.

The product of the Dwight-Lloyd process is somewhat similar to that of the Heberlein, Savelsberg, etc., differing only in that it is in thin layers and more porous, and on that account generally better desulphurized, but the process differs radically in the method of admitting the air and should have a distinct name.

I think down draft sintering will properly describe and differentiate it from all other processes.

While there are other methods of sintering the distinguishing adjectives herein suggested are at least as readily used as any and are far more descriptive of the process—which is neither converting nor roasting nor a combination of the two, as the terms are ordinarily used by metallurgists.

New York City

The New Blast Furnace of the Bethlehem Steel Company—II.

By W. S. Landis, M.S.

(Concluded from May issue, p. 231.)

The blast furnaces of this company are grouped on a narrow strip of land running east and west between the old and the new rights of way of the Lehigh Valley Railroad. This strip is just sufficiently wide to accommodate the engine house, ore bins and cast houses of the older furnaces. With the advent of new furnace construction the first large furnace "E" was arranged to run in an east and west direction, contrary to the others.

The furnace "D" being placed between the old and the new furnaces had to be most compactly arranged, as "E" cut it off from extending in any direction but across the narrow strip of property allotted to the blast-furnace division.

This compact arrangement is shown by the general ground plan, Fig. 6.

The ore, stone and coke pockets on the right are the ones

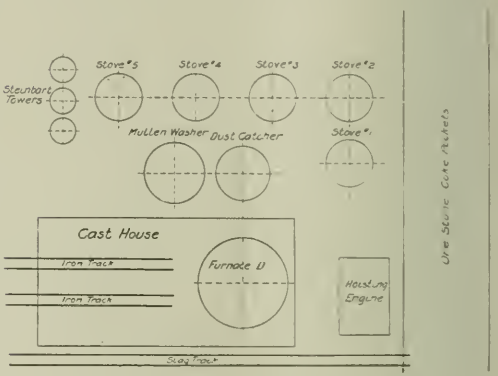


FIG. 6.—GENERAL PLAN OF BLAST FURNACE PLANT.

shown in Fig. 1 of the May issue. The gas-driven blowing engines and dynamos will be placed in a new power house to be erected on the site of the steam shovel in this first photograph.

It is now the intention to electrify all the shops and smaller rolling mills of the immense plant, the power being generated in this new engine house. With the completion of the four large furnaces and this engine house every boiler plant in the works will be dismantled with the exception of the blast-furnace boilers burning from arc gas to supply the steam hammers in the forge shops.

This will require a power generation which will exceed 25,000 hp.

Fig. 7 shows the general lines of the furnace stack. Its principal dimensions are as follows: Height, 90 ft.; diameter of hearth, 15 ft. 6 in.; diameter of top of bosh, 22 ft.; diameter on stock line, 16 ft.; depth of hearth, 8 ft.; height of bosh, 12 ft.; 10 tuyeres, 6 in. in diameter.

The hearth is jacketed with a heavy cast-iron, water-cooled plate extending 6 ft. above and below the center line of the iron notch.

Forty and one-half inches of brick comprise the lining of the hearth.

The bosh walls are 22½ in. thick surrounded by a water box, shown in detail in Fig. 8. A somewhat similar box has been in use on furnace "E" for several years, no trouble having been experienced with this system of cooling so long as care was taken to insure a constant supply of water to it. The inside walls were smooth and regular after blowing out, the ridges

and steps so prominent with plate-cooled boshes being noticeably absent. In one campaign of furnace "E," when through careless attention the water supply was interrupted over one section of the box and a hot spot developed, it was found quite difficult to overcome it as the water when again turned on refused to flow down over the hot plates.

The system of water-cooling employed by Scott in his new furnace bears considerable resemblance to this water-box system which has been in use by the Bethlehem Steel Company for several years.

Above the mantel are placed seven courses of cooling plates 3 ft. apart, 12 plates to the course. A steel wear plate extends downward 12 ft. from just above the stock line.

For charging, the usual inclined bridge carrying balanced skips of 125 cu. ft. capacity is used. A 200-hp motor driving a 6-ft. drum furnishes the power for hoisting. A modified form of the Baker top is used on the furnace, the essential difference being in giving lesser angle between successive positions of the distributor than is found with the ordinary top. In this top considerable lap is given to the distributions.

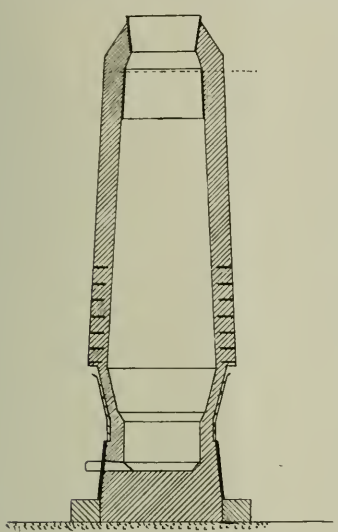


FIG. 7.—GENERAL LINES OF BLAST FURNACE STACK.

Several years ago the distributor on furnace "E" gave considerable trouble by refusing to revolve, although the indicator recorded proper distribution. The result was that the inwall was worn almost through before the trouble was located. It was so far gone that it was either a question of blowing out the furnace or else attempting a repair with the furnace in blast. A patch was decided upon. The furnace was blown down about 40 ft. and 6 ft. to 8 ft. of green sod was rammed in on top of the hot stock as tightly as possible. With this plug in place bricklayers descended 25 ft. down the stack inside and tore out the worn brick, replacing them with new, the work being so skillfully directed that not one was even overcome in the slightest by the furnace gas, though it is true that on account of the great heat the shifts were very short. The furnace ran for several months afterwards on this patch before being blown out. The distributor on "D" has been designed to absolutely avoid any such stoppage of revolution.

The cast house is 56 ft. wide and 111 ft. long. A 25-ton traveling crane serves the casting floor and has its run way extended some 10 ft. beyond the outer end of the house. On a prolongation to the inclined bridge at the top of the furnace is bracketed an overhanging crane with both transverse and longitudinal movement, for serving the top. To enable this crane to reach the ground a panel of the cast-house roof, including a truss, is made removable, a vertical from the crane dropping well in front of the furnace in the cast house to the hot-metal tracks shown in Fig. 6.

The greater portion of the pig iron made by the Bethlehem Steel Company goes direct to the basic steel furnaces, but a smaller portion reaching a casting machine of the Davis type (modified by the steel company to avoid water chilling). The ladles are brought in on two parallel tracks underneath the floor off the cast house. Six ladles can be accommodated at one time, the flow of metal being regulated by the gates and runners shown in Fig. 3 of the May issue of this journal. The slag track is outside of the cast house, the track being extended sufficiently far back to permit of the storage of a car for the reception of clean-up material, etc., from around the furnace.

The ores used by the company require a high temperature for successful smelting.

To this end five 22 ft. x 100 ft. McClure type hot-blast stoves are in use. To ensure their working at their maximum efficiency for a long period of time a thorough cleaning of the gases had to be assured.

After leaving the furnace the gas passes through a conical dust-catcher 25 ft. in diameter. From this it was originally planned to lead the gases through two Mullen washers in series before going to the stoves.

This arrangement was later modified to include only one 28-ft. Mullen washer and in place of the second to use a series of Steinbart towers.

These towers, three in number, each 12 ft. in diameter and 75 ft. high, are connected in multiple between the Mullen and the stoves.

The essential feature of these washers is the use of a rapidly revolving stream of water falling on a rain-making distributor. This insures a very uniform distribution of drops throughout the whole area of the tower, a test made with buckets set at various places in the bottom of the tower showing practically simultaneous filling of them all. A by-pass to the stoves is provided for the gas in case the washers are out of commission. No valves are used in this distribution of gas around the Steinbart towers; instead, the flow is controlled by the emptying or filling of the siphons shown in Fig. 4 of the last issue. At the present time the towers are not yet in operation; there is no question of their cleaning the gases, but the amount of moisture carried over by the gases might be a serious cause of trouble in the future.

The gas is led from the cleaning system directly to the stoves. When all five are in commission four are on gas four hours and one is on air one hour. The question has arisen as to whether there is not too liberal stove capacity when all five are working. With 400,000 brick in each stove, the passage of the blast through one for an hour will cause a drop in temperature of over 100° C. (average drop for the whole mass of brick). This is a rather large drop for uniform conditions regarding blast temperature and for minimum wear and tear on stove lining. It seems that the utilization of one of the stoves as an equalizer would settle the difficulty, certainly much more efficiently than the use of the McCarthy equalizing valve as at present installed in the hot-blast main.

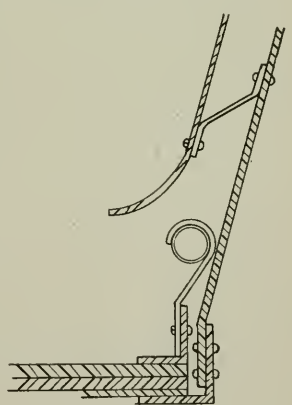
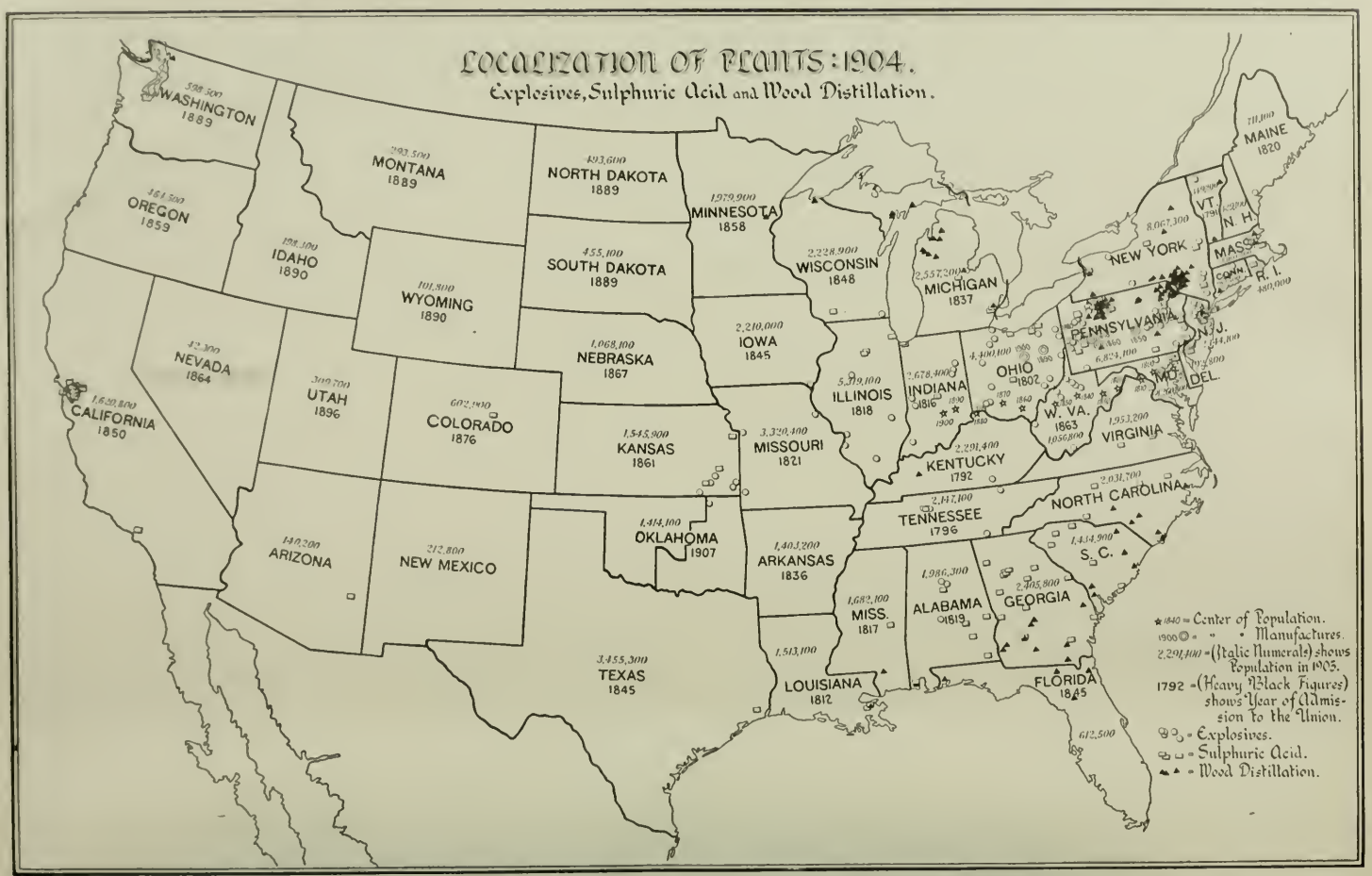


FIG. 8.—DETAILS OF WATER BOX.

On the whole it is a question if these is another furnace in the country more complete in details and appointments than this one, representative as it is of the $13,000,000 worth of improvements now under construction at this Eastern steel plant. In conclusion the author extends his most sincere thanks to the general manager, E. G. Grace, for the many courtesies extended during the preparation of this description.



MAP OF SULPHURIC ACID, EXPLOSIVE, AND WOOD DISTILLATION PLANTS IN UNITED STATES.

Map of Sulphuric Acid, Explosive, and Wood Distillation Plants in the United States in 1904

In Prof. Charles E. Munroe's address, delivered before the American Institute of Chemical Engineers, on the chemical industries of America, he mentioned (page 269 of our May issue) a map which he had, through the courtesy of the director of the Bureau of Census, had prepared of the United States showing the location of the establishments, both principal and subsidiary, manufacturing sulphuric acid, those making explosives, and those engaged in wood distillation, each being a typical industry, and the sulphuric acid industry being generally accepted as of fundamental importance. This map is now reproduced on page 320, through the courtesy of the American Institute of Chemical Engineers.

From this chart it appears that 13 States and Territories, namely Arkansas, Idaho, Montana, New Mexico, North Dakota, South Dakota, Utah, Iowa, Nebraska, New Hampshire, Nevada, Oregon, and Wyoming, containing 7,648,000 out of the 76,303,387 inhabitants of the continental United States in 1905, or over 10 per cent of the whole, did not possess a single establishment devoted to any one of these industries.

Considering sulphuric acid only, which is so important an industry that it has frequently been referred to as an index of the state of civilization of a people, we find that 23 States and Territories, namely, the 13 just enumerated, together with Delaware, District of Columbia, Kentucky, Maine, Minnesota, Missouri, Oklahoma, Washington and West Virginia, containing 19,562,200 population, or 25.6 per cent of the total, did not possess a single sulphuric acid plant within their borders.

Turning now to the East, we find that 11 out of the 13 original colonies, viz: Connecticut, Georgia, Maryland, Massachusetts, New Jersey, New York, North Carolina, Pennsylvania, Rhode Island, South Carolina and Virginia contained 30,695,000 population, or 40.2 per cent of the total, and 100 sulphuric acid factories, or 67.1 per cent of the total number existing in the country. Analysis of the statistics of the separate States shows that the number of these establishments does not follow the population, Georgia, for instance, with about one-fourth the population of New York, having twice the number of sulphuric acid factories that New York had.

Prof. Munroe remarks that the number of establishments in an industry is an unsafe criterion as to the magnitude or importance of that industry, but this feature has been chosen as lending itself most easily to graphic demonstration. He had, therefore, assembled, by geographic divisions, in Table VII (our May issue, p. 270) data for the quantity of sulphuric acid produced, and inspection of this leads to much the same result as to that which was drawn from the consideration of the distribution of the establishments.

As Dr. Munroe points out, all investigations show that there is an enormous extent of fairly well-populated area in this country yet awaiting development by the chemical engineer.

Chandler and Raymond Dinners.

On the evening of April 30 two memorable banquets were held, one at the Waldorf-Astoria in New York in honor of Dr. Charles F. Chandler, the other at the Plaza Hotel in honor of Dr. Rossiter W. Raymond. Both were very well attended, very successful, and will long be remembered.

At the Chandler dinner Dr. Schieffelin was the toastmaster. The large bronze bust of the guest of honor was not quite ready, but was presented to Columbia University by Professor Bogert in a graceful speech and accepted by Dr. Butler. The smaller replica was, however, finished and was presented to Mrs. Chandler, who in turn presented it to the Chemists' Club, where it will be one of the features of the new club house.

At the Raymond dinner Dr. James Douglas presided, who presented to Dr. and Mrs. Raymond a magnificent silver service, while Mr. W. L. Saunders, in a graceful address to Mrs. Raymond, presented her with 47 American Beauty roses, one for each year of her married life.

An Efficient Laboratory Electric Hot Plate.

The use of the electric hot-plate has contributed greatly to the improved conditions under which the most of the laboratory work of to-day is conducted. It has superseded the old coal-fired hot-bath and the oil or gasoline stove that was formerly so common in smelter laboratories and has provided a means of working under cleanly and comfortable conditions. Where electric power is reasonably cheap, the electric hot-plate can be made very efficient for a large volume of work, and since most of the former objections to its use have been overcome it is growing in use and favor.

High first cost and high expense of operation were originally the objections to the use of the electric hot-plate, and these were multiplied greatly in cases where capacity for a large volume of work was required. But with the decreasing cost of electric power and the ingenuity of the chemist who wants to improve conditions in his laboratory, it is not uncommon to find in use efficient electric hot-plates made at the plant from materials already on hand or easily obtained.

Following is a description of an electric hot-plate in use in the laboratory of the Empire Zinc Company, Canon City, Col. The device as used at present is an evolution from former apparatus that had shown weak spots. The plate is 34 in. x 17 in. and has a thin sheet-iron base on which is a piece of asbestos board ¼ in. thick. This is covered with sheet mica on which rest the heating coils. Above the latter is additional insulation in the shape of sheet mica. Finally, the top, which is of ½ in. planed cast iron, completes the apparatus. The whole can conveniently be placed on standards of any desired height, and forms a plate about 2 in. thick and of the area given above.

The construction of the heating coils is the interesting feature of this plate. They consist of fire-clay bars with No. 18 German silver wire. Iron rods, wrapped with mica, were tried originally, but proved unsatisfactory. The fire-clay rods now used are made according to a pattern designed for the purpose. They are 15⅞ in. long and rather triangular in cross-section. The dimensions of the latter are 1¼ in. x ¾ in., the larger dimension being the base of the triangle and the smaller its height from base to apex. The surfaces of the rods have slight lateral corrugations or grooves to facilitate the winding of the wire and to hold it in place.

The rods are wound in pairs and there are four pairs in the plate. They are laid with their length extending the width of the plates, and are spaced about 4 in. center to center. Four switches are provided, one for each pair of rods—so that different sections of the plate may be used as needed. Each of the end pairs of rods is wrapped with 73 ft. of No. 18 German silver wire, while each of the middle pairs is wrapped with 75 ft. of the same material. For convenience in getting at the coils, the top is made readily removable, being attached to the base by bolts.

The current is taken from the 112-volt, alternating-current lighting circuit, and the power consumed when the four sets of coils are in use is 48 kw.

Records of power consumption have been kept for nearly two years, and while the volume of work done cannot be accurately stated, it is estimated that 2000 chemical determinations are made on a consumption of about 500 kw-hours per month. For the past 10 months the average consumption has been 524 kw-hours.

Considering the efficiency of the apparatus and the cleanliness of the work, the cost of operation compares very favorably with that of the gasoline plate which was formerly used, although, of course, current is actually more expensive. Plenty of heat is available for all analytical purposes, sulphuric acid being readily run to white fumes.

The Chemical Export Trade of the United States.

By William Pollard Digby.

Introductory.—Of all manufacturing industries, that of the chemical trade offers a special case for consideration. Its main activity depends upon the prosperity of other manufacturing industries. In their periods of expansion it has to meet increased demands for products to be used in the treatment of their raw materials. Paper mill, wood-pulp factory, textile bleaching or dyeing are among the foremost of the dependent industries, whose numbers and whose respective spheres are yearly extending.

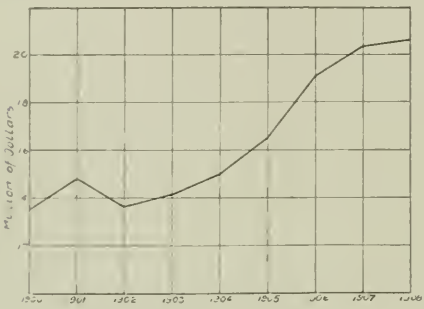


FIG. 1.—TOTAL EXPORT TRADE.

A checking of trade activity due to poor harvests, or to overproduction reacts indirectly upon the maker of alkali or aniline dyes, through the diminished demand for articles of clothing. In this enormous complexity of the interrelation of industrial questions, the phenomenon of the interdependence of manufacturing chemistry with other industries is typical. The trend of motion is, of course, complex, swayed as it is by both economic and technical questions.

On the latter side there is the dependence of certain manufacturing processes upon materials, a demand met either locally or from an external source. In new countries and in the establishment of new industries such demands are almost invariably met by importations. The effects of national fiscal policies tend in favor of the manufacturing of the raw material (actual or contributory, actual in the sense, say, of a material such as cotton yarn requiring bleaching and contributory in the sense of permitting or facilitating a charge in the nature of the actual raw material, such as the "bleach" liquor used in bleaching the yarn) as near as possible to the point of utilization.

Scientific education is now obtainable in many quarters, and the graduate of a good university is fitted for employment in new industries. Then, too, industrial conditions are continually changing, industries migrate just as races migrate, while the progress of scientific invention may displace existing processes just as the ammonia soda process displaced the Le Blanc, only to be threatened in its turn by electrolytic processes.

Under conditions, therefore, economically and technically in a state of flux, this industry both serves as a fair index to the state of trade generally, and also to indicate the amount of industrial progress achieved. As to the state of trade generally, there is the small directly domestic trade for small quantities of such articles as bleaching powder or washing soda for the small house-holder, and the indirectly domestic trade for large quantities of the aforementioned or kindred articles for utilization as contributory raw material.

In the first place, in any new country the directly and indirectly domestic trade are at first incapable of local cultivation despite the imposition of fiscal duties. As the local demand becomes greater the question of whether the contributory raw material can be made locally causes the tentative introduction of its manufacture. Thus, while the output of the finished material may continue to increase, the external purchases of the contributory raw material at first fail to keep pace with the increase and then fall in volume. Despite the fact that science knows no bounds of a racial or national character, all industries, however, cannot bear translation. Sometimes they still flourish in the country which has first adopted them, while their offshoots have failed to take root on transplanting. It is not possible, of course, within the limits of an article of this nature, to plunge deeply into vexed economic questions such as these.

One can merely indicate that these forces are always tending to direct or to deflect the courses of commerce, and point out that they, while subordinate to the swirling tide of supply and demand, in the barter between food producer and food consumer, are not without their marked effect.

The Chemical Export Trade.—Taking the ten years ending in 1908, the total export trade of the United States in chemicals, drugs, dyes and medicines has varied from a minimum of $11,410,000 in 1899 to a maximum of $20,873,000 in 1908.

The annual fluctuations in the sum total of this trade for each fiscal year up to and including June 30, 1908, are illustrated in Fig. 1. The generic description of chemicals, drugs, dyes and medicines is nebulous; a detailed examination of this heading would obviously be of so much greater value to pharmaceutical chemists than to the engineering and manufacturing readers of this paper, hence drugs and medicines are not considered in this review.

The distribution of individual items, both of general chemical interest and of allied electrometallurgical industries (as distinguished, of course, from purely metallurgical industries, such as iron and steel) is worthy of analytical treatment, for which purpose the world's markets have been roughly divided into six

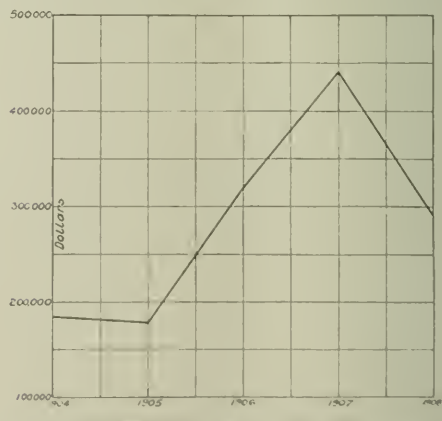


FIG. 2.—EXPORT TRADE IN ALUMINUM.

zones, viz.: Great Britain, British North America, remainder of British Possessions, Germany, remaining European countries, and South America.

Wood Alcohol, for instance, has made large strides, and now in the last year covered by our review was over $800,000. This is one of the few American chemical exports which only enter Canada to a negligible extent. Indeed, all the indications point to Canada as a future alternative source of supply for the European purchasers. Chief among these is Great Britain, which takes close upon one-half of the American exports.

In 1907 the exports to Germany were about 7 per cent greater than those to the United Kingdom, but in 1908 British purchases had been increased by 30 per cent, and German purchases reduced in almost the same ratio. The demands of the remaining European countries are of a fluctuating nature. Table 1 sets out the distribution during each of the last five years.

TABLE 1.—EXPORTS OF WOOD ALCOHOL.

	1904.	1905.	1906.	1907.	1908.
Great Britain.....	$335,964	$362,032	$259,850	$343,840	$451,406
Br. North America..	692	1,607	12	382	133
Germany	186,783	209,010	164,754	367,704	241,660
Rest of Europe....	38,849	13,769	32,600	149,118	118,895
British Colonies....	8,872	5,257	5,446	.....	4,334
South America.....	913	60	728	.....	210
Rest of World.....	13,286	11,660	3,077	1,775	3,115
Total	$585,359	$603,385	$466,467	$862,819	$819,753

The average declared value per gallon exported was as follows:

1904.....	$0.490
1905.....	0.550
1906.....	0.597
1907.....	0.401
1908.....	0.418

The external trade in **pot and pearl ashes** is quite small. In 1907 the best of the five years under review, the total value was only $83,000, of which about one-half went to Canada.

TABLE 2.—EXPORTS OF POT AND PEARL ASHES.

	1904.	1905.	1906.	1907.	1908.
Great Britain.....	$31,082	$9,320	$16,179	$22,631	$26,697
Br. North America..	9,013	9,250	30,930	39,634	14,572
Germany	.....	13	3,032	5,939	.....
Rest of Europe....	12,926	7,317	9,257	12,853	9,290
British Colonies....	20	724	25	752	851
South America.....	454	383	370	159	546
Rest of World.....	4,305	3,119	1,452	1,383	7,632
Total	$56,800	$30,156	$61,245	$83,350	$59,588

The trade in **acetate of lime** is of much larger dimensions. Canadian purchases are negligible. The distribution of these exports is subject to wide fluctuations. British purchases were at their highest value, viz., $140,736 in 1905, and at their lowest, viz., $55,676 in 1908. Sales to Germany and the rest of Europe are indicated below.

TABLE 3.—EXPORTS OF ACETATE OF LIME.

	1904.	1905.	1906.	1907.	1908.
Great Britain.....	$114,290	$140,736	$88,002	$134,869	$55,676
Brit. No. America.	1,354	.....	14	19	2
Germany	294,290	411,956	635,260	719,704	432,850
Rest of Europe....	683,505	676,655	1,221,496	1,560,732	1,027,399
Brit. Colonies	32	5,026	2,761	3,412	.....
South America.....	.....	.....	.....	.....	.....
Rest of World.....	9,868	10,403	3,300	.....	27,513
Total	$1,103,389	$1,245,776	$1,950,833	$2,418,736	$1,543,440

The export of **crude sulphur** is an industry which has made remarkable progress. Not of sufficient magnitude to merit separate enumeration in 1904, the figures given below show a trade increased five fold by the end of 1908.

TABLE 4.—EXPORTS OF SULPHUR.

	1905.	1906.	1907.	1908.
Great Britain	.....	$6,000	$11,920	$10,400
British North America.....	.....	194,446	147,456	349,981
Germany	$51,113	91,504	57,750	131,460
Rest of Europe.....	81,303	65,558	143,560	263,393
British Colonies	.....	.....	.....	.....
South America	.....	.....	.....	115
Rest of World.....	.....	40	6,544	29,986
Total	$132,716	$351,548	$367,230	$785,335

The average declared value per ton exported was as follows:

1905	$1.97
1906	2.10
1907	2.03
1908	2.04

The item **coal tar** is not worth freight to Great Britain or Germany where bituminous coal is plentiful and where gas meets such a large part of the demand for street and house lighting. Exports have mainly gone to Canada and trade having increased from $14,499 in 1904 to $148,812 in 1908. Sales to European countries other than Great Britain and Germany have fallen from $28,395 in 1905 to $11,715 in 1908.

Lime is mainly supplied to Canada, whose demands have risen from $35,533 in 1904 to $82,690 in 1908. The distribution during five years to all other countries are less than $15,200.

Carbon black, gas black and lamp black have gone mainly to Great Britain. The trade is mainly European, as shown below:

TABLE 5.—EXPORTS OF CARBON BLACK, GAS BLACK AND LAMP BLACK.

	1904.	1905.	1906.	1907.	1908.
Great Britain.....$123,722	$204,903	$272,813	$169,337	$200,025
British North America.	29,939	33,168	35,914	35,058	24,944
Germany	105,498	136,528	164,285	128,718	131,946
Rest of Europe.....	73,566	106,992	120,730	112,448	117,672
British Colonies.....	1,236	1,775	1,942	1,684	3,177
South America.....	990	7,347	847	132	153
Rest of World.....	3,062	9,023	27,622	5,638	15,990
Total $329,013	$501,736	$624,153	$453,015	$493,907

The exports of **oxide of zinc** are widely distributed. The British demand seems steady, while that of Germany and Canada are falling. The total distribution to the remaining European countries is fairly steady.

TABLE 6.—EXPORTS OF ZINC OXIDE.

	1904.	1905.	1906.	1907.	1908.
Great Britain.....	$405,250	$306,040	$423,519	$429,663	$451,505
Brit. N. America..	52,756	34,777	97,483	54,865	32,460
Germany	8,251	45,875	137,965	115,125	107,420
Rest of Europe....	205,444	218,069	450,930	393,614	441,329
British Colonies....	266	115	3,887	252	14,187
South America.....	.....	13	308	.....	.....
Rest of World.....	469	637	780	3,471	3,547
Total	$672,436	$605,526	$1,114,872	$996,990	$1,050,448

FIG. 3.—EXPORT TRADE OF EMERY AND CORUNDUM.

Prices per pound have varied to the following extent:

1904	$0.040
1905	0.036
1906	0.037
1907	0.038
1908	0.039

Sulphate of copper is a declining American export. The British and German demands had fallen to zero in 1908, while the Canadian trade in that year reached its lowest value. Of chief importance is the loss of trade to the rest of the Continent of Europe, which in 1908 was almost one-fourth of the value in 1904. The price of this commodity is intimately associated with that of copper; to discuss copper prices and export distribution is beyond the scope of this article.

TABLE 7.—EXPORTS OF COPPER SULPHATE.

	1904.	1905.	1906.	1907.	1908.
Great Britain.....	$1,231	$7,559	$18,529	.....	.....
Brit. N. America..	38,843	39,736	43,523	$34,342	$33,650
Germany	138,624	159,008	71,872	11,060	.....
Rest of Europe....	799,252	416,879	690,217	267,390	201,783
British Colonies....	.....	6,072	12,893	140	.....
South America.....	36,266	49,628	86,877	77,001	115,047
Rest of World.....	116,470	111,140	111,741	35,977	57,577
Total $1,133,686	$790,024	$1,035,052	$425,910	$408,057

While **aluminium** ought logically to be tabooed along

with copper, this industry is so typical of modern electrometallurgy that its brief statistical treatment along with other electric-furnace products is justifiable in these columns. In regard to aluminium, the demands of the purchasing countries have been irregular. Thus, British demands were increased nearly three fold in 1908 as compared with 1907, but Canadian trade fell to less than one-fourth. The general distribution is set out in the following table:

TABLE 8.—EXPORTS OF ALUMINIUM.

	1904.	1905.	1906.	1907.	1908.
Great Britain.....	$18,068	$9,653	$27,120	$47,967	$124,699
Brit. N. America..	137,334	79,567	192,046	296,546	69,101
Germany	1,255	2,989	19,980	5,716	1,739
Rest of Europe....	4,485	59,822	35,337	8,654	38,311
British Colonies....	6,634	10,649	20,556	32,045	17,632
South America.....	11,169	7,802	8,107	13,668	16,393
Rest of World.....	4,252	5,377	15,385	38,991	22,141
Total	$183,197	$175,859	$18,531	$442,987	$290,016

Emery and corundum have proved to be exports of increasing importance. The large upward stride in 1908 is most striking. The distribution of exports is given in the following table:

TABLE 9.—EXPORTS OF EMERY AND CORUNDUM.

Great Britain.....	$5,207	$3,892	$13,373	$4,202	$51,536
British North America.....	1,204	540	1,579	2,998	23,308
Germany	115	15,139	38,354	22,767	81,165
Rest of Europe.....	10,008	16,403	35,147	27,744	71,154
British Colonies.....	8,144	3,980	4,123	752	4,533
South America	59	1,838	413	725	3 324
Rest of World..	538	647	486	2,699	5 771
Total	$25.475	$42,446	$93,475	$61,887	$240,791

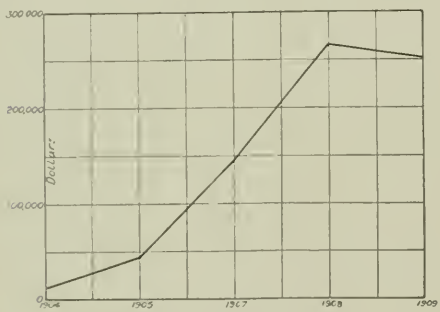


FIG. 4.—EXPORT TRADE IN GRAPHITE.

The export of **graphite** is an example of the rapid expansion of a promising industry, and nothing could be more gratifying to those concerned than the reflection that an export trade less in value than $12,500 should in five years have grown to over $230,000. The distribution during each year is given below:

TABLE 10.—EXPORTS OF GRAPHITE.

	1904.	1905.	1906.	1907.	1908.
Great Britain	$2,448	$8,338	$42,264	$35,384	$45,159
British North America ..	2,264	2,629	2,359	9,324	6,876
Germany	4,569	10,188	43,098	93,480	52,222
Rest of Europe	2,716	16,340	38,542	74,855	93,888
British Colonies.....	121	1,705	5,237	27,532	29,084
South America.....	149	1,804	1,875	3,058	4,711
Rest of World.....	189	3,505	10,337	22,372	10,538
Total	$12,417	$43,609	$113,712	$266,055	$251,283

The export trade in **acids** has fluctuated somewhat from a minimum of $246,000 in 1904 to a maximum of $368,000 in 1907. As regards distribution, by far the largest market is Canada, which accounts for more than half the exports. A comparatively small trade is carried on with Europe. The distribution each year is as follows:

TABLE 11.—EXPORTS OF ACIDS.

	1904.	1905.	1906.	1907.	1908.
Great Britain	$1,470	$14,176	$9,145	$12,106	$18,228
Brit. North America...	70,498	70,922	110,081	162,134	64,135
Germany	..	2,767	2,405	712	60
Rest of Europe . . .	2,111	8,190	1,920	2,696	1,648
British Colonies ..	28,551	31,650	17,620	9,217	9,826
South America...	19,567	12,531	20,777	23,711	21,026
Rest of World ..	121,507	315,272	141,474	160,086	214,021
Total	$245,813	$385,020	$314,122	$367,682	$328,034

Two tables are appended affording an analysis of the above tables which have been divided into two groups, viz: the first including electrochemical products, which included aluminium, carborundum and graphite, and the second group including all others.

ELECTRO-CHEMICAL PRODUCTS.

	1904.	1905.	1906.	1907.	1908.
Great Britain.....	$25,663	$21,883	$82,759	$87,553	$221,394
Brit. North America....	140,802	82,736	195,984	308,918	99,285
Germany	5,939	28,316	101,432	121,963	135,126
Rest of Europe.....	17,209	92,565	109,026	111,253	203,353
British Colonies.....	14,899	16,334	29,916	60.229	52,149
South America.....	11,568	11,444	10,395	16.851	24,428
Rest of World.....	4.949	9.529	26,208	64,062	44,450
Total $221,089	$261,914	$555,718	$770.929	$782,090	

Other Products.

OTHER PRODUCTS.

	1904.	1905.	1906.	1907.	1908.
Great Britain.....$1,016,000	$1,045,000	$1,095,000	$1,125.000	$1,214,000	
Br. North America..	241,000	271,000	633,000	630,000	751,000
Germany	733,000	1,017,000	1,269,000	1,409,000	1,026.000
Rest of Europe....	1,824,000	1,557,000	2,618,000	2,666,000	2,193,000
British Colonies....	39,000	51,000	45,000	16,000	33,000
South America.....	60,000	73,000	111,000	100.000	142,000
Rest of World.....	183,000	371.000	310,000	233,000	352.000
Total $4,198,764	$3,349.561	$6.070,186	$4,171,626	$6,750,698	

On the Equilibrium of the System Consisting of Lime, Carbon, Calcium Carbide, and Carbon Monoxide.

By M. deKay Thompson.

(*Concluded from page 279.*)

EXPERIMENT I.

The charge consisted of lime, carbon and calcium carbide mixed together. A loosely fitting lid with a ¼-in. hole in the center covered the crucible. The mixture was placed in the furnace, the furnace was evacuated and the charge heated to 1000° for an hour to drive off gases that invariably come off on the first heating, and particularly to get rid of any water contained as hydrate of calcium. If this were not done water would come off during the run and react with the carbide present. The furnace was then evacuated to a pressure of 0.05 cm of mercury and carbon monoxide let in to 1.25 cm. This was generated from strong sulphuric acid and potassium ferrocyanide and was washed with two drying towers of soda lime and a phosphorous pentoxide tube. Hydrogen was then admitted to a final pressure of 63.6 cm. This was generated from hydrochloric acid and zinc and was purified by two bottles of permanganate, a hot copper gauze, two towers of soda lime and a phosphorous pentoxide tube. The furnace was filled with hydrogen in three-quarters of an hour. The volume of the furnace, after allowing for the solids present during a run, was 19.9 liters. The run began at 9.45 a. m. and lasted till 4 p. m. The power was held constant at 12.0 kw corresponding to 1485° C. The following table gives the analysis for carbon monoxide, made by drawing off 100 cu. cm into a Hempel burette and absorbing with acid cuprous chloride solution.

Time	Per cent carbon monoxide
9:45 a. m. Sample taken as furnace warmed up.	1 05
1:42 p.m.	Less than 0 1.

It was evident from this result that the quantity of gas corresponding to equilibrium at this temperature could not be analyzed by a Hempel apparatus. The experiment was continued till 4 p. m. to make sure equilibrium had been reached. The method used to determine the small quantity of carbon monoxide present in this and all the following experiments was to draw about half the gas in the furnace through two Liebig bulbs sealed together and filled with cuprous chloride solution. These were tilted at an angle so the gas bubbled through the liquid on leaving each of the five spheres of which a Liebig bulb is composed. The gas then passed a column 7 cm long of

soda lime and another similar one of phosphorous pentoxide. This whole apparatus was made entirely of glass closed by two glass stopcocks. The bulbs, in which the air was displaced by hydrogen, were hung in the balance case by a platinum wire the day before the final weight was taken. The air in the balance case was dried by two beakers of sulphuric acid and the temperature was read from a thermometer in the case. The volume of the bulbs was determined by the bottle method for specific gravity, in which a large desiccator took the place of the bottle. This was necessary in order to be able to reduce the weighings to vacuo. From the total weight in grams of carbon monoxide absorbed the number of moles is formed by dividing by 28, the molecular weight of the gas. This, however, gives only a fraction of the total amount in the furnace. The total amount is calculated as follows: If $n_1 =$ the total number of moles in the furnace before any gas is removed, n_2 the number after a certain amount had been drawn off through the absorption bulbs, $p_1 =$ the pressure in the furnace when the absorption began and p_2 the pressure at the end, then

$$p_1 v = n_1 R T_1$$
$$p_2 v = n_2 R T_2$$

where v equals the volume of the furnace. The temperatures were equal to those of the water surrounding the furnace and were made equal to each other at the start and finish.

Therefore

$$n_1/n_2 = p_1/p_2$$

also

$$n_1 - n_2 = m$$

if $m =$ the number of moles absorbed.

Solving $n_1 = \dfrac{m}{1 - p_2/p_1}$

If $p_3 =$ the total pressure during the run, which is greater than p_1 on account of the higher temperature, the pressure in millimeters of carbon monoxide is computed by the formula

$$p = \frac{m \times 0.0821 \times T \times 760 \times p_3}{19.9 \times p_1}$$

in which T is the absolute temperature of the gas in the furnace at the beginning and at the end of the absorption.

At the end of the absorption the pressure of hydrogen in the absorption bulbs was only about half an atmosphere, consequently enough hydrogen had to be let in to bring the pressure to one atmosphere, after which the bulbs were again hung in the balance case and weighed the following day. The variation due to temperature and pressure change in the weight of hydrogen filling the bulbs was negligible. All weighings given in the following are reduced to vacuo. The data thus obtained after the above run were the following:

Initial weight bulbs.....	175.3392 grams
Final " "	175.3482 "
Gain in weight.....	0.0090 grams

The time taken for absorbing the gas was six hours.

$p_1 = 68.5$ cm of mercury
$p_2 = 38.6$ " "
$p_3 = 89.0$ " "

whence $p_{CO} = \dfrac{0.00074 \times 0.0821 \times 285 \times 760 \times 89}{19.9 \times 68.5}$

$= 0.86$ mm of mercury.

On opening the furnace, white powder was found on the lid.

EXPERIMENT 2.

The same charge as used in experiment 1 was ground up and replaced in the crucible. Part was tested with water and gave off acetylene vigorously. It was heated for an hour to 1000° and evacuated to a pressure of 0.05 cm of mercury. No carbon monoxide was admitted. Hydrogen was let in to 67.2 cm in 1 hour 40 minutes.

Duration of run, 6 hours.
Power, 11.7 kw.
Temperature, 1465° C.

Initial weight bulbs.....	182.5989 grams
Final " "	182.6061 "
Gain	0.0072 grams

Time taken for absorption, 4½ hours.

$p_1 = 67.2$ cm of mercury
$p_2 = 38.1$ " "
$p_3 = 91.8$ " "

$$\therefore p_{CO} = \frac{0.000589 \times 0.0821 \times 288 \times 760 \times 91.8}{19.9 \times 67.2} = 0.73 \text{ mm},$$

On opening the furnace somewhat more white powder found on the walls than in experiment 1.

This experiment was carried out with the idea of approaching the equilibrium from the side which generates carbon monoxide. To decide whether this had been done in the above experiment it was necessary to see whether the bulbs would gain no weight if the furnace were filled with hydrogen and part then drawn through the bulbs. The following blank experiment was therefore carried out: The furnace was evacuated to a pressure of 0.15 cm of mercury, hydrogen was let in to 1 cm and again evacuated to 0.15. This operation was repeated and hydrogen then let in to 67.6 cm. The final filling took 1 hour 40 minutes.

The gas was then drawn through the weighted bulbs for 3 hours 45 minutes.

$p_1 = 67.1$ cm of mercury
$p_2 = 38.1$ " "

Initial weight reduced.....	182.606 grams
Final " "	182.627 "
	0.021 grams

If the whole amount of gas could have been drawn through the gain in weight would have been 0.049 gram. This gain in weight must have been due to oxygen, which might not have been removed or which might have gotten in while filling the furnace. This would have been converted to carbon monoxide by the hot carbon spiral giving too high a pressure for equilibrium. Equilibrium in experiment 2 was therefore approached from the same side as in experiment 1. This remark holds good for all the following experiments.

EXPERIMENT 3.

As the previous experiments agreed fairly well, it was thought desirable to try a lower temperature, to make sure that the gain in weight of the absorption bulbs was really due to carbon monoxide and not to some impurity in the hydrogen.

The charge consisted of fresh carbide, lime and carbon. The furnace was evacuated to 0.15 cm and was heated to 1000° till the occluded gases coming off gave a pressure of 6 cm, which required about 10 minutes. It was then evacuated to 0.15 cm with the furnace still at 1000°. Hydrogen was let in to 2.4 cm and evacuated 0.15. The furnace was then cooled and filled with hydrogen to a pressure of 67.0 in 1 hour 40 minutes.

Power, 8.24 kw.
Temperature, 1250°.
Duration of run, 6 hours.

The solution of cuprous chloride had been used in a previous experiment.

Initial weight of absorption bulbs....	183.6340 grams
Final " " " "	183.6372 "
Gain	0.0032 grams

$p_1 = 67.2$ cm of mercury
$p_2 = 34.8$ " "
$p_3 = 90.4$ " "

$$\therefore p_{CO} = \frac{0.000248 \times 0.0821 \times 286 \times 760 \times 90.4}{19.9 \times 67.2} = 0.30 \text{ mm}.$$

EXPERIMENT 4.

The charge was the same material as in the previous experiment with some lime and carbon added and mixed up with the rest.

The furnace was evacuated to a pressure of 0.2 cm and heated to 900° for two hours. It was then evacuated to 0.15 cm, hydrogen was admitted to 2.4 and again evacuated to 0.15. It was finally filled with hydrogen to 67.3 cm in 1 hour 40 minutes.

Power, 8.8 kw.

Temperature, 1270°.

Duration of run, 7 hours 10 minutes.

The absorption bulbs were refilled.

Initial weight	182.9303 grams
Final "	182.9316 "
Gain	0.0013 grams

$p_1 = 66.8$ cm of mercury
$p_2 = 37.2$ " "
$p_3 = 90.3$ " "

$$\therefore p_{CO} = \frac{0.000103 \times 0.0821 \times 285 \times 760 \times 90.3}{19.9 \times 66.8} = 0.13 \text{ mm.}$$

EXPERIMENT 5.

The object of the following experiment was to see if measurements might not be carried out at a somewhat higher temperature where the pressure would be greater and the determination, therefore, more accurate.

The charge was the same carbide used in experiment 4 to which about one-half as much lime and carbon, previously heated to redness, was added. The furnace was then evacuated to 0.2 cm and heated to 900° for 1¼ hours. It was then evacuated to 0.2 cm and hydrogen let in to 2.3; again evacuated to 0.12 cm and filled with hydrogen to 67.7.

The charge was then heated seven hours with 100 kw corresponding to 1370°. This must have established equilibrium at this temperature. The power then was raised to 12.6 kw corresponding to 1525° for 4½ hours.

The cuprous chloride was the same used in experiment 4.

Initial weight	182.9243 grams
Final "	182.9277 "
Gain	0.0034 grams

Time taken for absorption, three hours.

$p_1 = 66.3$ cm of mercury
$p_2 = 37.7$ " "
$p_3 = 90.0$ " "

$$\therefore p_{CO} = \frac{0.000280 \times 0.0821 \times 285 \times 760 \times 90.0}{19.9 \times 663} = 0.34 \text{ mm.}$$

On opening the furnace a larger amount of powder than any other experiments here given was found on the walls. This, taken in connection with the small pressure found and the experiments referred to in the introduction seem to indicate that at this temperature the carbon monoxide was removed by calcium coming from the decomposition of carbide.

It is true that this equilibrium was really approached from the side of too little carbon monoxide, but as the velocity of the reaction is the same in both directions at equilibrium, this cannot account for the low pressure of carbon monoxide.

EXPERIMENT 6.

The hydrogen used in the following experiments was generated electrolytically on platinum electrodes dipping into sulphuric acid of 1.2 specific gravity. The cathodes were contained in a porous cup closed at the top by a cork stopper covered with paraffine, through which projected glass tubes, into which the electrodes were sealed. There was also a tube through which hydrogen could escape. The porous cup stood in a small battery jar. The hydrogen tube was connected to a mercury manometer so that the pressure in the cathode compartment could be kept from 0.1 cm to 1.0 cm above the atmosphere, thereby preventing air from leaking in. In experiment 6 only one such electrolytic cell was used, but for the last two experiments another cell was connected in series with the first, thus requiring only half the time for filling the furnace. The hydrogen first passed through a soda lime tube, then the hot copper gauze used in the previous experiments, then two soda lime towers and phosphorous pentoxide tube. Hydrogen was passed over the hot copper for at least half an hour before any was let into the furnace, in order to sweep out the air in the tube. The object in using electrolytic hydrogen was to show that the above gains in weight were not due to impurities in the hydrogen generated from zinc and hydrochloric acid.

The carbon monoxide used in the following experiments was generated by allowing formic acid to drop from a separatory funnel into concentrated sulphuric acid.

In order to see if all the carbon monoxide was absorbed by the two Liebig bulbs containing cuprous chloride, in the following experiments a second absorbing apparatus similar to the above was used with one Liebig bulb in place of two. This was filled with a 3 per cent solution of neutral gold chloride. This has been found to oxidize carbon monoxide to dioxide without affecting hydrogen.[13] Gold chloride in an excess of potassium hydrate is even more sensitive to carbon monoxide, but it was found that hydrogen reduced the gold in the alkaline solution to a black powder if left in contact with the solution over night.

The charge consisted of about equal portions of powdered carbide and a mixture of lime and carbon. It had been used in a previous run.

The cuprous chloride in the Liebig bulbs had been used in three previous experiments, but as a little was tested with water and gave a heavy white precipitate it was not thought necessary to change the solution.

The furnace was evacuated to a pressure of 0.28 cm and hydrogen was let in to 1.0 cm; then evacuated to 0.1 and carbon monoxide let in to 0.3 cm. Hydrogen was then admitted to 67.3 cm requiring three hours with a current of about 14 amp.

Duration of run, 6¼ hours.

Power, 11.8 kw.

Temperature, 1475°.

Initial weight cuprous chloride bulbs	173.1312 grams
Final " " " "	173.1383 "
Gain	0.0073 grams
Initial weight gold chloride bulb	116.8119 grams
Final " " " "	116.8158 "
Gain	0.0036 grams
Total gain	0.0110 "

The gain in the gold chloride bulbs was relatively large, probably on account of the cuprous chloride having gotten so much carbon monoxide in solution that it was not so good an absorber as when fresh.

$p_1 = 65.7$ cm of mercury
$p_2 = 34.5$ " "
$p_3 = 92.0$ " "

$$\therefore p_{CO} = \frac{0.000703 \times 0.0821 \times 287 \times 760 \times 0.2}{19.9 \times 65.7} = 0.88 \text{ mm.}$$

EXPERIMENT 7.

The charge consisted of powdered carbide with some coarser pieces on top. It was heated in the furnace at 1050° for an hour and a quarter and evacuated to 0.1 cm. Hydrogen was then let in to a pressure of 1.6 cm, the furnace was evacuated to 0.10 and carbon monoxide let in to 0.25 cm. Finally, hydrogen was let in to 68.3 cm, requiring 1½ hours with 14 amp.

Duration of run, six hours.

Power, 11.8 kw.

Temperature, 1475°.

The cuprous chloride bulbs were refilled, but not the gold chloride.

Initial weight cuprous chloride bulbs	173.7646 grams
Final " " " "	173.7695 "
Gain	0.0049 grams

[13] Phillips, *Journ. Am. Chem. Soc.* 16, 273 (1894).

Initial weight gold chloride bulb.... 116.8121 grams
Final " " " " .... 116.8126 "

Gain ..... 0.0005 grams
Total gain ..... 0.0054 "

Time required for absorption, three hours.

$$\therefore p_{CO} = \frac{0.00065 \times 0.0821 \times 291 \times 760 \times 95}{19.9 \times 69.3} = 0.81 \text{ mm.}$$

EXPERIMENT 8.

The charge was the same material as used in experiment 7.

The furnace was evacuated and heated for 1 hour 50 minutes at 1050°. It was then evacuated to a pressure of 0.12 cm and hydrogen let in to 2.0, again evacuated to 0.15 and carbon monoxide let in to 0.28 cm. Hydrogen was then admitted to 77.6 cm, requiring 1¼ hours.

Duration of run, 6 hours 10 minutes.

Power, 11.4 kw.

Temperature, 1445° C.

The cuprous chloride bulbs were refilled.

Initial weight cuprous chloride bulbs.. 167.4274 grams
Final " " " " .. 167.4312 "

0.0038 grams

The tube previously used for gold chloride was filled with cuprous chloride.

Initial weight 2d cuprous chloride tube..119.3358 grams
Final " " " " ..119.3363 "

Gain ..... 0.0005 grams
Total gain ..... 0.0043 "

$p_1 = 67.1$ cm of mercury
$p_2 = 37.7$ " " "
$p_3 = 92.7$ " " "

The time taken for absorption was 3¼ hours.

$$\therefore p_{CO} = \frac{0.000350 \times 0.0821 \times 287 \times 760 \times 92.7}{19.9 \times 67.1} = 0.44 \text{ mm.}$$

3. DISCUSSION OF RESULTS.

For convenience the results obtained above are collected in the following table:

TABLE 4.

No. of Exp.	Duration in hours.	Kilowatts.	Temp. Centigrade.	Gain in weight of 1st bulb.	Gain in weight of 2d bulb.	Time taken for absorption, hours.	Initial pressure CO in mm Hg.	Final pressure CO in mm Hg.
1	6½	12.0	1485	0.0090		6	12.5	0.86
2	6	11.7	1465	0.0072		4½	0.0	0.73
3	6	8.2	1250	0.0032		3½	0.0	0.30
4	7	8.8	1270	0.0013		2⅔	0.0	0.13
5	4½	12.6	1525	0.0034		3	0,0	0.34
6	6¼	11.8	1475	0.0073	0.0036	4	2.0	0.88
7	6	11.8	1475	0.0049	0.0005	3	1.5	0.81
8	6	11.4	1445	0.0038	0.0005	3¼	1.3	0.44

In all of these experiments, even at 1250°, there was some white powder on the walls of the furnace. Whether a slight decomposition of carbide into its elements takes place at this temperature could not be decided by this means, as the white powder may have been due to two causes, both the decomposition of carbide and the volatilization of some impurity in the lime or carbon. The best evidence that the carbide does not break up at 1475° and does break up at 1525° is that equilibrium could be measured at the former but not at the latter temperature. Attention was called to the possibility of lime itself being somewhat volatile at 1500°, since a piece of Merck's lime heated at the melting point of platinum for an hour also produced a layer of white powder on the walls of the furnace.

As experiments 1 and 2 were carried out at temperatures equally above and below the temperature in experiments 6 and 7, the average of these four may be taken, with the result

p_{CO} at 1475° C. $= 0.82 \pm 0.02$ mm.

Though these results were obtained from the same side of the equilibrium, different amounts of carbon monoxide were present at the beginning in each case, which makes the evidence that equilibrium had been reached conclusive.

From this result, the pressure obtained in experiment 8 at a temperature 30° lower may be checked by the integrated van't Hoff equation:

$$4.57 \log_{10} (p_2/p_1) = Q \left(\frac{1}{T_1} - \frac{1}{T_2} \right)$$

where p_2 and p_1 and the pressures of carbon monoxide corresponding to the absolute temperatures T_1 and T_2 and Q is the heat absorbed by the reaction, when it proceeds from left to right. Q has been calculated[14] to be 121,000 calories at room temperature, with a negative temperature coefficient of 3.3 calories per degree. Therefore

$$Q = 121{,}000 - 3.3\,t$$

where t equals Centigrade degrees above room temperature, which for high temperatures may be considered as degrees

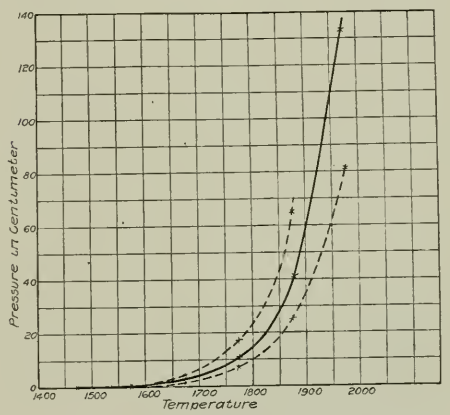


FIG. 3.—PRESSURE OF CARBON MONOXIDE COMPUTED FROM THE VALUE OBTAINED AT 1475°.

above zero. For 1460° C., Q, therefore, equals 116,000 calories. Substituting in the above equation the absolute temperatures corresponding to 1475° and 1445°, the value of p_2/p_1 comes out 1.79. The ration between the pressures found by experiment is 1.86, which is very satisfactory agreement.

If the pressure at 1270° is calculated from that at 1475°, using the value of Q corresponding to the mean temperature

TABLE 4.

Pressures of Carbon Monoxide Computed from the Value Determined at 1475°.

Temperature Degrees Centigrade.	Equilibrium Pressure of Carbon Monoxide in Centimeters.		
	I. Lower Limit.	II. Mean.	III. Upper Limit.
1475	0.05	0.08	0.13
1575	0.31	0.50	0.79
1675	1.54	2.53	4.00
1775	6.6	10.7	17.00
1875	25.0	40.5	64.00
1975	81 0	133.0	210.00

1370°, the result is 0.0093 mm, that is, it is below a measurable quantity. The fact that in one case 0.13 and in another 0.3 mm

[14] Trans. Am. Electrochem. Soc., Vol. 15, 197 (1909).

were found is due to the insufficient time allowed to absorb this very small amount of carbon monoxide.

From the value of the equilibrium pressure obtained at 1475°, it is possible by the above formula to calculate the pressure at higher temperatures and see approximately what is the shape of the pressure temperature curve. The value of Q corresponding to the mean of each set of temperatures is used; 1475° is always taken as the lower temperature. The results of this computation are given in table 4 and plot 3.

It is evident that the error in this curve is due practically entirely to the error in the temperature measurements, for while the value of p_1 is accurate to 2.5 per cent, the temperature is uncertain by 25°, and the value 0.82 mm might correspond to 1500° or 1450° as the two extremes. This would mean the true value at 1475° might be 1.3 mm or 0.5 mm as the two extremes. If now the curve be computed first with the value 1.3 in place of 0.82, and again with 0.5, the values under I and III in table 4 are obtained. These values are plotted in plot 3 in broken curves. From these curves it is seen the temperature corresponding to one-third of an atmosphere lies between 1800° and 1875°, with which Rudolphi's value agrees the best of the three referred to in the introduction.

The free energy increase of the reaction taken from left to right at 1475° C. is

$$F = RT \log \frac{760}{0.82}$$

$$= +23{,}700 \text{ calories.}$$

As the temperature rises ΔF decreases till at 1920°, where the equilibrium pressure equals an atmosphere, $\Delta F = 0$. Above 1920° ΔF becomes negative.

SUMMARY OF RESULTS.

1. The equilibrium pressure of carbon monoxide in the reaction

$$CaO + 3C \rightleftarrows CaC_2 + CO$$

was measured at 1475° and 1445°. The results were in good thermo-dynamic agreement.

2. A little below 1445° C. the pressure becomes too small to measure; a little above 1475° decomposition of calcium carbide into its elements prevents measurement of equilibrium.

3. With the aid of the heat of the reaction the vapor pressure curve at higher temperatures was computed which cannot be realized experimentally on account of the decomposition of calcium carbide.

4. The free energy increase of the reaction

$$CaO + 3C = CaC_2 + CO$$

at 1475° is +23,700 calories.

Electrochemical Laboratory,
Massachusetts Institute of Technology,
Boston, Mass.

Tests of Bakelite.

As will be remembered from Dr L. H. Backeland's papers (our Vol. VII, pages 111 and 276), bakelite is a new synthethic product resulting from the chemical condensations of phenols and formaldehyde. Since the first announcement of its remarkable properties and wide usefulness, bakelite has undergone continually many tests, not only in the inventor's own laboratory, but in practical use by others for a great many different purposes. Naturally much of interest has been detected. For the information contained in the following notes, which will be found to contain much that is new, we are obliged to Dr. Backeland.

Bakelite in its final stage (C) is hard, infusible, and insoluble; it then resists all solvents and almost all chemicals; it stands temperatures as high as 300° C. or over; at the temperature of melting glass it is destroyed and chars, without entering into fusion; it is charred by boiling concentrated sulphuric acid and by nitric acid; it is also attacked by bromine, which seems very remarkable, as it stands very well chlorine and hydrochloric acid and cold sulphuric acid; boiling dilute sulphuric acid has no effect on it. Certain varieties can be made to stand hot caustic solutions.

Mixed with mineral substances like asbestos, its resistance to heat is considerably increased. In unmixed condition, it may appear as a perfectly transparent amber-colored substance, which is much stronger and more resistant than amber and can be turned and polished to a high finish.

Except for jewelry purposes, bakelite is most used compounded with other materials, for instance, asbestos, wood-fiber, mica, etc. On account of the great impregnating power of the initial raw products even quantities as small as 10 per cent of this binder are sufficient to produce very strong, compact, molded masses.

The specific electric resistance, dielectric strength, mechanical strength, maximum working temperature, etc., of bakelite vary considerably according to the nature of the compound. For instance, compounds with asbestos will stand considerably higher temperatures than those made with wood fiber. The latter are more adapted to turning, sawing and all purposes where great toughness is required. Bakelite compounds containing asbestos withstand very well high-pressure steam, even if the latter be at a temperature of 200° C. Pure bakelite electrizes by friction, which denotes high dielectric properties.

Asbestos-bakelite insulators do not show such high dielectric constants, but are very well adapted for voltages not exceeding, say, 40,000 volts. They have the considerable advantage over shellac or rubber insulators that they do not soften by heat and are considerably stronger.

Bakelite compounded with wood pulp shows higher dielectric strength on account of the fact that wood pulp does not contain disturbing materials, like iron, which is always present in asbestos. On the other hand, such wood-fiber bakelite insulators cannot well be used where the insulators are submitted for long periods to temperatures above 150° C., but they resist water and steam, oil and solvents.

As an example as to the general behavior of bakelite compounds, the following indications may be given:

A stick of pure bakelite was boiled continuously for 260 hours in 10 per cent sulphuric acid solid without alteration. A similar stick was submitted for two months, day and night, to a 10 per cent caustic soda solution at 70-80° C. without appreciable effect.

A steel pipe was lined with a thin coating of bakelite and kept for 21 days and 21 nights in hot damp chlorine gas at 60° C., without alteration.

Bakelite was kept in a steam digester for 12 hours at 200° C. (226 lb. steam pressure) without appreciable effect.

The results of a series of electrical and mechanical tests of bakelite and wood pulp and of bakelite and asbestos are given in the following table:

PROPERTIES OF BAKELITE INSULATING MATERIAL.

	Bakelite and wood pulp.	Bakelite and asbestos.
1. Specific resistance in megohms per cubic centimeter	35×10^6	0.01×10^9
2. Dielectric strength of thickness 0.375 inch in volts at moment of puncture	115,000	42,500
3. Dielectric constant in kilovolts per centimeter	121	45.
4. Mechanical strength in pounds per square inch tension	650	1,200
5. Mechanical strength in pounds per square inch compression	1,000	18 000
6. Maximum working temperature in degrees centigrade	90	450

The electrical value of bakelite depends entirely on the composition and the way in which it is molded and cured. Values as high as 66,000 volts puncture on a thickness of 0.02 in. have been obtained by special treatment of this material.

Bakelite compositions can be made very strong mechanically, and will resist the impact of suddenly applied loads.

The material weathers perfectly without any signs of hydrolysis, and surfaces so exposed show a peculiar tendency toward preventing surface creeping or leakage.

Pittsburgh Meeting of the American Electrochemical Society

The seventeenth general meeting (and the ninth annual meeting) of the American Electrochemical Society was held in Pittsburgh from May 4 to 7 and turned out an immense success. It was the best attended meeting so far held by this society; the attendance reached almost the 500 mark, there being some 460 names in the last printed registry list.

The meeting was exceedingly well organized and the highest credit is due to the members of the Pittsburgh local committees for the carefulness and completeness with which all arrangements had been made. Mr. Charles F. Scott, of the Westinghouse Electric & Manufacturing Company, was the chairman of the executive committee, and a wonderfully successful and unfailingly courteous chairman he was. Messrs. F. Crabtree, H. K. Hitchcock, P. Kemery, H. H. Laughlin, R. S. Orr, Hugh Rodman, and S. S. Wales were the other members of the executive committee and the chairman of the various subcommittees. But it was really whole industrial Pittsburgh that heartily welcomed and royally entertained the society.

L. H. BAEKELAND, RETIRING PRESIDENT AMERICAN ELECTROCHEMICAL SOCIETY.

An account of the social functions and of the various most enjoyable and instructive excursions is given at the end of this report.

Thursday Morning Session.

The first professional session was called to order by the president, Dr. LEO H. BAEKELAND, on Thursday, May 5, at 10 a. m., in the lecture hall of the Fort Pitt Hotel.

Mr. B. F. MORSE, president of the Engineers' Society of Western Pennsylvania, welcomed the society to Pittsburgh in a very felicitous little speech, full of interesting personal reminiscences concerning the development of industrial Pittsburgh since the Pennsylvania Railroad came to the city in 1852.

He said that what will be remembered as the greatest achievement of the Roosevelt administration is the inauguration of a systematic organization of the policy of conservation of our natural resources. In days to come the big stick will be forgotten, but the policy of conservation will remain. In this field much is to be expected from the work of electrochemists, also in connection directly with Pittsburgh. There is a tendency of the center of industrial activity to move gradually away from Pittsburgh to the West. Much can be done by electrochemists to keep the center of gravity fixed at Pittsburgh.

To show how it can be done, Mr. Morse gave an account of the work of the Pittsburgh "flood commission." Forty or 50 public-spirited engineers are giving their services free to the work of surveying the headwaters of the rivers around Pittsburgh for the purpose of locating the rainsheds that cause the rivers to overflow their banks at certain seasons. In one flood there was a direct loss of $4,000,000 in two days. The object of the flood commission is to prevent such damage, to harness the waters and get electricity out of them, and employ it usefully in the Pittsburgh mills. In this development electrochemists can and must help.

Mr. CHARLES F. SCOTT, as chairman of the local executive committee, followed with a speech of welcome to industrial Pittsburgh. The doors of all the magnificent industrial establishments would be wide open to the society. In the great history of Pittsburgh output and quantity has been the keyword of the past, quality and economy will be the keyword of the future.

The annual report of the board of directors was then presented by the secretary of the society, Dr. Joseph W. Richards. The society is in a flourishing condition financially and the membership is rapidly increasing. Two hundred and ninety new members were elected in 1909, the total number of members at the end of 1909 was 1066; it is now above 1200. The formation of a Chicago local section has been authorized by the board of directors.

The results of the election of officers were then announced. Dr. William H. Walker, of the Institute of Technology of Boston, is the new president of the society. Mr. Pedro G. Salom and Dr. Joseph W. Richards were re-elected treasurer and secretary, respectively. Dr. E. F. Roeber, Mr. S. S. Sadtler, and Dr. Louis Kahlenberg were elected vice-presidents for two years; Prof. Charles F. Burgess, Mr. C. P. Townsend, and Dr. W. R. Whitney managers for three years.

Laws of Furnace Electrodes.

Two papers were presented by Mr. CARL HERING, of Philadelphia, Pa., on the laws of furnace electrodes. The first paper was entitled "Determinations of the Constants of Materials for Furnace Electrodes" made of carbon, graphite, iron, and copper. The method of determination is that described in Mr. Hering's former Am. Electrochem. Soc'y paper (our Vol. VII, page 514). The results of these determinations were already given in Mr. Hering's recent Am. Inst. Elec. Eng. paper, abstracted in our May issue, page 238. His present paper describes the details of these tests.

W. H. WALKER, PRESIDENT-ELECT, AMERICAN ELECTROCHEMICAL SOCIETY.

Briefly described, the method consists in imbedding a rod of the material to be tested in a heat-insulating material, allowing the two ends to project sufficiently for terminals. A steady current is then passed through it which will heat it; it becomes hottest midway between the ends. The temperatures at the middle and at the ends, the voltage at the two ends, and the current are then measured. From this data all the necessary physical constants are then calculated.

The mean resistance is the quotient of the volts and the amperes; from this and the size of the rod, the mean resistivity is

calculated. The product of the volts and amperes gives the heat flow in watts, which flows out at the two ends, and from this and the dimensions the heat conductivity is calculated.

The rod is equivalent to two electrodes butting together at their hot ends, the interior of the furnace itself being omitted; hence the entire flow of heat is that due to the resistance of the electrodes, which is the condition under which a well-proportioned electrode should operate. The rod is assumed to be perfectly heat-insulated; this is nearly accomplished by surrounding it with a cage of similar rods through each of which the same current passes.

A stable state of temperature is, of course, essential, and the calculated constants are the correct mean values under electrode conditions; these mean values represent those which an equivalent electrode would have whose conductivities are the same from end to end.

The electrodes were circular rods 18 in. long and about ⅝ in. diameter. Seven equal electrodes were used in series in each test; the rod under test being in the center, with the six others arranged as a cage around it. The refractory material used for imbedding the rods was very finely pulverized magnesite.

The couplings at the outside ends were water-cooled and were soldered to the ends of the rods; this took up 1 in. at each end, having 16 in. clear length of rods between terminals, thus forming two 8-in. electrodes butting together at their hot ends. The couplings connected all the rods in series. The two for the two ends of the middle rod had slots cut into them permitting a thermometer or thermo-couple and pointed wires for voltage measurements to be inserted to touch the electrode under test at the exact point of emergence from the furnace; hence the rod under test was exactly 16 in. long, and the heat which might be generated in the contacts with the terminals was eliminated.

A pyrometer was inserted with its end just over the middle point of the test electrode for measuring the high temperatures. The current was supplied by a motor driven, separately excited, direct-current dynamo of low voltage and high amperage.

Each test consisted in passing a current through the series of rods until the stable state of temperature had been reached, as shown by the pyrometer. The temperature in the center, those at the two ends, the drop of voltage between the two ends, and the current, were then measured. This was repeated for each of a number of different temperatures. And these tests were made for carbon, graphite, iron, and copper.

The paper contains considerable details of the arrangement of the tests and records various observations made in carrying them out. It is very important to wait with each measurement until the steady state of temperature is reached. This may require several hours, depending upon the conditions. To determine whether the steady state is reached, it is necessary to take repeated readings of the pyrometer to see whether it has ceased rising.

The high temperatures were measured with a platinum, platinum-rhodium thermo-couple made especially for this test with long wires, the electromotive forces being measured with a potentiometer, thereby avoiding the error due to a current. The use of a potentiometer with this thermo-couple also enabled widely different temperatures and small changes in them to be measured with very great accuracy.

The great importance of this accuracy was not to measure the temperatures so closely, but to determine the stable state—that is, to determine accurately whether all the interior parts had reached their maximum temperature.

Notes are given on special protection of the test tubes and on their deterioration at higher temperatures. If the gases in the granular refractory material are the cause of the deterioration, Mr. Hering suggests to carry out the test in a vacuum furnace.

* * *

The second paper of Mr. Carl Hering is entitled "Empirical Laws of Furnace Electrodes." Its object is to analyze his experimental results with a view of finding whether any of the physical properties involved follow some approximate general laws. This refers especially to the "watts per amperes" E and the "section per ampere per inch length" S'. (See definitions on p. 238 of our May issue.)

The relation between the quantity E (watts per ampere) and the temperature drop T is given by a hyperbolic formula

$$E^2 = aT + bT^2,$$

where the constants a and b have the following values:

	a	b
Carbon	0.00373	0.000000366
Graphite	0.00256	—0.000000710
Iron	0.0000369	0.000000116
Copper	0.0000176	0.0000000399

Since b is negative for graphite, the curve is an ellipse for graphite.

The relation between S' (square inches per ampere per inch length or square centimeters per ampere per centimeter length) and the temperature drop T is given by the formula

$$S' = m + n/T$$

where the constants m and n have the following values:

	Inches.		Centimeters.	
	m	n	m	n
Carbon	0.000200	0.483	0.000508	1.23
Graphite	0.000173	0.0665	0.000440	0.169
Iron	0.0000967	0.00262	0.000246	0.00666
Copper	0.0000129	0.000888	0.0000329	0.00226

The author also discusses the reciprocal of S' (the current for 1 in. length per square inch section) and the conductivities and gives some other deductions.

In the discussion which followed Mr. Hering's papers, some theoretical notes communicated by Dr. A. E. Kennelly, of Harvard University, were first presented. The principal result of his analysis is that Mr. Hering's empirical relations between the various properties of iron and copper may be accounted for, first, by the Wiedemann-Franz law, and secondly, by the approximate constancy of the thermal conductivity and thermal resistivity of these metals throughout the range of temperature drop covered by the observations.

Mr. H. W. Gillett, of Cornell University, thought it was not the normal case that the tip of the electrode was at the temperature of the furnace itself so as to prevent heat outflow through the electrode. This condition could certainly not be obtained in an arc furnace. He gave some experimentally determined temperature gradients in graphite and carbon electrodes, at each 15 minutes of the run of a carborundum furnace, and emphasized the great similarity in the temperature curves for graphite and carbon. "Hence it seems that Mr. Hering takes too pessimistic a view of the carbon electrode in commercial furnaces. The similarity of the curves for carbon and graphite is not due to graphitization of the tip," since the tip of the carbon electrode was not graphitized to any appreciable extent. "Finally, since in the carborundum furnace at least it is not at all difficult to measure temperatures between 1500° C. and 2500 C. to from 10° to 20°, not only concordantly, but accurately, and since this is the range in which most commercial furnaces operate, we feel that any electrode formulas should be really tested on carbon and graphite electrodes to temperatures beyond the 900° C. given as the highest point in Mr. Hering's table."

Mr. Hering replied that the furnace in Mr. Gillett's experiments was not operated under the conditions specified in Mr. Hering's analysis that the hot electrode end should be at exactly the same temperature as the furnace interior, so that no heat flows through this end in either direction. This, he emphasized, is the proper or correct way of operating a furnace or designing an electrode, and it is possible to make the arrangements in such a way as to fulfill this condition, but it was not fulfilled in Mr. Gillett's experiment, nor did he reach the stable temperature condition to which the theory applies. Mr.

Gillett replied that the closeness of the final temperature curves showed that the stable condition had almost been reached.

Mr. Lidbury asked whether Mr. Hering's values above 900° were all extrapolated. This was confirmed by Mr. Hering.

Dr. J. W. Richards expressed his appreciation of the value of the experimental data of Mr. Hering, but called attention to a great discrepancy between some heat conductivity values given by Mr. Hering and experimental determinations of others. Mr. Hering emphasized that his figures of mean conductivities were not based on any assumptions, but were derived directly from the experiments. Mr. Queneau stated that in experiments on the relative conductivity of refractory brick he had found values for carbon and graphite which confirmed Mr. Hering's figures.

Heavy Alternating-Current Conductors.

A paper by Mr. R. C. RANDALL, of the Westinghouse Electric Laboratories, was entitled "a study in heavy alternating-current conductors for electric furnaces." The author first emphasized the importance of considering the nature of the service for which the conductors are intended.

When currents of 10,000 amp and up to 40,000 amp or 50,000 amp are contemplated, conductors very heavy in section are immediately suggested. Viewed from considerations of resistance alone the problem is quite simple, but to alternating current, unlike direct current, the simple resistance of a conductor is not a measure of its suitability. Alternating current does not take full advantage of the conductivity of an ordinary conductor; it does not distribute itself uniformly throughout the section, but favors the area near the surface and shuns the central areas of the conductor section. A portion of the center of the section of a cylindrical conductor might be removed without appreciably impairing the effective conductivity for the alternating current.

Skin Effect.—The surface distribution of the alternating current is termed "skin effect," and is more pronounced for higher than for the lower frequencies; that is, as the frequency more nearly approaches direct-current conditions, namely, zero frequency, so does the importance of skin effect diminish. With small conductors of ½ in. or even 1 in. in diameter, the skin effect at 25 cycles is negligible, but with a 2-in. round conductor the skin effect has already reached such a value that the effective resistance has increased 25 per cent compared with direct-current conditions.

For 60 cycles, the ½-in. conductor begins to experience a trifling increase in the effective resistance; the 1-in. conductor experiences an 11 per cent increase and the 2-in. conductor more than 80 per cent increase—that is, its effective resistance is nearly doubled. Furthermore, this is true independent of the current density which is employed. For example, a 2-in. copper rod (at 20° C.) 78 ft. long will carry 5000 amp direct current with an ohmic drop of 1 volt and an energy loss of 5000 watts. With 25-cycle alternating current and the same current, the ohmic drop would be 1.26 volts and energy loss 6300 watts; with 60-cycle current, 1.82 volts and 9100 watts loss.

These considerations would lead one to infer that for 60-cycle service it would be necessary to practically double the conductor section, and for 25 cycles to increase the section about 25 per cent in order to comply with direct conditions. This conclusion, however, is misleading, for if a 2-in. conductor had been increased 82 per cent in section, the effective resistance of the conductor to 60-cycle current would still be about 25 per cent greater than the original 2-in. conductor, to the passage of direct current. Fortunately, the conductance of heavy conductors can be improved even for fairly high frequency alternating currents without the excessive increase in section, as outlined above.

Shape of Section and Form of Circuit.—In addition to the sectional area of the individual conductor, the shape must be considered when alternating current is to be transmitted. For a given conductor area, as far as effective conductance is concerned, the poorest solid section form is round, and the best is a very thin, flat sheet. If the sheet be rolled into a thin-walled tube, it represents less resistance than even the solid rod. Or the solid round conductor might be replaced by a laminated unit made up of a large number of separate small conductors, each of which has the same resistance and averages the same mechanical position throughout the length of the transmission. Each small unit would then be equally attractive to the alternating current, that is, each would carry a like current and the distribution throughout the combined conductor would be uniform and the "skin effect" would be eliminated. In practice, such an arrangement cannot be carried out conveniently, and approximations must suffice.

Besides the consideration of the individual conductor as to size and shape, it is necessary to go further and consider the form of the heavy current circuit—that is, the relative position of the coming and going conductors. Here the currents in the going and return conductors take the inner areas of closely adjacent conductors (as shown in end section by circles), thus not enjoying the full conductivity afforded by the conductor sections. The same remedy as for bad distribution applies, namely, laminating or breaking up into many individual conductors.

From the considerations so far, it is seen that in addition to considering the simple ohmic resistance which the direct current encounters in a circuit, it is necessary in alternating-current work to contemplate the shape and size of the individual conductor and also the form and dimensions of the heavy-current circuit.

In order to discuss some of the limitations encountered in practice, a furnace installation operating at 100 volts, 25 cycles, is assumed, the conductor arrangement and service being such that a power-factor of 90 per cent is obtained at rated full-load current. As the impressed voltage is 100 and the useful or working voltage is 90, the reactance voltage is 43. These 43 volts—the measure of the self-induction in the circuit—cause little concern under the assumed conditions.

But assume, further, that it becomes necessary to operate at 60 cycles. The reactance voltage is proportional to the frequency and current, so that on 60 cycles and the same current their value would become $60/25 \times 43 = 103$ volts, or more than the total impressed voltage. This means that this furnace, even with the electrodes short-circuited, would not quite take full current from a 60-cycle circuit, and similarly on the original 25-cycle circuit something less than $100 \div 43 = 2\frac{1}{3}$ times full-load current would be delivered under short-circuit conditions. If it were desired to increase the electrical input to the furnace, as by increasing the current through some adjustment of the electrodes, a maximum of approximately 130 per cent could be delivered. For with the fixed 100-volt supply an increase of current will increase the value of the reactance voltage and correspondingly decrease the working voltage.

The maximum input (on 25 cycles) occurs when the base and altitude are equal, namely, when each is 71 volts, a condition obtained by a current input of 165 per cent. The product of current (165 per cent) and voltage (71 per cent) is (118 per cent) proportional to the energy input and is approximately 30 per cent greater than (100 per cent × 90 per cent = 90 per cent) the rated load input to the furnace.

Capacity Limitation by Frequency and Voltage.—For a 60-cycle circuit the normal full-load current is much too large to give the maximum input. In general, the maximum energy that can be fed into a circuit at different frequencies will be inversely proportional to the frequencies, and will exist at a power-factor of 71 per cent. The assumed installation, though quite satisfactory for 25 cycles, would not be good for 60 cycles. A change in the arrangement of the conductor system so as to reduce the self-induction (reactance volts) would be necessary.

If, instead of 100 volts, but 60 volts had been used, a maximum input of 36 per cent would be obtained with 25-cycle cur-

rent as compared with the maximum input for the 100-volt service. At 60 cycles the maximum input would be but 42 per cent of 36 per cent, or approximately 15 per cent. Similarly an increase in the service voltage would increase the possible energy consumption. For example, 120 volts, 25-cycle supply would make the maximum input about 145 per cent and correspondingly 60 per cent for 60-cycle current. In general, energy input to a circuit for a given frequency is proportional to the square of the impressed voltage.

If there be an appreciable loss in the transmission circuit, the power-factor of 71 per cent for the whole system will not correspond to the maximum energy dissipation in the furnace, although it does give the condition for maximum energy in the whole circuit, including leads, terminals, electrodes, etc. The power-factor corresponding to maximum energy in the furnace is dependent on the resistance R and the reactance pL of the feeding system, and has a value between 0.71 and unity. It has a value of 0.71 when $R =$ zero (the condition assumed in the installation discussed), and has the value unity when $L =$ zero, a condition which can only be approached in ordinary practice.

As the function of an electric furnace is one of heating materials, and its capacity is in a measure determined by the amount of energy consumed in it, it may be said that the capacity of the furnace is determined by the maximum energy which can be dissipated within it, and this is determined by the operating power-factor, as pointed out.

The influence of power-factor on the capacity of the generating station is not discussed, but there is an evident relation of power-factor to this part of a system.

The weight of conductors for very heavy currents is quite considerable, and good mechanical provisions must be made for their support. Besides having mechanical strength, the supports must be insulators, although the potentials involved will usually be only about 100 volts. The necessity for laminating and interlacing the conductors in heavy-current work complicates the support problem. The contacts and connections between different parts of the circuit and the terminals of the apparatus, where the heavy current is to do its work, are also serious problems of this work.

A popular method of arranging conductors for moderate capacity installations employing 2000 amp to 5000 amp is the use of a number of stranded cables disposed alternately of opposite signs (from opposite sides of the circuit), as in this way an effective interlacing is accomplished which terminates at the service point where the cables of like sign are joined to the proper terminals.

Besides the questions of self-induction and effective resistance as influenced by shape and size of conductor and form of circuit, there exists the additional question of the use of different materials, notably copper and aluminium for conductors. These materials offer different resistances to the passage of direct current, and their factors for "skin effect" are not the same. For the same section the factor is somewhat greater for copper but practicaly the same for sections of equal resistance.

The efficiency of a heavy current conducting system is, in importance, on a par with the importance of the efficiency of the transformer and line from which the power will usually be obtained. Therefore, the question of loss in the heavy conductor system may be viewed in the same way as the loss in the other parts of the transmission and transformer system. And according to the importance of these considerations will the warrant for expenditure to reduce losses be determined.

In the preceding it is pointed out that conditions for maximum energy consumption in a circuit correspond to a power-factor of aproximately 70 per cent, and although this is true, the conclusion should not be that a power-factor of 70 per cent is most desirable, because in general this is not the case; the higher the power-factor the more satisfactory the operation. But it is interesting to note that when a furnace or similar energy consuming device must be forced to its maximum energy consumption from the constant-voltage circuit, this condition does not correspond to high power-factor operation. The limitations under these special conditions are given in the following:

(1) The maximum power in a system containing reactance occurs when the total ohmic voltage (CR) equals 71 per cent of the impressed voltage.

(2) The maximum power occurs in a furnace (resistance load) when the power-factor of the system has a value given by

$$0.71\sqrt{1+\frac{R}{\sqrt{p^2L^2+R^2}}}$$

where R and L are, respectively, the resistance and self-induction in the feeding circuit.

(3) Input to a circuit for a given frequency is proportional to the square of the impressed voltage.

(4) The maximum energy that can be fed into a constant-voltage circuit at different frequencies will be inversely proportional to the frequencies, and will exist at a power-factor of 71 per cent.

In general as a summary the following points may be noted:

The size and shape of conductors and operating frequency determine the magnitude of "skin effect."

The form and dimensions of the circuit and the frequency determine the self-induction.

"Skin effect" and self-induction are reduced by laminating the individual conductors, by the selection of conductors of the proper size and shape, and by interlacing the conductors.

The reduction of the self-induction in the circuit is the most important and most difficult.

In the discussion which followed, Mr. C. A. Hansen gave figures on the influence of the leads on the total resistance and reactance of a 10,000-amp furnace, and also gave a table which showed the maximum power input of the furnace at constant voltage taking place at a power-factor of 0.71.

Dr. M. G. Lloyd referred to the use of two concentric tubes as carriers of an alternating current, the inner tube being water cooled. There is no external magnetic field with such an arrangement.

Electric Steel Refining.

Mr. A. L. Queneau presented a preliminary paper on "A New Electric Steel Furnace." He briefly sketched the object of his furnace design, which was already noticed and illustrated in our March issue, page 149.

Mr. Queneau proposes to utilize the pinch effect to produce regular pulsations in an electric furnace charge in order to thoroughly mix it. One or more electrodes are in the bottom, and for a steel furnace these electrodes are rods of steel; their upper ends are in direct contact with the molten steel charge. These upper ends of the steel electrodes are molten and by providing a small restriction of the cross-section of the rod at that point the pinch effect is to be produced. The molten column of steel is thereby interrupted, but is immediately closed again by the weight of the charge above. In this way a series of regular pulsations is to be produced. The erection of this furnace has been somewhat delayed, and for this reason no results of experiments could be given in the paper. They are promised for a later meeting.

Replying to a question of Mr. Hansen as to the material of the refractory surrounding the molten steel where the continual make-and-break is to be established, Mr Queneau said that he proposed to use magnesite laid in tar. Mr. Hansen questioned the possibility of keeping the refractory lining intact at that place under the severe conditions of operation.

Gases in Steel.

Dr. Paul Héroult, of La Praz, France, presented a brief but suggestive paper on "gases in steel," maintaining that they are generally harmless and are present to about the same extent in all kinds of steel.

It is well known that when a thoroughly sound steel ingot

is placed in a vacuum chamber, a certain amount of hydrogen and nitrogen gases is exhaled and thrown off. The popular doctrine is that the percentage of the gases present is in proportion to the opportunities of absorbing them as a result of the process employed in producing the steel. But Dr. Héroult emphasizes that in the course of thorough and repeated investigations the quantities of gases so liberated have proven to be about the same, whether it be Bessemer steel or basic or acid open-hearth steel or crucible steel or electric steel. Dr. Héroult maintains that these gases are "absolutely innocent" and are not the cause of the production or presence of blow-holes.

On the other hand, blow-holes in steel ingots invariably contain hydrogen and nitrogen, often with only traces of carbon monoxide. Dr. Héroult, therefore, discusses the way in which blow-holes form. With the exception of accidental blow-holes due to a poor condition of the molds, blow-holes are the result of a disengagement of carbon monoxide. But this carbon monoxide does not pre-exist in the steel; it is only produced when the steel cools down and part of it has become solid. Steel that will produce blow-holes contains in the molten state carbon and at the same time iron protoxide. As long as the steel is molten and hot, these two bodies can keep apart, and for each temperature and each composition of steel there is a state of equilibrium at which no chemical reaction takes place. The heat is then what the Germans call "gar"—that is to say, dead melted.

"If the temperature is increased the reducing action of carbon will be intensified, and carbon monoxide will be evolved. If, on the contrary, the temperature is lowered, nothing happens till the steel gets partially solidified, with the effect that carbon and iron protoxide are crowded in a small space, in what we might call the mother liquor, and monoxide of carbon is evolved. The blow-holes produced are filled with carbon monoxide at a high temperature. As this gas cools down, however, it creates a vacuum and we then repeat the conditions of the ingot in the vacuum chamber."

That is, the vacuum now causes hydrogen and nitrogen to be given off from the steel, and this is the reason why blow-holes contain hydrogen and nitrogen.

In conclusion Dr. Héroult emphasized again that "steel does not contain any gases to amount to anything, and whatever small quantities it does contain are not injurious or detrimental to its quality. As a matter of fact, Bessemer steel, either acid or basic, which has every possible opportunity of absorbing gases, does not contain any more than any other steel."

In the brief discussion which followed, Dr. R. Amberg referred to the recent work of some French investigators on gases in steel. Boudouard (*Revue de Metallurgie*) and Belloc (*Bull. Soc. d'Encour.*) both worked in a vacuum and found that the steel exhales gases only after a continued treatment for a certain time, which in all cases was longer than industrial processes would permit to apply. He also referred to some recent patents of Girod who proposes to produce steel free from gases by cooling a superoxidized bath or a bath still containing a certain amount of combustible elements down to low temperature by the addition of cold scrap, to 700° C. to 900° C. in one case or to 1100° C. in another, while it was stated that 400° C. to 500° C. would be still better if attainable. Dr. Amberg had tried this reaction both in superheated and in non-superheated steel in an electric furnace and could not observe any evolution of gases, below, say, 1200° C. In the first moment, of course, while throwing cold metal into a hot furnace at 1800° C. a development of gases takes place which is due, however, to quite another cause. Dr. Héroult replied that a method of removing gases at low temperatures would not be applicable in practice in his opinion. Dr. J. W. Richards pointed out that as a simple calculation shows, even a very small percentage of oxygen in steel (in form of iron oxide) when combining with carbon will give rise to a very large volume of gas; hence the attempts to eliminate any oxygen whatsoever from the steel.

Electrolytic Preparation of Magnesium.

The preparation of magnesium by electrolysis of various fused electrolytes was the subject of a paper by Prof. SAMUEL A. TUCKER and Mr. FAREL L. JOUARD, of Columbia University.

According to text books, electrolysis of fused magnesium chloride gives the metal and chlorine. This method was first tried by the authors. The bath should consist of a mixture of magnesium and potassium chloride in molecular proportions. This combination is found in the mineral carnallite, but the mineral itself is unsuitable for the process. The effect

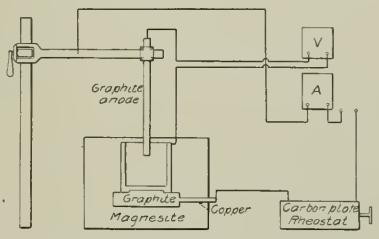


FIG. 1.—CARBON CRUCIBLE ELECTROLYSIS OF MAGNESIUM CHLORIDE.

of the potassium chloride is to prevent the decomposition of the magnesium chloride and also to lower the melting point of the bath. In fusing the two chlorides together it is found desirable to add ammonium chloride, as this also hinders the decomposition of the magnesium chloride. There are two varieties of magnesium chloride on the market—the crystals $MgCl_2$, $6H_2O$ and the so-called "fused." The latter appears to contain almost as much water as the former, and it is, therefore, better to start with the crystallized compound.

A mixture is therefore made of $MgCl_2.6H_2O$, 203 parts; KCl, 74 parts; NH_4Cl sublimed, 50 parts.

As the magnesium chloride is deliquescent the resulting fusion will probably not contain the two chlorides in molecular proportions, but experience has shown that this makes little difference in the subsequent electrolysis.

At first graphite crucibles were used to fuse down the mixture, heating with a gas flame and also in some cases in a granular carbon electric furnace. If heated by a gas flame, care should be taken to keep the products of combustion from being absorbed by the fusion. This can be accomplished by the use of an asbestos shield with lateral openings for the escape of the gases. One advantage of graphite crucibles for this operation is from the reduction of any sulphates present with elimination of the sulphur dioxide, but graphite seems to be porous to some extent to the fused chlorides and the yield of fused material is not very good.

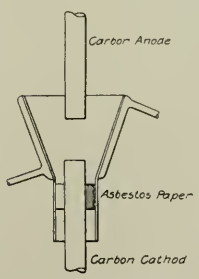


FIG. 2.—MUTHMANN CRUCIBLE.

In the subsequent work platinum was used, the same precautions being taken to keep out the products of combustion from the blast lamp. The mixture is added in small portions, allowing time for the steam to escape and also keeping the temperature of the mass low enough to prevent decomposition. When the mass is uniformly melted the ammonium chloride will have been eliminated, and if the temperature is raised decomposition of the magnesium chloride will result and the melt will become cloudy. It should, therefore, be poured into a mold before this stage is reached and, after breaking up, bottled at once. During the fusion stirring with a carbon rod tends to break up the sulphates. The yield of fused material is about 50 per cent of the fusion mixture.

It is important that overheating of the mass of chlorides dur-

ing fusion be avoided, as experiment showed a loss of 37.7 per cent of magnesium by decomposition and 1.12 per cent by volatilization when the mixed chlorides were kept at low red heat for 30 minutes. These results were obtained by analysis of the starting mixture and the subsequent melt.

After some experiments on a very small scale (in which the addition of a small quantity of calcium fluoride proved beneficial by tending to unite the small globules of metal) the apparatus (Fig. 1) was derived.

The graphite crucible, 2¾ in. diameter, 3½ in. high, is entirely surrounded by heat-retaining layers of magnesite and the passage of the current is sufficient to maintain the bath in fusion. The current used was 150 amp at 30 volts. The whole apparatus was placed in a hood to take off the chlorine which otherwise is unbearable. The apparatus can be started with a cold charge by drawing an arc with a small carbon rod, but it is easier to add the charge in the molten condition.

It was found best to make the vertical rod 1¼ in. diameter, the anode and the crucible cathode, though the reverse will work fairly well. If the temperature is not raised too high, the metal collects at the bottom and side of the crucible in a spongy form which is easy to remove from time to time. The sponge is afterward united with an excess of calcium fluoride and some magnesium potassium chloride, so that it unites to a large globule.

The authors describe in detail six different runs which they made with this apparatus. From the results it would appear that the presence of calcium fluoride is beneficial, while the presence of magnesium oxide is to be avoided.

Further experiments were made in a crucible designed by Muthmann[1]. The apparatus consists of a small conical copper crucible having a tube or opening at the bottom through which one of the carbon electrodes enters. The sides of the crucible are provided with a copper water jacket which congeals the charge so that the copper is protected. The other electrode is a vertical electrode of carbon which can be adjusted vertically (Fig. 2.)

The mixed chlorides were added to the crucible cold, and were easily melted by a small arc between the electrodes, the upper electrode was made anode and the metal collected at the lower one in one globule and as it increased in size it would rise to the surface, interfering somewhat with the position of the anode. In a run of 28 minutes, with a mean current of 33.6 amp, 3.34 grams of metal were obtained, the theoretical yield being 7.12 grams.

The apparatus worked very well, using very small quantities of charge, and on cooling a slight tap at the lower electrode easily detaches the charge. As Muthmann has succeeded in using the crucible to advantage in other electrolytic work it is described how the crucible was made.

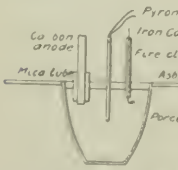


FIG. 3.—PORCELAIN CRUCIBLE.

A piece of copper tube, 1¼ in. diameter, 1/16 in. wall, and 4 in. long, is spun to a cone at one end, beginning with the lathe and finishing on the anvil with hammer; the diameter of the cone at the end was 2⅛ in. Sheet copper is used for the jacket about 1/32 in thick, which can be rolled to cover the outside with edges hammered over so as to allow for passage of the water. This and the small copper tubes for entrance and exit of the water are brazed with silver solder using the oxy-acetylene flame. The silver solder was made by uniting together metallic silver with half its weight of yellow brass. It is used on copper with borax glass as flux.

The lower electrode can be of carbon or graphite and is separated from the crucible by winding asbestos string or paper at this point.

The influence of temperature, current density, and the separation of the magnesium from the chlorine evolved at the anode was investigated by a series of experiments carried on in a small porcelain crucible (Fig. 3) containing the fused mixed chlorides. The temperature was determined by a thermo-couple and the anode, which was of carbon, was surrounded by a mica shield to prevent the chlorine from diffusing through the bath. The cathode consisted of an iron wire loop which could be easily varied in the surface exposed. The sketch shows these details.

Changes of temperature were made from 440° C. to 600° C. and 700° C. At the lower temperature the metal can be removed in the spongy form, at the higher temperature the metal melts and is less easy to recover. The current density at the cathode was raised from 50 amp to 500 amp per sq. dcm. These changes made but little difference in the current yield of metal and show that these factors have less influence on the yield than might have been expected. The metal obtained from some of the previous runs was analyzed and found to contain 98.86 per cent Mg.

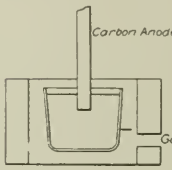


FIG. 4.—IRON CRUCIBLE.

The second part of the paper deals with experiments in which **magnesium fluoride** was used as electrolyte (patents of Seward and Kügelgen). A charge was first made up of 72 parts magnesium fluoride, 36 calcium fluoride, to which MgO was added to the extent of 5 per cent of the magnesium fluoride in two portions. The quantity of magnesium produced was insignificant and it could not be recovered. In another experiment 50 parts lithium fluoride, 25 magnesium fluoride, and 25 calcium fluoride were used, but not a trace of metal was obtained. There seems to be a solvent action by such a mixture of fluorides for magnesium, as was actually shown by placing a weighted quantity of metallic magnesium in the melted fluorides; it was found to have lost 40 per cent of weight.

Another series of experiments was made with a double-step process, producing first an **alloy of magnesium**, to be treated then in the second step as anode. Thus, in a bath consisting of chlorides of magnesium with other additions (see above), and using a molten aluminium cathode, it is easy to produce an alloy of magnesium and aluminium. In the second step this alloy would then be made anode in order to recover metallic magnesium at the cathode.

But using a chloride bath, with the addition of some barium chloride, the authors found it difficult, if not impossible, to recover a fair proportion of the magnesium. The authors do not see any special advantage in such a scheme as the current efficiency is no better with aluminium present and the second step of the process is very far from satisfactory.

Alloys of copper are easily made in the same way as the aluminium alloys and show a rather better current efficiency; in one experiment an efficiency of 56 per cent was obtained.

The authors sum up their results as follows: Magnesium can be obtained by electrolysis of its molten chloride in presence of potassium chloride, preferably in a carbon or graphite crucible, but the bath is troublesome and expensive to prepare. The current efficiency may run as high as 60 per cent. The presence of MgO is to be avoided. A small proportion of calcium fluoride tends to consolidate the metal.

In this process changes of current density and of temperature have comparatively little effect unless these changes are very great.

Electrolysis of magnesium fluoride and other fluorides as additions for dissolving oxide of magnesium gave practically no metal and there seems to be direct solution of magnesium in such molten compounds.

Alloys of magnesium with aluminium or copper are easily made by electrolysis with the use of a chloride bath, but they have no special advantage from the standpoint of current effi-

[1] W. Muthmann, H. Hofer and L. Weiss, *Annalen*, 1902, 320 (2), 231.

ciency as compared to the process for making pure magnesium.

The electrolytic decomposition of the alloy containing aluminium for the recovery of the magnesium was unsuccessful.

Science and Industry.

The address of the retiring president, Dr. LEO HENDRIK BAEKELAND, was presented in the evening session. It dealt in an inspiring way with "science and industry."

The present age surpasses all previous epochs of history by the intense activity of the human race. All great feats recorded in the history of our race sink to nothingness if compared to what human activity is now accomplishing every day since ignorant, arrogant, emotional, spasmodic efforts are slowly but surely giving place to methodical and persistent work based on exact scientific knowledge.

Whether or not the human race has been made happier, certainly our opportunities for happiness have considerably increased.

Nowhere have the changes of this century been so accentuated as in our industrial enterprises. The two most powerful men of our generation are the scientist and the engineer. Society at large is far from realizing this fact, simply for the reason that the scientist and the engineer manifest their power not as despots, not as cruel tyrants. Their unassuming life is devoted not to slaughter, destruction, or coercion, but to the service of mankind. They do not build useless pyramids cemented with the sweat and blood of overabundant slaves, monuments to vain-glorious despots.

But the modern engineer, applying the principles of science, raises buildings far superior in size and conception than any architecture bygone ages can boast of; edifices incomparably more comfortable, more hygienic, more appropriate than anything built before. He raises those gigantic structures in as many days as it took years to build a temple.

In fact, after a few years, he is ready to pull the same building down, to erect better and bigger ones, to suit advanced conditions, and nobody cares about the name of the architect or the engineer, nor does the builder care himself.

And why should anybody care? The dynamics of the age are producing changes at such a rapid rate that nowadays any building, of whatever size it be, is begun with the feeling that before long it will have to come down to give place to new conditions.

The modern engineer and the scientist realize that much more enduring monuments than stone, brick or bronze will mark the work of this period; they know that the diffusion and application of exact knowledge is shaping the destiny of future generations and will afford more lasting evidence of their efforts than temples or statues; they believe that their work will not count merely for material betterment, but that improved material opportunities created by them will bring forth better people, higher ideals, a better society.

"To put it tersely, I dare say that the last hundred years, under the influence of the modern engineer and the scientist, have done more for the betterment of the race than all the art, all the civilizing efforts, all the so-called classical literature, of past ages, for which some respectable people want us to have such an exaggerated reverence." . . .

"It has been asserted so often by respectable people that science and industry cater only to our material welfare, and have little in common with culture, refinement or moral development. Therefore, I feel compelled to put special emphasis on this side of the question and to insist on the enormity of this error; on the contrary, the development of our industries, or our material prosperity, as well as the study and application of science are the surest and most immediate forerunners of higher civic ideals, of an improved society, of a better race.

"A clean, properly fed and well-housed individual, who can enjoy the comforts and advantages of modern surroundings, and leads an active, intelligent, productive, self-supporting and self-respecting life, is certainly more of a man and a credit to his race than were some ancient saints who lived from alms and who spent their life in prayer and inaction, or who, for further edification of their followers, vowed never to change their clothes, nor wash nor shave nor comb themselves; he is more of a blessing to his fellow men than the useless drone who lives on the work of others and gives nothing in return but arrogant presumption based on fortune, rank or title inherited from his father.

"If this be then the age of rational industrialism, of applied science, how then is it that in some industries quality is going down, while prices are soaring upwards?

"Here again it is a noteworthy fact that just such commodities as are produced by so-called scientific industries are sold cheaper and are of better quality than ever before, and this cheapening of price or bettering in quality is almost proportionate to the amount of scientific knowledge involved in their production. Let us take, for instance, the chemical and the electrical industries, both based nearly exclusively on well-developed scientific data.

"In both these groups of industries the chemist or the physicist have had full sway and the engineer has embodied their work in a practical form. Free and rational competition based on intellectual superiority has been their paramount factor of development. Competition based on artificial privileges, like labor unions, tariff legislation, have played only a secondary role. While flour, meat, clothing, and houses were considerably less expensive a hundred years ago than they are now, we find that acids, alkalies, salts, solvents, dyes, and in general almost all chemicals are incomparably cheaper and of better quality than they were in the good olden times.

"In some cases, the changes are remarkable. For instance, a ton of sulphuric acid sells now at the same price as two pounds of the same article were sold about 150 years ago.

"A similar cheapening can be found in many other chemicals, although their demand has immensely increased. Without going to extreme cases, we can state that there has been a steady improvement in most chemical manufacturing processes and that the public at large has had the benefit thereof. The same can be said of the electrical industry.

"Compare this with industries which are still under the sway of the rule-of-thumb, which means the rule of the ignorant, or where competition is based on political protection; you will find that just such rule-of-thumb commodities where science plays no role are those for which the public has to pay the highest price in return for the poorest article. Married men may follow this assertion from butcher's bills to ladies' hats; from house rents to servant girls.

"For the poor chemist, it is almost an irony of fate that his science, by developing the 'cyanide process,' made gold cheaper and thereby reduced considerably the purchasing equivalent of his meager salary. In order to get square he will have to put himself now to the task of helping the engineer in the cheaper production of foodstuffs, or clothing, or take a hand in such economic reforms which may bring about a reduction of rent or may lessen other anomalies.

"Notwithstanding all our progress, it is evident that we live in a transitory stage; next to enterprises and industries embodying the highest intellectual conceptions our century can offer, we find even in the most advanced countries examples of conditions of affairs which seem truly an anachronism.

"To many of you it has happened to visit factories or mills where ignorance and greed seemed the two dominant factors, where the class of men and women employed, not to speak of child labor, seemed to have undergone the full curse of their sordid surroundings. Such places are to be found often where the mental condition of the directors does not enable them to go beyond the conception of size and where the whole tendency has been toward more, more, more, instead of toward better, better, better.

"How different is this from some of our better engineering

and chemical enterprises where everything bears the imprint of a steady effort toward progress and where employer and employed alike seem to undergo the uplifting force of intellectual aims. Such a happy condition of affairs is most likely to be encountered where the head is himself the scientific pioneer who has built up the enterprise.

"Matters are not always so satisfactory where large organizations have gotten into the hands of a board of directors, who know little else of the technical side of the business than that it pays dividends, and for whom the main interesting factor is the value of the shares they own.

"Whenever undertakings are ruled by such a class of men, we must not be astonished if their corporation counsel is more in evidence than their chemists or their engineers. What do they care if certain improvements in their processes might net them 5 per cent more or mean better goods, if on the other hand they know that by a clever trick of law they can extract from the consuming public many times more; no wonder then if they have less time and less mental fitness for a principle of science or engineering involved in a new process than for a conference with eminent law counsel. If they cannot alter nature's atomic weights, they may find a way of improving their invoice weights for the custom house to the detriment of Uncle Sam. I might use for our industries the forceful quotation of Shakespeare in Hamlet about the State of Denmark, as long as corporation lawyers of reputation are paid incomparably better and their services are sought for so much more eagerly than the very best chemists or the ablest engineers.

"This brings to my mind the case of a company which held a charter to supply a certain city with illuminating gas, and which after enjoying a fortune-making monopoly for many years, found one day that special legislation had reduced the selling price of their product. Certain again of being able to upset this law, the company entered in long litigation, but finally, after repeated efforts, had to realize that even its best lawyers could not change matters. From that moment on they began to inquire actively about better manufacturing processes. A friend of mine, who was requested to give his suggestion as to how they could improve their methods, replied as follows: 'Up till now your company has been making *law*—now make *gas* and everything will come out all right.'

"Then again we find that resourceful as the modern engineer or chemist is his power is often simply a tool in the hands of ignorant but cunning men. In fact, our modern laws and society insure better reward for cunningness or slyness than for true intellectuality.

"The very abundance of our natural resources may be partly to blame for this condition of affairs; in other countries, like Germany with comparatively small natural means, competition shapes itself more toward technical perfection. If we want to learn how to reduce what I would call our 'national waste,' then surely our German friends can give us valuable lessons. It is significant, too, that in large German engineering or chemical enterprises, the board of directors is made up mostly of scientifically trained men, engineers, chemists, and physicists."

On the other hand, industries "have sometimes been handicapped by a too one-sided scientific organization; I know of some instances, especially in Germany, where very respectable enterprises have not utilized their available opportunities to the proper extent, because their scientific managers lacked good business sense. I have seen some industrial enterprises much held back by too much red tape and a choking amount of paper wisdom. The most learned man without common sense or practical abilities can accomplish little except disappointments. Here is where the keen business man, with a practical turn of mind, with directness of purpose and good judgment, will every time show his advantages.

"An overspecialized man, whether he be a biologist, a physicist, a chemist, or an engineer may lack the broadness of conception and action which characterizes true great men of many-sided development.

"Then again, quite frequently the real field of usefulness of scientifically trained men is much misunderstood. For instance, it is a common mistake, made even by some engineers and physicians, as well as by business men, to imagine that the main work of the chemist is confined to performing chemical analyses. This conception is about as absurd as to think that the usefulness of an electrical engineer consists in making electrical tests or that the essential work of the merchant is bookkeeping.

"Many a good chemist has been thus prevented from showing his best abilities by the sheer ignorance of those who employed him.

"In the development of some of our industries, nothing has played such an important role as scientific research work."

Dr. Baekeland contrasted the old electric machine of Benjamin Franklin with the gigantic electric installations at Niagara Falls and continued:

"Let me ask a fair question to those who underestimate the value of research: Has that stupendous gap between Franklin's toy and the power companies of Niagara Falls been bridged by anything but by scientific research of the highest order?

"Some of the better educated people in this country begin to understand more and more the necessity of scientific research. Not so long ago, research work was only carried out in the laboratories of universities or in a few highly developed chemical or electrical companies; nowadays we find many intelligently conducted enterprises devoting a considerable annual outlay for systematic research work, where the resources of the chemist, the physicist, and the biologist are used to good purpose.

"Unfortunately, the scope and method of scientific research is difficult to understand for the uninitiated. Some people have only the haziest conceptions on this subject. Some manufacturers, totally unaware of the requirements involved in this work, in a half skeptical way, grudgingly conclude to organize a research department, sometimes as a last resort, to help them through some difficulties; others do it 'to be in style,' and simply to imitate their more successful competitors. Frequently they engage a young man with little experience, who, outside of what he studied in the technical school or at the university, has everything to learn and who besides that is usually entrusted at the very start with the most difficult problems. His salary is none too high and his action is very much restricted; sometimes he is forbidden to study the current practical methods, or so-called 'manufacturing secrets,' and is thus prevented from getting acquainted with the very problems he is supposed to solve. I have seen other cases where the time of the research chemist was filled with odd jobs of every kind. After a while, when practical results are not forthcoming fast enough, the bookkeeper confronts him with the list of expenses which have been incurred by his work; naturally some comments are ready at hand how the same money spent on a good salesman would have shown immediate results, and so forth. Things go along that way for a while until the research department is abolished with the recurring remark: 'Research does not pay; we've tried it.'

"In other cases, where some results are obtained, the matter is taken out of the hands of the chemist before he has had time to fairly try and develop it on a large scale. The subject is now entrusted to the superintendent or the foreman, who seldom is a friend of the scientifically trained man, and nearly always resents anything which might diminish the prestige of established practical experience. Like in all new processes, defects are soon shown and in the natural order of things repeated failures and renewed trials on a practical scale are required before there is any possibility of regular utilization. The research chemist is allowed very little intervention at this stage of the work, and after some time, remarks are heard how imperfect the whole thing was 'before so-and-so, the practical man, had his say.' Finally, initial expenses are charged against the research department, and profits credited to the practical man.

"A research department is a very difficult thing to organize

and to run. It is not enough to provide a building and the necessary appliances; it is not enough to provide typewriters, card-indexing systems, and office force, and all the red tape connected with it; it is not sufficient to engage one or more well-behaved university or college graduates with the necessary helpers and to let them work under an orderly businesslike manager. You might as well try to produce masterly paintings by installing an office management and a well-organized paint and brush department, and a library containing all that has been written on the art of painting next to a splendidly equipped studio and then leave out the real artist who will do the painting. Nay, the most important, the almost exclusive factor in a successful research laboratory is the research chemist himself. If he is not a man who has a soul alive with his subject, if he is not enthusiastically imbued with his opportunities, if he is not qualified for his task, not only by scientific training, but specially by a natural gift of discrimination between what is most important in a problem and what is secondary to it, you might as well fill a hall with the marble statues of Greek poets and imagine that they will write poetry for you.

"Then if you find the man who has all the true qualifications, you may still paralyze his action by too much red tape, too much interference in his work. A good research chemist will do more and better work with pots and pans from the '10-cent store' in a shed or in a barn where he is his own master than in a splendidly equipped laboratory where he gets irritated and interfered with by others who do not understand him.

"I sometimes doubted whether it was really worth while for a young man to take up research work single handed when so many people with abundant facilities were at work. What show, for instance, does an organic chemist have in studying a problem for which in Germany some large chemical companies employ hundreds of research chemists? To this I can answer that some of the most striking examples of successful research were the result of privately conducted work with modest means; in fact, I know of several instances where a research chemist who had created himself a reputation by work carried out privately under adverse circumstances, showed disappointing results as soon as he became part of a vast organization.

"Even if you have the best qualified research chemists, do not expect immediate results. Do not forget that problems, appearingly most simple, require considerable time before they are thoroughly studied. Even in successful cases, it may easily require many years before a subject is so thoroughly elucidated that it can be taken up in practice.

"Research is what gives a young man of strong individuality a chance to compete with those big industrial consolidations, the trusts, who, like elephants, look more imposing by their size than by their agility or perfection, and who as that pachyderm have many vulnerable spots, and are just as much handicapped by their lack of flexibility and by their ponderosity. Some steel manufacturers may be unable to think about anything but tonnage, and yet the work of some chemists has already indicated that the quality of steel of the future, or of its alloys, may be improved to such a degree that probably the average steel of to-day will look to our children as brittle and imperfect as pig iron appears to us. Neither should we lose sight of the fact that even to the most exclusive mechanical enterprises, there is a chemical side, although the importance of the latter may not be apparent to the man who is not a chemist.

"Let me give also a warning to such manufacturers who try to secure only by uncompromising secrecy the money-talking end of their industries. As far as my experience goes, exaggerated secrecy is very often an indication of lack of knowledge of industrial feebleness and incompetency; a miser is most of the time a man of small means.

"If the chemists had been holding their results from each other, we would still be in the dark ages of the alchemist. No secrecy, however jealously carried out, can outweigh enlightened research work, protected by wise patent legislation. If our patent laws do not protect enough, then our prime duty becomes to change them until they answer their purpose as defined by the constitution of the United States. The care with which patent laws are administered is a direct measure of the industrial importance of a country. Piracy cannot flourish, neither on the seas nor in intellectual property, if ethics of justice and equity can be made to prevail.

"Every recorded success of the scientist or the engineer is an additional evidence that ignorant greed and brutal rapacity cannot forever have full sway in this world, and that the rule of the sly and the shy leads to the abortion of progress.

"Furthermore, the results of their work, which bars our 'chance,' 'luck,' or 'happenings,' is their most eloquent language to convince their fellow men that if lawmakers may still think that laws are made or unmade by them in Albany or Washington or Harrisburg, there is at least one law which cannot be amended; at least one law which even the cleverest lawyers cannot make to be interpreted in two different ways; a law which rules all men, large or small, poor or rich, to whatever nation they may belong; a law which rules the dead and the unborn as well as the living; a law which requires no Supreme Court to test its validity; a law that cannot be trifled with, which nobody and nothing can escape; the great unchangeable law of Nature which rules the universe, mocks at men-made statutes and ordinances, and upsets and destroys everything which comes in conflict with her; the rigidly enforced law which tries to teach us our mistakes by suffering, by misery, by industrial or political crisis, by unhappiness, by war, so as to awaken us from our ignorant sleep, to show us our misguided aims, and to command us to prepare a sounder, a happier condition for our children and future generations, while building up, during the trend of centuries, a slowly rising foundation for a higher humanity, a more god-like race."

Cheap Power in Pittsburgh.

Prof. F. CRABTREE, of the Carnegie Technical Schools of Pittsburgh, followed with an interesting address on the "Possibilities of Cheap Power in the Pittsburgh District." Since the paper was not printed in advance and contained a formidable array of figures, it is impossible to give here more than a brief outline of the contents. The paper will be printed in full in the *Transactions* of the society.

Professor Crabtree first referred to the wasteful coking process in bee-hive ovens in the Pittsburgh district. From this source alone he estimated 450,000 hp might be developed. The investment would be somewhat high, but the cost of operation would be low. He estimated the cost of the kw-hour for a 100 per cent load factor as about 0.25 cent, and for a 25 per cent load factor as 1 cent.

Professor Crabtree then discussed the utilization of blast-furnace gases which could also be developed into an important source of power in the Pittsburgh district, at an estimated cost of ½ cent per kw-hour.

The production of power from natural gas was next discussed. The cost was estimated as slightly less than half a cent for a 100 per cent load factor. With producer gas the cost would be slightly higher. The possibilities of low-pressure and mixed pressure steam turbines were also discussed.

In summing up Professor Crabtree thought that the cost of the kw-hour in the Pittsburgh district should be 0.5 to 1.2 cent, and water-power stations cannot do very much better in general.

In the discussion which followed, Dr. Baekeland pointed out that in a great many instances where an electrochemical industry requires a large amount of power, it also requires coal. For instance, in sodium chloride electrolysis, for the production of caustic soda and chlorine, coal is needed for evaporation of the caustic solution. In all such cases the advantages of Pittsburgh deserved careful attention.

Induction Furnace Progress.

A review of the progress made in recent years in the design, construction and applications of the induction furnace, both the simple induction furnace type and the combination

furnace type, was given in a paper by Mr. T. Rowlands.

Simple Induction Furnace.—These are of the Kjellin-Colby design and intended for melting purposes only, as the bath forms an annular channel in a crucible of suitable refractory material, and the crucible is so designed to allow the center leg of the transformer, together with the primary coil, to pass through it (Fig. 1).

The simple induction furnace is running in competition with the crucible process in the manufacture of steel. The comparative cost is given by Mr. Rowlands as follows:

The best practice in the old coke method of melting, as followed in Sheffield crucible process practice, shows that fuel and crucibles alone cost about $14 per ton, and, in this country, using plumbago and crucibles working with gas producers, the cost is from $10 to $11 per ton.

If a medium-sized induction furnace, capable of melting about four tons per 24 hours, pouring only 1000 lb. per cast and working on a high power consumption, is taken for comparison and a high cost is allowed for furnace lining, as follows:

750 kw-hours per gross ton at 0.5 cent per kw-hour..	$3.75
Lining per ton.....	1.00
	$4.75

it will be seen that the saving effected on melting for fuel and crucibles only amounts to from $5 to $10 per ton.

The assumed figure of a consumption of 750 kw-hours per ton is stated to be high. Moreover, the labor question has not been considered. Excluding the labor for the casting pit, which is about the same in electric and crucible processes, the electric furnace hands are fewer, and it is quite the regular thing for one man and a boy to do all the work on a furnace tapping one ton each heat. The actual saving will be between $7 and $13 per ton of ingots.

The regularity of the composition of the steel is another factor in favor of electric melting. Any irregularity in the composition of the steel can be adjusted directly in the furnace before tapping. This, of course, cannot be so finely regulated in crucible practice.

Fig. 2 illustrates the power consumption and power-factor on a charge of over 800 lb.

The simple induction furnace has also been used for the melting of other metals, like nickel and brass.

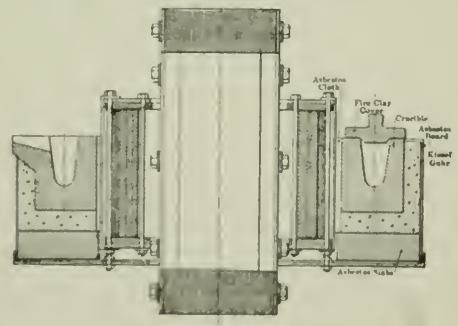


FIG. 1.—SIMPLE INDUCTION FURNACE.

Experiments on *nickel* melting were made at Niagara Falls with a 60-kw furnace. The molten nickel charge was 168 lb. Details of these experiments are given in Mr. Rowlands' paper. Of special interest is the rapidity of melting down. For the first charge, although the furnace was not very hot, and including delays, it took only 1½ hours, and in the second they commenced charging 50 lb. at 4:12, obtaining a complete melt at 4:22 and teemed at 4:35.

This work was experimental and consequently they were not working under the best conditions. The furnace was giving a power factor of about 90 per cent, therefore, with an average kilowatt of 48.4; the power consumption for this quantity was at the rate of 325 kw-hours per ton for melting only, which is very good for so small a furnace.

Brass melting in the simple induction furnace has been carried out in England. To overcome troubles with the pinch effect, a special crucible was employed, together with specially constructed coil on the furnace, whereby the electrical conditions could be so changed that at any desired moment most of the current could be carried either by the metal in the bath or by the crucible itself. By this arrangement the crucible is first heated, and, when sufficiently hot, the brass scrap is charged. This soon melts, completing a circuit. The switch is then thrown over, and the metal itself carries most of the current; the crucible, when exposed, gradually becomes cooler, the cooling being visible.

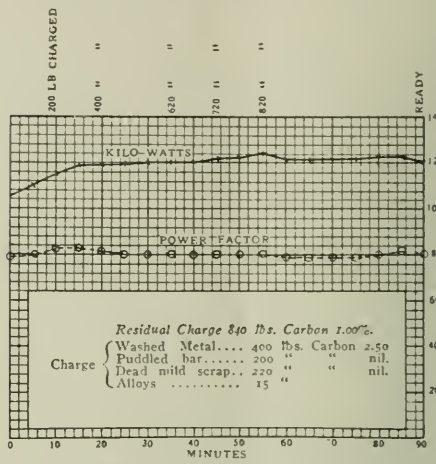


FIG. 2.—POWER CONSUMPTION AND POWER FACTOR. CHARGE 800 LB.

After the first melt was obtained, 112 lb. of brass were charged and melted in about 30 minutes, with a power supply of about 30 kw, and twice the quantity only required 50 kw, melting in the same time. The first heating of the crucible required 42 kw-hours, and took 1½ hours.

The brass produced was remarkably sound, and, when tested against that produced in the ordinary way, showed excellent results.

In addition to this work, other experiments have been carried out on a three-phase furnace, the lining being of firebrick. The result of these experiments shows a power consumption of 18 kw-hours per 100 kg, or about 185 kw-hours per gross ton.

The "pinch" effect is far less intense in the use of three-phase current as compared with single-phase.

Combination Induction Furnace.—This is known as the Roechling-Rodenhauser furnace and is used for melting and refining. (See former descriptions in this journal, especially Vol. VI, pp. 10 and 458.)

The author gives a great many details of the operation in a 2-ton, three-phase, Roechling-Rodenhauser furnace. Figs. 3 and 4 are diagrams showing the gradual procedure of the refining operations in course of time. Fig. 3 gives the analyses of slag samples. Fig. 4 the analyses of steel samples.

Details of a run in an 8-ton furnace are next given. The timetable of the various operations, giving also the power in kilowatts at the different times (the voltage varying between 3180 and 4620 and the current between 115 amp and 116 amp) is as follows:

12.15 Charged molten Bessemer metal, 440 kw....1st sample.
12.50 Charged portion cold scrap, 500 kw.
1.50 Charged final portion cold scrap, 600 kw.
2.30 All melted, 570 kw.....2d sample.
2.50 95 kilos lump lime and 50 kilos roll scale added, 580 kw.

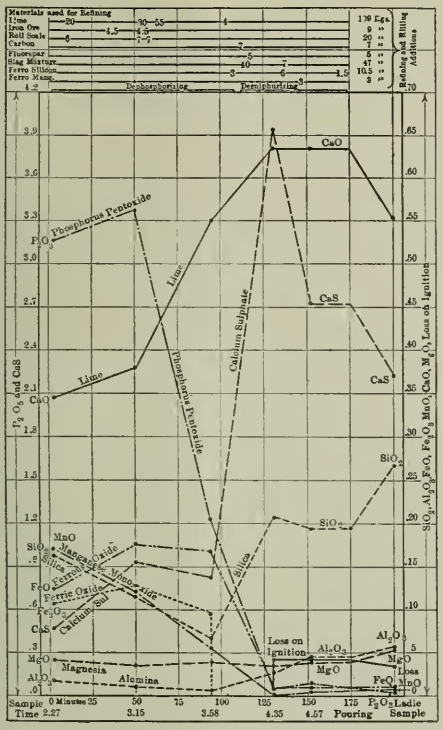


FIG. 3.—SLAG SAMPLES FROM 2-TON, THREE-PHASE COMBINATION FURNACE.

3.00 Slag fluid, 580 kw.....3d sample.
3.10 160 kilos lime added to stiffen slag.
3.30 Removing slag. 45 kilos lime added to stiffen remaining slag. 500 kw.
4.00 Removing slag. 395 kw.
4.10 Charged 28 kilos carbon and 12 kilos Fe-Si.
4.15 355 kw.....4th sample.
4.20 100 kilos desulphurizing slag mixture charged.
4.25 10 kilos fluospar charged.
4.27 Stirred bath.
4.32 295 kw.....5th sample.
4.35 25 kilos Fe-Si thrown over slag.
4.38 Rabbled slag (brown color).
4.42 4 kilos Fe-Si and 4 kilos slag added.
4.45 Charged 45 kilos Fe-Mn. 295 kw.
4.50 Rabbled slag.
4.55 Rabbled slag. 450 kw.
5.00 450 kw.....6th sample.
5.15 Tapped. 400 kw.
15 kilos Fe-Si added to ladle when casting....7th sample.

The steel was desired low in carbon, phosphorous and sulphur, and, on that account, extra refining materials were used.

On referring to the timetable and purification curve (Fig. 5) it will be seen that the phosphorus was at its lowest just as soon as the cold metal was melted. The process of desulphurizing was carried on for another 1½ hours, thereby consuming 800 kw-hours, together with a large quantity of lime and roll scale. The extra cost of these materials amounted to 6½ cents per ton, and the labor would be equivalent to about 20 cents per ton, and power 66½ cents per ton, the total amounting to about 93 cents per ton. The lining would also have produced extra

APPROXIMATE AMOUNT OF STEEL, 6 TONS.

Fluxes, etc., Used	Kilos per Charge	Kilos per Ton			
Lime used (255 + 45)	300	50	at $1.00	per ton	$0.05
Roll scale used	50	8⅓	" 2.50	" "	0.021
Ferro-Silicon (12 + 25 + 4 + 18)	56	9⅓	" 66.00	" "	0.616
Ferro-Manganese	45	7½	" 45.00	" "	0.30
Desulphurizing slag mixture	104	17⅓	" 5.00	" "	0.10
Fluospar	10	1⅔	" 5.00	" "	0.009
Carbon	28	4⅔	" 5.00	" "	0.023
					$1.12

tonnage. This waste would not be allowed in this country, but, on considering that high-grade steel is produced, for which a very high price is obtained, the extra cost per pound amounts to 0.04 cent, which is very little.

This must not be regarded as the usual practice, but was due to a lack of care on the part of the workmen.

Not only did the extra oxidizing cost the 6½ cents for the

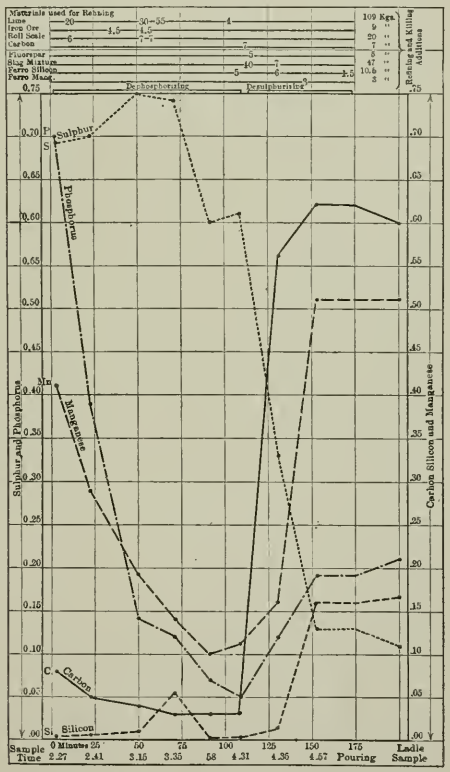


FIG. 4.—STEEL SAMPLES FROM 2-TON, THREE-PHASE COMBINATION. FURNACE.

lime and oxide, but, on calculating it out, it was found that the 28 lb. of ferro-silicon was required to take care of it, which is equivalent to 30 cents per ton, and the costs as previously given are really higher than they ought to be.

The total amount of power consumed was 2400 kw-hours, equivalent to 400 kw-hours per ton, using part cold metal.

The costs, then, on this material are as follows:

Refining materials	$1.12
Power, 400 kw-hours, at 5 cents	2.00
Labor, three men $9, at 14⅔ tons per turn	0.62
Lining	0.30
	$4.04

But it has been shown that as much as $1.23 were wasted on power, labor, etc., to which must be added another 7 cents'

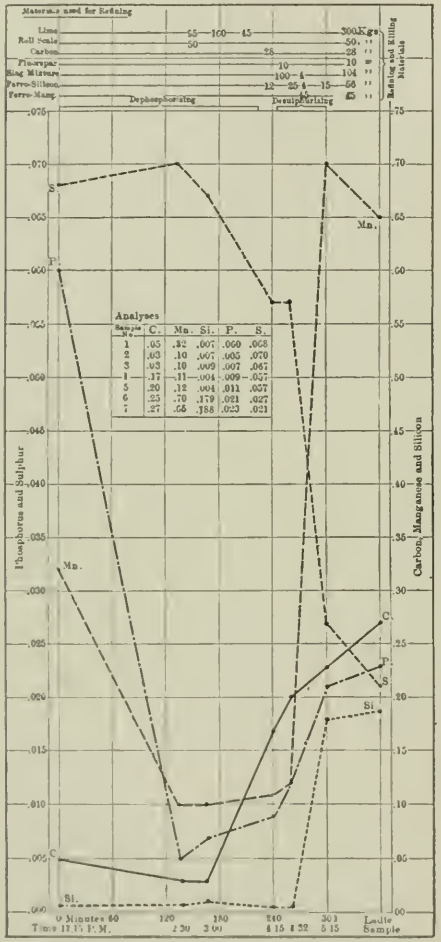


FIG. 5.—STEEL SAMPLES FROM 8-TON SINGLE-PHASE COMBINATION FURNACE.

waste on lining, making a total of $1.30 to be deducted; therefore, the costs should be:

As observed	$4.05
Less waste	1.30
Cost should be	$2.75

Finally, an estimate is given of the operation of a 16-ton refining furnace.

As to the results of regular operation of the 8-ton furnace in Volklingen, the following table gives the analyses of 42 consecutive tests.

Basic Steel P.	Basic Steel S.	Electro Steel C.	Electro Steel Mn.	Electro Steel Si.	Electro Steel P.	Electro Steel S.	Remarks
0.049	0.061	1.01	0.33	0.15	0.020	0.022	For heating 4,994 kw.-hrs.
0.059	0.065	1.04	0.33	0.18	0.012	0.024	
0.050	0.067	0.79	0.33	0.26	0.012	0.029	
0.060	0.069	0.16	0.33	0.015	Trc.	0.032	It was afterwards found that the lime was running high in sulphur, which would account for the erratic results.
0.054	0.067	0.64	0.89	0.16	0.020	0.016	
0.072	0.079	0.96	0.33	0.084	Trc.	0.014	
0.067	0.067	0.55	0.83	0.15	0.026	0.018	
0.048	0.059	1.20	0.33	0.12	0.015	0.020	
0.055	0.087	0.92	1.06	1.12	0.018	0.020	
0.059	0.079	0.23	0.47	0.019	0.012	0.055	
0.053	0.071	0.57	0.87	0.24	0.018	0.029	
.052	.063	1.02	0.39	.18	.016	.018	
0.050	0.067	0.82	0.34	0.096	0.024	.016	
.058	.061	0.80	0.37	.054	.021	.020	Most of the high sulphur steels are low in carbon.
.059	.065	0.91	0.40	0.16	.020	.018	
.043	.089	0.92	0.34	.070	.013	.024	
.070	.075	0.18	0.40	.024	Trc.	.038	
.072	.069	0.86	0.40	0.10	.023	.019	
.058	.081	0.62	0.96	0.16	.024	.020	
.060	.067	0.07	0.37	.028	Trc.	.059	
.056	.097	0.66	0.82	.066	.023	.032	
.089	.091	0.08	0.44	.024	.021	.030	
.076	.055	1.00	0.84	0.14	.010	.018	
.053	.123	0.10	0.40	.030	Trc.	.038	
.065	.079	0.93	0.87	0.11	.024	.020	
0.062	.071	1.01	0.84	0.19	.020	.024	
.078	.073	0.82	0.81	0.11	.025	.027	
.061	.067	0.88	0.31	0.25	.016	.032	
.051	.077	0.59	0.78	0.22	.019	.029	
.052	.059	1.06	0.75	0.20	.015	.024	
.035	.057	1.10	0.86	0.20	.013	.020	
.060	.063	1.10	0.34	.047	.015	.022	
.065	.079	0.98	0.34	0.23	.012	.014	
.071	.048	1.02	0.34	0.22	.013	.018	
.052	.077	0.83	0.87	0.20	.019	.020	
.064	.061	0.92	0.84	0.16	.020	.014	
.063	.063	0.83	0.79	0.13	.014	.012	
.060	.055	0.92	0.37	0.17	.010	.020	
.068	.075	1.00	0.23	0.15	Trc.	.024	
.072	.079	1.08	0.28	0.22	Trc.	.016	
.053	.065	0.88	0.76	0.13	.012	.020	
.075	.071	0.30	0.76	0.22	.014	.029	

Ductile Tungsten and Molybdenum.

A paper by Dr. Colin G. Fink gave the first official information on the latest achievement of the Research Laboratory of the General Electric Company, which has attained success in producing ductile tungsten and molybdenum. This achievement is of utmost importance for the further development of the tungsten-filament lamp.

Tungsten is described in all of the text books as a brittle gray metal and numerous attempts have been made to reduce it to ductile form, as is evidenced by publications emanating from various research laboratories. Roscoe and Schorlemmer, in their latest (1907) edition of their "Treatise on Chemistry," state that "the purest forms of tungsten at present obtainable are hard and brittle and are not ductile, either at ordinary temperatures or when heated."

The metal has ordinarily been obtained in commerce in the form of a dark gray powder usually made by the reduction of the oxide by hydrogen or by carbon. This powder, when obtained in the open market, is generally impure, and is purified by various well-known methods, particularly if the metal is to be used for filaments for incandescent lamps. These filaments are now in commercial use in this country and abroad, and have been made on a large scale. Even in ordinary commercial lamps the filaments are of a degree of purity so high that no impurities can be discovered by the most searching known methods of chemical analysis. Not only is this true, but these filaments are during the course of commercial production exposed to temperatures high enough to drive out by mere vaporization almost any conceivable impurity.

Nevertheless, these filaments show no traces whatever of ductility or even pliability, but, on the contrary, though they are strong enough for mounting in commercial lamps, they are exceedingly brittle and incapable of taking a permanent set.

Attempts have hitherto been made to produce ductile tungsten by various purification processes, but all without success, as by varying the ore from which the tungsten is obtained, as by trying first Wolframite, an iron magnesium tungstate, and then Scheelite, the calcium tungstate. Whichever ore is used it is customary to produce from it the yellow oxide, and additional purity has been sought by repeated precipitations.

Various methods of reduction have been tried, and among other reducing agents used have been hydrogen, carbon, and aluminium. Reduction has also been effected by electrolytic

methods. Since tungsten produced in this way has been so pure that no impurities could be detected by ordinary chemical or physical means, and yet retained its characteristic hardness and brittleness, it has generally been concluded that the metal is entirely lacking in that physical property which is ordinarily called ductility.

The ductile tungsten now produced at the Research Laboratory of the General Electric Company would seem to be a new substance from the point of view of the physical chemist, and for this reason its properties described in Dr. Fink's paper are particularly interesting.

Ductile tungsten is a bright, tough, steel-colored metal, which can be drawn into the finest wire much below one-thousandth of an inch. The tensile strength of the wire increases as the drawing proceeds, or, in other words, the more the metal is mechanically worked, the tougher it gets. In the following table a few figures on the strength of tungsten wires are given. They are the average obtained from a large number of measurements.

Tensile Strength.

	a. Tungsten Wire.		
	Mils.		
5.0	2.8	1.5	1.2
	Pounds Per Square Inch.		
460,000	480,000	550,000	580,000
490,000	530,000	600,000	610,000
	b. Molybdenum Wire.		
200,000	230,000	270,000	.....
260,000	270,000	310,000	.....

The values of tensile strength are given in pounds per square inch and the diameters in mils or thousandths of an inch. A piece of hard-drawn piano wire, tested with the same apparatus, registered, on the average, 507,000 lb., the diameter of the wire being 0.003 in.

Aluminium behaves very similarly. Cast aluminium measures but 17,000 lb. per square inch, whereas the drawn metal has a tensile strength of 36,000 to 39,000 lb. per square inch.

As regards the density or specific gravity of ductile tungsten, the values likewise increase with the amount of working. The values for ductile molybenum were also determined by Dr. Fink.

Specific Gravity.

	Tungsten.		Molybdenum.
	Before Drawing.		
	18.81		10.02
	After Drawing.		
Mils.			
150	19.30	19.30	
10	19.58	19.64	10.04
1½	19.86	20.19	10.29
		(21.82)	10.32

Martin (1907) found the density of melted tungsten (analyzing 98.96 per cent) to be 16.28; Moissan (1896) and Weiss give the values 18.70 and 18.72 for the brittle metal. As is seen from the table, the density increases very markedly with the amount of mechanical working applied. This same phenomenon is well known in the case of copper; according to Marchand and Scheerer its density is 8.92 and that of rolled and hammered copper 9.95. Distilled zinc has a density of 6.92 and wrought zinc 7.25.

The electrical resistivity and the temperature coefficient of the two metals are given in the next table, determined by the Wheatstone bridge method. The resistance was measured at room temperature and at 170°, employing two oil thermostats.

	Resistivity (25° C.) Microhms per Cubic Centimeter.		Temp. Coefficient from 0° to 170° C.
Tungsten	d	6.2	0.0051
	a	5.0	
Molybdenum	d	5.6	0.0050
	a	4.8	

The values marked d are for hard drawn wires; those marked a were obtained after annealing. These resistivity values are a good deal lower than those given by Gin.

The coefficient for Cu (0° to 160°) is 0.00445 (Reichardt). The hardness of both W and Mo depend very much upon the amount of mechanical working the metals have undergone and upon the presence of impurities.

Chemically, the two ductile metals behave similarly in most respects. The drawn wire retains its luster almost indefinitely. Both metals are readily attacked by fused oxidizing salts, such as $NaNO_3$, $KHSO_4$ and Na_2O_2. Acids (HCl, HNO_3, H_2SO_4) attack tungsten very slowly, but molybdenum rather steadily. Dr. Fink has heated fine-drawn tungsten wire in a mixture of chromic and sulphuric acid for 16 hours, but could detect only a very small loss in weight.

Original weight, 16.7330 grams; after 16 hours, 16.7329 grams.
" " 1.3638 " " 14 " 1.3635 "

In conclusion Dr. Fink states that this work was carried out at the Research Laboratory of the General Electric Company, at Schenectady, N. Y., and that naturally a number of chemists and research men have been active in it, foremost among whom is Dr. Coolidge.

Treatment of Cobalt-Nickel Ores.

A paper by Mr. CAMILLO C. CITO, of the Irvington Smelting & Refining Works, of Irvington, N. J., described a new process for the treatment of cobalt-nickel ores, rich in silver and arsenic, by combined melting and electrolytic operations.

The so-called Canadian cobalt ores have attracted the attention of investors and refiners since their discovery in 1903 because of their unusual richness in silver. By 1907 the yield of silver from these ores amounted to 5 per cent of the entire world's supply, but this was obtained mainly from the richest veins, the lower-grade ore being thrown aside to await the development of an economical method of extraction.

Besides silver these ores contain nickel, cobalt, and arsenic, the latter element often being present to the extent of 40 per cent. The arsenic has proven a serious obstacle in the working of the ore, since, if wasted and the arsenic allowed to escape, it endangers the lives of the community besides entailing a loss of silver equal to about double the amount of arsenic removed. If these volatilized products are condensed expensive chambers must be provided and the condensed products again separated.

The mines are situated in northern Ontario, Canada, to the east of Lake Superior, and with the center in Cobalt City.

According to the eighteenth annual report of the Bureau of Mines, Ontario, 1909, Vol. 18, Part 1, the mining camp consists now of 40 different mines, which shipped from 1904 to 1907 inclusive, a total of 48,544 tons of ore. The rapid increase in the output is best shown in the fact that in 1908 alone the total shipments amounted to 25,362 tons.

The more important mines of the district are at present: La Rose, Nipissing, O'Brien, Drummond, Crown Reserve, Buffalo, City of Cobalt, Temiskaming, Right of Way, Temiskaming & Hudson Bay, Trethewey, etc. They all ship between 500 and 5000 tons of ore a year.

The rock consists mostly of greenstone, crossed by granite of a later formation. The ore deposits are found in this granite, partly mixed with quartz, feldspar, pyrites, magnetite, and chlorites. The veins differ widely as regards thickness, as well as disposition of the mass; they contain chiefly the following minerals: Argentite, pyrargyrite, smaltite, cobaltite, erythrite, niccolite, gersdorffite, chloantite, arsenopyrite, etc.

Experiments.—The idea suggested itself to try out first the practicability in this special case of all the methods, either wet or dry, already in use for the treatment of similar silver ores. Mr. Cito mentions only briefly these experiments, as all gave more or less unsatisfactory results.

After many failures he succeeded in finding a combined method consisting of a melting process followed by an elec-

trolytic separation. The process is already patented, or filed for patent, in all the principal countries. (U. S. A. Patents 949,058, 949,059, 949,261).[1]

The experiments described below were made with an ore of the following composition:

4.12 per cent silver (1200 oz. per ton).
19.6 per cent arsenic.
11.2 per cent cobalt.
5.7 per cent nickel.
No lead.
No copper.
0.5 per cent antimony.
18.5 per cent iron and alumina.
11.0 per cent silica.
1.7 per cent calcium oxide.
1.9 per cent sulphur.

I. Experiments in Mechanical Concentration.—The ore was ground and passed through a sieve of 200 meshes per square centimeter. Fifty kilograms of ore gave in this way

46 kg fines = 92 per cent.
4 " metallics = 8 " "

The distribution of the principal metals was the following:

	Fines.	Metallics.
Silver	2.3 per cent.	25.0 per cent.
Arsenic.....	18.5 " "	32.5 " "
Cobalt.....	11.5 " "	8.3 " "
Nickel.....	5.8 " "	5.1 " "

As far as the principal metal, silver, is concerned, it can be seen that about half of it is concentrated in the metallics. The fines, however, are for the same amount poorer and their further treatment, therefore, less profitable.

II. Wet Processes.—The leading idea in these experiments was either to leach the silver out of the raw or roasted ore and collect nickel and cobalt and arsenic in the residues, or to dissolve first nickel-cobalt and arsenic and to recover the silver from the slimes.

All the wet processes known as being in practical use were tried with both raw and roasted ore, at different temperatures and with varying concentration of the dissolving agents, as follows:

(a) *With raw ore:* Leaching with potassium cyanide, hydrochloric acid and sulphuric acid.

(b) *With roasted ore:* Leaching with sodium chloride (Augustin process), sodium thiosulphate (Patera process), sodium-copper thiosulphate (Russel process), potassium cyanide (cyanide process), acids (hydrochloric and sulphuric acid), magnesium chloride, ammonia.

The solutions as well as the residues obtained in the enumerated experiments have been analyzed as far as the principal metals (silver, nickel, cobalt, arsenic) are concerned. Although the experiments were carried out on a small scale, and therefore the conditions for complete reactions were favorable, not in a single case was a satisfactory result obtained. With both raw and roasted ore the silver, on the one hand, did not dissolve completely, even with large quantities of solvent; nickel, cobalt, and arsenic, on the other hand, when they ought to have dissolved, remained to a large percentage in the residue, especially arsenic, while considerable quantities of silver passed into solution.

The working up of the solutions and the residues would, therefore, lead to a great many intermediate products and would become very complicated.

III. Dry Processes.—(1) Roasting Experiments.—The object was:

(a) To drive out the arsenic by roasting the ore alone or with certain fluxes.

(b) To transform the silver into a soluble compound with the aid of certain fluxes, and to extract it by one of the methods before mentioned.

For the experiments on a small scale the fines passed through the 200 meshes per square centimeter were used; the roasting was done in a small muffle furnace.

When roasting without fluxes at a steadily increasing temperature the ore was dead-roasted after 8 hours. The charge showed a loss of 8 to 12 per cent of its original weight, corresponding to a loss of silver of from 15 to 24 per cent, while the contents of arsenic changed but little.

In order to prevent the formation of arsenates the same fines were mixed with different amounts of powdered charcoal and roasted for six hours. The roasting losses in these cases varied from 5 to 10 per cent. At the same time the loss of silver varied from 20 per cent and 30 per cent, with only a trifling loss of arsenic.

With the idea of eliminating the arsenic in the form of sulphide of arsenic, the fines were mixed with raw pyrites and roasted at a slowly increasing temperature for 8 hours. The loss in the weight of the mixture amounted to 8 per cent, according to a loss of 6.3 per cent silver and 7 per cent of arsenic.

For a chlorination process the same fines as before were first dead-roasted, then mixed with charcoal powder and roasted a second time in order to decompose the arsenious salts; this product was heated a third time at a dark red heat with a certain amount of sodium chloride. After this third treatment 22 per cent of the original arsenic contents were eliminated, but at the same time there was a loss of 35 per cent of the silver.

The products of these roasting experiments were used for testing the different wet methods mentioned above.

The results obtained when roasting the fines in small quantities were fully confirmed by an experiment on a commercial basis; 9,060 kg of ore was given a preliminary roasting in a reverberatory furnace, then mixed with galena ore and roasted a second time in order to get the material in a lumpy form for subsequent treatment in the blast furnace.

The ore contained 3.77 per cent silver, equal to 341.56 kg in the whole charge. The roasted material weighed 8,045 kg, assaying 3.42 per cent, or 275.13 kg of silver. Therefore the loss of silver in this roasting amounted to 66.43 kg, or 19 per cent of the original amount, not including the contents of the flue dust.

At the same time it was ascertained that only a small amount of the arsenic was eliminated. The original 9060 kg of ore contained 18.9 per cent of arsenic, equal to 1712 kg, while in the roasted 8045 kg still 19.2 per cent, or 1545 kg, of arsenic were found, corresponding to a loss of 167 kg, or 9.8 per cent of the original amount.

(2) Melting Process.

(a) Melting with Lead Fluxes.—The ore which had been roasted with charcoal and galena, as described above, was melted in a blast furnace with the usual fluxes, including some lead-bearing material, producing metallic lead, a speiss, a matte and a slag. Although the furnace was running well, the quality of the products was quite unsatisfactory. The lead was very brittle, the speiss could not be cleanly separated from the matte by mechanical means, and the slag was not fluid. The distribution of the principal metals in these four products was as follows:

	Lead, Per Cent.	Speiss, Per Cent.	Matte, Per Cent.	Slag, Per Cent.
Silver	1.78	0.32	1.52	0.013
Lead		4.	4.3	3.3
Copper		0.8	9.	Trace
Arsenic		34.	0.3	
Nickel-Cobalt		41.	5.5	1.5
Sulphur		4.	21.	

Of 290.76 kg silver contained in the original charge (including the fluxes), 275.03 kg were recovered in the four products given above. The loss of silver amounts therefore to 24.73 kg, equal to 8 per cent, not including the flue dust. If no account is taken of the loss in the roasting process, the output in silver in the blast-furnace operation is fairly satisfactory. The accumu-

[1] See our March issue, page 148.

lation of nickel and cobalt in the speiss and matte, however, is deficient; indeed, of 1246 kg of nickel and cobalt in the original charge, only 319 kg appears in speiss and matte, corresponding to 25.6 per cent, by far the greater part of these metals passing into the slag.

From the above it results clearly that this kind of ore cannot be melted successfully in a blast furnace with a lead charge, even without considering the loss of silver which already occurs in the roasting operation. Nickel and cobalt and especially silver are found in all the four products of the furnace in such amounts that the further treatment of these would be very complicated and uneconomical.

(b) Melting with Copper Fluxes.—In order to avoid the many products (lead, speiss, matte, slag) which result from melting these ores with lead fluxes, a great number of experi-

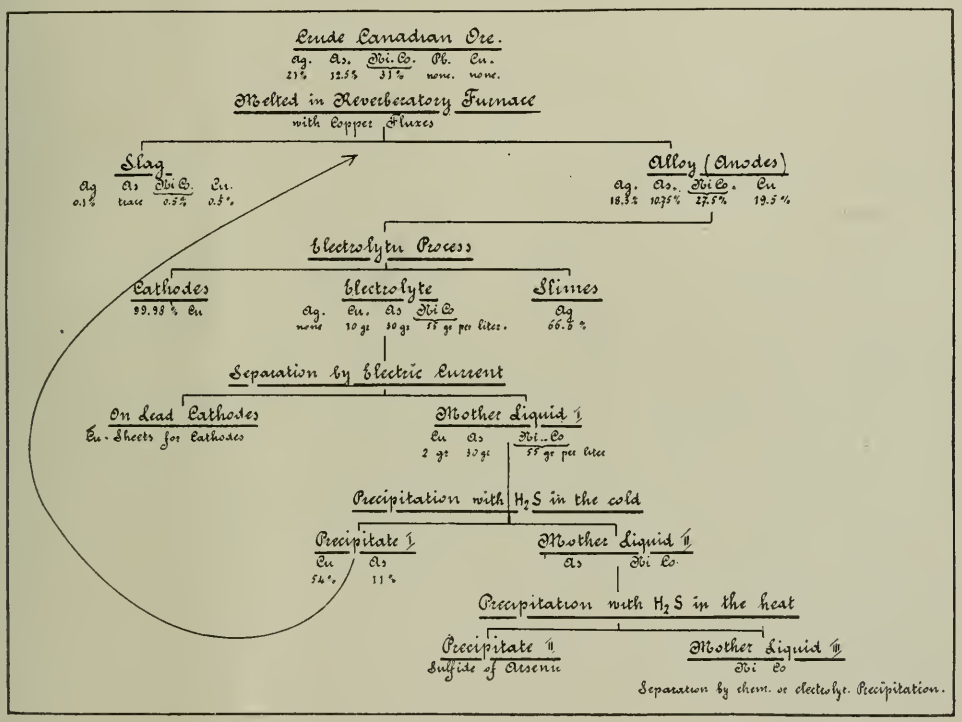


DIAGRAM OF PROCESS FOR THE TREATMENT OF COBALT-NICKEL ORES.

mental meltings with copper fluxes were carried out with the intention of producing an alloy containing all the silver, nickel and cobalt and a slag as poor as possible in these metals.

It should be expected that, owing to the small amount of sulphur in the ore, no matte would be formed. The absence of lead is likewise of the utmost importance, and the greatest care is taken that, contrary to the usual processes of treating silver ore, this metal does not enter the material.

To avoid silver losses by roasting, only the raw ore was taken into consideration for these experiments, because it is not possible, as shown in the before-mentioned roasting experiments, to drive out the arsenic to any extent by roasting without losing more or less silver.

The final object of this part of the research consisted in producing an alloy which could be worked up as an anode in an electrolytic bath. At the same time this alloy had to take up as nearly as possible all the arsenic, because, first, as shown before, even small volatilization of arsenic is always accompanied by considerable loss of silver, and, secondly, it is highly important to reduce the escape of the extremely dangerous arsenical fumes to a minimum.

After a number of failures it was found that in order to be successfully worked up as anode, the alloy had to contain the different metals in certain definite proportions.

The amount of copper added has to vary according to the amount of silver, arsenic, nickel and cobalt present in the ore.

As far as the principal metals are concerned, the ores run as follows:

Silver	300—10,000 oz. per ton.
Nickel and cobalt	10—30 per cent
Arsenic	5—40
Lead	None.

The treatment of the ore consists of two operations, viz.: 1, the smelting process; 2, the electrolytic process.

A. The Smelting Process.—The ore is smelted without any preliminary treatment, such as roasting, etc., in a reverberatory furnace with a calculated quantity of copper and the necessary amount of the usual fluxes.

The addition of the copper practically eliminates arsenic from the fumes in the smelting process because the arsenic is absorbed by the metallic copper.

Only two products result from this smelting process:

(1) *An alloy*, containing, besides copper, all the silver, nickel and cobalt, and nearly all the arsenic. On account of the subsequent electrolytic refining of this alloy, it is important that the different metals composing it, as mentioned before, should be present in a certain proportion, viz., it is necessary that: (a) The percentage of copper be two-thirds of that of the nickel and cobalt, but (b) at least the same as that of the arsenic, and

preferably not less than the silver. The alloy is tapped from the furnace directly into the anode molds. Besides the alloy there is obtained:

(2) *A slag*, containing a very low percentage of silver, nickel and cobalt. This slag can be used to advantage as a flux in the blast furnace, where the last traces of the metals can be easily recovered.

For instance, when melting 174 lb. of ore with the proper amount of copper and the necessary fluxes, no fumes were visible. The result of the melting is to be seen in the following table:

	Canadian ore before melting 174 lbs.	Alloy after melting 195 lbs.	Slag after melting 56 lbs.
Silver	6200. oz. p. ton	5500. oz. p. ton	20. oz. p. ton
Arsenic	12.5 per cent	10.75 per cent	Trace
Nickel and cobalt	31. per cent	27.5 per cent	0.5 per cent
Lead	None	None	None
Copper	None	19.5 per cent	0.5 per cent
Iron	3.5 per cent	0.5 per cent	7.6 per cent
Sulphur	1. per cent	1.2 per cent	
Insoluble	9.64 per cent		

All the valuable metals pass into the alloy; only a small fraction of the silver appears in the slag (29 oz. per ton). The data concerning the arsenic are noteworthy: 174 lb. of raw ore containing 12.5 per cent arsenic, after the melting, yields 195 lb. of alloy containing 10.75 per cent arsenic; allowing for small errors in the analysis, it results clearly from these figures that in melting practically no arsenic has escaped in the fumes.

B. The Electrolytic Process.—The different metals of the alloy are separated from the anode in an electrolytic bath by electric current.

A great number of experiments were carried out with alloys of unvarying composition in small electrolytic baths, in which the utilized anode surface reached 7 square decimeters. Later on, the electrolytic process was tried out in large baths with a utilized anode surface of ½ square meter; the results obtained here were identical with those in the small baths.

The electrolyte consists of a solution of copper sulphate in water, which during the whole electrolytic process contains at least 10 grams of copper and 5 grams of free sulphuric acid per liter.

The cathodes are sheets of pure copper.

The density of the electric current has to be regulated according to the composition of the anodes and electrolyte and according to the temperature.

The temperature is kept advantageously at 60 deg. Cent. or higher, and is otherwise regulated by the percentage of arsenic in the anodes

In case proper conditions prevail, viz., if the electrolyte as well as the temperature and the electric current correspond to the composition of the alloy, the following results will be obtained:

(1) The copper contained in the anode is deposited in the form of pure electrolytic copper on the cathode.

(2) The silver of the anode is quantitatively precipitated in the slimes.

(3) All the nickel and cobalt dissolved from the anode remains in solution in the electrolyte.

(4) At the proper temperature the larger part of the arsenic is found dissolved in the electrolyte, the balance going into the slimes with the silver

Further Treatment of the Different Products of the Electrolysis.—This is being done in three steps.

(1) The *copper* deposited on the cathode is exceedingly pure. A determination of the copper yielded 99.98 per cent Cu. The great advantage of this simple, as well as cheap, process is obvious, as the copper originally added as a flux is recovered as a commercial product.

(2) The *slimes*, of course, are rich or poor according to the percentage of silver in the anodes. Under all circumstances the slimes contain all the silver of the anode, as neither in the cathode nor in the electrolyte can traces of silver be determined. The small quantities of arsenic which go into the slimes, together with the silver, can be easily removed by a light roasting or dissolving process.

For instance, an ore, originally containing 600 oz. silver per ton, 19 per cent arsenic and 17.5 per cent nickel-cobalt, yielded slimes of 12,000 oz. silver per ton; while the slimes of another ore that ran 3500 oz. silver per ton, contained 24,500 oz. silver per ton.

These silver slimes can, without any difficulty, then be refined in the test furnace by the usual cupellation process. Also, here the advantage of the process can be clearly seen, as the silver, being the principal component of the ores, is extracted in a very simple as well as safe and especially rapid way.

(3) The *electrolyte* containing the whole of the nickel-cobalt and the larger part of the arsenic can be brought up to the following concentrations:

Silver	None.
Copper	10. grams per liter.
Nickel-cobalt	55. grams per liter.
Arsenic	30. grams per liter.

By using insoluble anodes almost all the copper contained in the electrolyte can be precipitated as pure copper on thin lead cathodes. The density of the current has to be lowered according to the decrease of copper in the solution, and at the same time the temperature has to be raised. With a current density of 150 to 200 amp per square meter and a temperature of 80 deg.-90 deg. Cent., the copper in the electrolyte can thus be reduced to about 2 grams per liter.

The precipitated copper is very smooth, and can easily be detached from the lead sheets and used as cathodes in the regular electrolytic baths.

The balance of the copper is advantageously precipitated by sulphuretted hydrogen in the cold, whereby very little sulphide of arsenic is formed. This copper product can be used over as flux.

All the arsenic can then be precipitated by sulphuretted hydrogen upon heating as practically pure sulphide of arsenic, which can be placed on the market as such.

Nickel and cobalt can now be recovered, together or separately, by any of the methods usually employed, such as electrolysis or by chemical precipitation.

The principal advantages of this process are:

(1) Utmost simplicity and consequently very low running expenses.

(2) Great rapidity in extracting the most important metals, and especially the silver.

(3) Remarkable safety by reducing to a minimum the danger from arsenic vapors which are liable to develop in the treatment of such material containing a large amount of arsenic.

(4) Direct output of the principal metals, limiting at the same time the forming of by-products.

In contrast to the above-described process the methods used at present for the direct production of silver from these ores seem to yield far less satisfactory results.

According to the eighteenth annual report of the Bureau of Mines of Ontario, 1909, Vol. 18, Part I, "there were three reduction plants in Ontario treating ore from the mines of Cobalt last year, namely, those of the Canadian Copper Company, at Copper Cliff; the Deloro Mining & Reduction Company, at Deloro, and the Coniagas Reduction Company, at Thorold. Through these works there were passed an aggregate of 6958 tons of ore containing 11,658,008 oz. of silver, of which 8,072,958 fine oz. were recovered, the remainder, 2,685,050 oz. 23 per cent, being contained in the *speiss* product reserved or exported for further treatment."

Considering, moreover, that probably a great part of the silver produced referred to in this report was shipped in the form of nuggets containing 75 to 85 per cent of silver, the output appears even less favorable, while in the process which forms the subject of this paper a yield of fine silver of 98 per cent of the contents of the ore can be directly obtained. The paper was briefly discussed

A New Radiation Pyrometer.

A paper by Mr. C. E. FOSTER, of the Taylor Instrument Companies, gave a description of a new radiation pyrometer.

For the measurement of the very high temperatures met with in electric furnace processes, and also in open hearth work, there is practically no instrument which can give any satisfaction, except those of the radiation type. There is also another field of temperature measurement which can only be handled by the radiation pyrometer, and that is what may be called a continuous furnace process. By this is meant any process in which the furnace is maintained at a steady temperature, while the material under treatment is fed into and out of the furnace in a more or less continuous manner. Accordingly, the temperature of the material will vary with the time it remains in the furnace. It is obvious that in such a case, even though the temperature may be low, the mere measurement of the furnace temperature by means of a thermo-couple will not give reliable results. What is wanted is the actual temperature of the material as it issues from the furnace.

To make successful measurements in either of these cases it is generally necessary that the operation of the pyrometer be very simple and quick. The more complications there are attached to its use, the more the likelihood of errors and unsatisfactory readings, because the conditions obtaining are not conducive to calm and accurate adjustments.

These considerations practically confine the use of the purely optical pyrometer to the laboratory, or at any rate, to the laboratory staff. With a view to providing a radiation pyrometer which is practically "fool-proof" and requires no manipulation, Mr. Foster has designed his fixed focus instrument. In this instrument simplicity has been the governing feature. At the same time its construction avoids errors in calibration and use.

Fig. 1 shows the instrument being used to measure the temperature in a furnace used for continuous processes and the user is pointing it so as to measure the temperature of the piece of steel about to be removed from the furnace.

Fig. 2 shows a diagram section of the tube. At the front end is a diaphragm E F, while at the opposite end is a concave mirror C. The radiant heat from any hot body, such as A B, enters the opening in the diaphragm and a fixed proportion of it strikes the mirror. It is then collected at a focus D. The arrangement is such that there will be, at D, a focused heat image of the diaphragm, filled with the radiant heat from the hot body A B. At D is placed the sensitive part of the instrument, and this is the hot junction of a thermo-couple.

Looking at the diagram in Fig. 2 it will be seen that a body covering any larger angle than A G B would not be able to radiate any more heat to the mirror C. Any extension of the hot body beyond the limits indicated by the dotted lines G A and G B would only be able to radiate heat to the walls of the tube and these are made non-reflecting. The dotted lines in the diagram define a cone whose apex is at G. For any given hot body the maximum working distance will be a fixed multiple of the diameter or other smaller dimension of the body. Any working distance less than this maximum will simple result in the outer edges of the hot body being out of the measurement. In all cases the action is just as if the hot body were actually at the diaphragm E F.

The instrument is designed so that the mirror throws on to the thermo-couple junction at D a focused image of the aperture E F. This fact also supplies the reason for the name "fixed focus." The position of the apex G is shown on the outside of the pyrometer tube by the metal ring which carries the fixing stud and nut. For the sake of convenience, the proportions of the hot body to the working distance are not shown to scale in the diagram. In the standard tube the actual proportion is 1 to 10.

The thermo-couple is composed of two very small wires of different alloys. These are joined together at one end. Their other ends are joined to a suitable electrical circuit including a flexible cable and an indicating millivoltmeter. The radiant heat is concentrated upon the common junction of the two alloys with the result that this junction is heated above the temperature of the junctions of these two alloys to the rest of the circuit. This heating sets up an electro-motive force and as a result a current flows through the millivoltmeter proportional to the heat concentrated, that is, proportional to the temperature of the hot body under examination.

The whole of this sensitive thermo-couple is contained within the pyrometer tube and, therefore, the indications of the instrument are independent of the general air temperature or that of the tube itself. Any change in this temperature will affect the thermo-couple as a whole and the excess temperature of the common or hot junction will depend only on the radiant heat concentrated upon it.

The calibration of the indicator scale is based upon the well-known Stefan-Boltzmann law. This law has received abundant support, both theoretical and practical, and the fixed-focus pyrometer follows it very closely. At the same time the actual calibration of the whole outfit is based on comparison with certain standards, both instruments being pointed at a high temperature furnace so that any departure from the Stefan-Boltzmann law is taken into account in the calibration and does not appear as an error.

FIG. 1.—FIXED-FOCUS PYROMETER IN USE.

Two standard scales up to 2400° Fahr. and up to 3200° Fahr. respectively are being made, and this latter is more particularly suitable for work upon the open-hearth furnace. Of course, there is no difficulty in scaling the instrument for still higher temperatures, as it has no upper temperature limit.

The efficiency of the receiving tube is very high and, accordingly, the indicator is made of a very robust moving-coil pivoted type. It does not require accurate levelling in use, nor is there any clamping or other manipulation necessary. In fact, the whole instrument is always available for immediate tempera-

ture measurement. It has simply to be pointed at the hot body and a steady reading is obtained in about 15 seconds. Each receiving tube is provided with a stud and wing nut mounted upon the center ring so that it can be clamped to any support for use permanently in one place. In this way it can be arranged so as to show the temperature of each billet as it comes from a billet heating furnace.

In the discussion which followed, Mr. Foster said that the

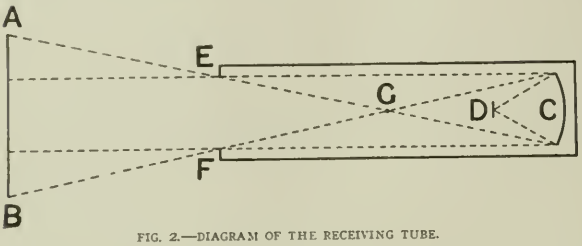


FIG. 2.—DIAGRAM OF THE RECEIVING TUBE.

instructions to the user are simply that the distance of the instrument in use from the furnace, the temperature of which is to be measured, should be less than a certain length, depending on the dimensions of the instrument.

Dry Cells.

The first paper presented at the Saturday session was a paper by Mr. D. L. Ordway, of the National Carbon Company, of Cleveland, on "The Evolution and Present Status of the Dry Cell."

The author first summed up the chief electrical characteristics of the dry cell and mentioned that the short-circuit current is often used as a measure of the "service capacity" of a cell. But this is quite wrong. As is evident from some tests of new and old cells given by Mr. Ordway, there exists practically no relation between short-circuit current and amp-hour capacity.

The decisive test of the quality of a cell should be amp-hour capacity or watt-hour capacity, but much depends on the way in which the cell is discharged. If a cell is discharged through a contact resistance, the value of which is chosen different in different tests, the discharge curve drops, of course, more rapidly the lower the resistance. The most abrupt drops of voltage are at the beginning and at the end of a discharge; these are due to different causes, which were discussed.

But while these curves of continuous discharge are of considerable theoretical interest, they are of small value in practice, since dry cells are generally used in practice on intermittent discharge. According to the kind of service, the use of the dry cell should differ, for instance, with respect to the number of cells in series and in parallel. In ignition service, more than one set of cells in multiple is advantageous, while in light service, like telephone service, it is a positive disadvantage to operate more than one set of cells in multiple.

According to the kind of service desired, the form of intermittent discharge by which the cell is to be tested, should be arranged. In tests of dry cells it is important to pay attention to the temperature. Only tests made at the same temperature should be compared.

As the principal uses of dry cells are in ignition service and in telephone service, special tests have been worked out for both purposes both by the manufacturers and consumers. These commercial tests are reviewed in detail in the paper, which will be printed in full in the *Transactions* of the society.

The paper was briefly discussed by Mr. C. Hering.

Electrolytic Separation of Oil from Condenser Water.

A paper by Prof. H. M. Goodwin and Mr. Ridsdale Ellis, of the Massachusetts Institute of Technology, deals with "the separation of oil from condenser water by electrolysis."

The investigation was suggested by an English patent (10,874 of the year 1902) of Davis and Perrett, who run the condenser water to be purified in a circuiting path through a vessel fitted with iron electrodes. They add to the water "at its inlet and as required a small and regulated quantity of ordinary clean water or of carbonate of soda or of potash dissolved in water." The small particles of oil were said to become "entangled with the detached minute particles of iron or other metal used for electrodes." It is then possible to remove the oil from the water by filtration. "The oily particles are not oxidized or decomposed to any appreciable degree, and the oil can be recovered in the usual manner from the scum that forms on the surface of the water in the vessel or from the precipitate on the filter cloths or in the settling tanks."

The effect of the current is, therefore, to so modify the character of the emulsion that the oil particles may be readily separated from the water by filtration. The process differs from other electrical methods which have been devised for the purification of ordinary water (not only feed-water) in that an electrolyte is here added to give conductivity to the water. In most other cases the impurities in the water form the electrolyte and the flocculant precipitate formed from aluminium or iron electrodes acts as clarifying agent. Of the other methods employed for the separation of oil from water the authors mention the following:

(a) Gravity and mechanical separation, as by a baffle plate separator which Carty has shown will separate 75 per cent of the oil fed to the cylinder of an engine.

(b) Addition of de-emulsifiers which destroy the film on the surface of oil particles, thus permitting them to coalesce.

(c) Filtration through various media.

When pure oil is violently agitated with pure water no permanent emulsion is produced, for if the mixture be allowed to stand, a complete separation of the two constituents takes place. The minute oil globules gradually rise owing to their lesser specific gravity and coalesce as they come into contact with one another. If, however, a small amount of emulsifying agent be present a portion of the oil will be permanently emulsified, i.e., distributed through the water in a so minutely divided state that it cannot be separated from the medium by gravity or ordinary filtration.

The role played by the emulsifier is to produce a film of solid matter over the oil globules so that coalescence in contact is prevented. The more common emulsifiers are such organic substances as soap, gums, proteids and also certain colloids such as ferric-hydrate. Ramsden has shown that exceedingly small amounts of proteid matter are sufficient to effect emulsification. Electrolytes as a class do not act as emulsifiers, although sodium carbonate or hydroxide will emulsify oils of fatty acids owing to the formation of soap films.

Certain solid substances, like lime and clay, also act as partial emulsifiers, forming so-called quasi-emulsions. In this case the substance is retained in suspension by association with the minute solid particles rather than being completely enveloped by a solid film. The emulsifying power of solids depends very largely on the minuteness of the particles.

No definite magnitude can be assigned to particles forming permanent emulsions, but generally speaking they are sufficiently large to be visible under the microscope. Their size, therefore, distinguishes them from colloidal suspensions and the still smaller particles forming true colloidal solutions. Although colloidal molecules are believed to be large, as indicated by their molecular weight, small osmotic pressure, rate of diffusion, etc., they are still so small as to be beyond the resolution of the best microscopes.

Colloids and emulsions possess the common property of migrating under the influence of a potential gradient, and are, therefore, electrically charged. Two distinct hypotheses have been proposed to account for the existence of the charge asso-

ciated with them, which is usually negative in character, as shown by the direction of migration of the particles to the anode. The first hypothesis is due to Coehn[1], who states the general proposition that a difference of potential is always produced at the junction of two dissimilar media, when brought into contact, that one assuming a positive charge which possesses the higher dielectric constant. This readily explains why water with its abnormally high dielectric constant of 80 is positively charged with respect to almost all particles suspended in it. The positive charge observed on such substances as colloidal ferric-hydrate seems an exception to this rule.

The second hypothesis assumes the charges to result from association or absorption of negative or positive ions with the colloidal or emulsified particles or an ionization of the colloidal molecules themselves (see Noyes[2], page 99).

The present authors do not attempt at present to decide the relative merits of these two hypotheses. The fact that the emulsified particles are electrically charged and that this charge which undoubtedly is largely instrumental in preventing them from separating under the influence of gravity from the medium containing them is the important thing to be here recognized.

The authors prepared their emulsions from distilled water and a pure valve cylinder oil. The results of their experiments are summed up as follows:

The oil particles in the emulsions investigated varied in diameter from about 0.0006 cm to 0.00006 cm and are negatively charged, as shown by their migration to the anode under the influence of a potential gradient.

The addition of dilute solutions of sodium carbonate does not produce de-emulsification.

Partial de-emulsification of oil (about 50 to 60 per cent) is produced by the action of an electric current using platinum electrodes.

Complete de-emulsification results from using iron electrodes.

The essential part of the process is the removal of the oil by the basic ferrous carbonate precipitated at the anode coming into contact with the oil particles as they migrate to the anode; mechanical agitation of the emulsion with this precipitate serves, however, equally well to remove the oil.

A microscopic and photomicrographic study of the phenomena shows the oil to be held in the meshes of precipitate.

Colloidal ferric hydrate, a positive colloid, if present in sufficient concentration, effects complete de-emulsification.

Sodium carbonate coagulates colloidal ferric hydrate at a concentration less than that necessary to de-emulsify the oil and hence if colloidal ferric hydrate be formed during the process of electrolysis, it will be coagulated by the carbonate present before it can become effective as de-emulsifying agent. Therefore, colloidal ferric hydrate plays no essential role in the process. It seems that the chief agent in de-emulsification is the freshly formed basic ferrous carbonate resulting from electrolysis. The migration of the oil to the anode assists, however, in a measure, in the association of the two substances. The subsequent oxidization to ferric salts is unessential to the process, although precipitated ferric hydrate was shown to act as a good filter.

The efficiency of a number of precipitates used as filters for separating oil was investigated. Ferric hydrate and red ochre were found to be among the best.

From a photomicrographic study of coagula produced by precipitating emulsifiers and non-emulsifiers in oil emulsions, it appears that in general these substances, such as basic ferrous sulphate, lime, and alumina, which act as emulsifiers, may also serve to de-emulsify a mixture. Precipitation of the positively charged particles takes place on the negatively charged oil globules as nuclei in a manner analogous perhaps to the precipitation of water vapor on negative electrons. With non-emulsifiers, such as barium sulphate and lead chloride, the precipitation does not occur on the globules, but entirely distinct from them.

[1] Coehn, A. Uber ein Gesetz der Electricitätserregung. *Ann. d. Phys.* (3) 64, 217-232, 1898.

[2] Noyes, A. A. Preparation and Properties of Colloidal Mixtures. *Jour. Amer. Chem. Soc.*, 27, 85, 1905.

University of Pittsburgh.

A courteous speech of welcome was then made by Dr. MCCORMICK, who, as the chancellor of the University of Pittsburgh, welcomed the society to the university. They have now a new and fine home in a growing district, where they can build up a big university.

Dr. McCormick spoke of man's kingship over nature. The Greek with all their fine brains did nothing to learn nature and natural forces and to harness them. This was left to the modern scientist and engineer. "Next to the minister of the gospel is the scientist." Not only does he add to the convenience and wealth of mankind, but he betters the place of man as man among the forces of nature.

Dr. McCormick indicated briefly the large plans which they have for the development of the University of Pittsburgh.

Tin Reduction.

A brief paper by Mr. W. O. SNELLING dealt with "an electrolytic method for the reduction of cassierite." The author thought the present methods of reduction of tin from the ores are complicated and expensive. The method suggested by him is based on an observation made in a laboratory experiment in which he had cassiterite in contact with zinc in hydrochloric acid. The cassiterite was directly reduced to metallic tin by cathodic reduction. This system represented a short-circuited galvanic cell with zinc as anode and cassiterite as cathode. When the zinc and cassiterite are in contact, a current passes through the hydrochloric acid solution from the zinc to the cassiterite. Zinc goes into solution and cassiterite is reduced to tin.

On this observation a double-compartment cell is based which was tried by the author on an experimental scale. The cassiterite is cathode in the first compartment and anode in the second compartment. When a current is passed through the first compartment to the cassiterite as cathode, the cassiterite is reduced to metallic tin. Then by the throw of a double-pole switch, the current in the first compartment is stopped and current is passed through the second compartment. In this the action is simple plating; the tin passes into solution and is deposited as tin on the cathode in the second compartment. It is practically 99 per cent pure.

Cuprous Chloride Solutions and Copper Refining.

A paper by Dr. M. DEKAY THOMPSON and Mr. E. R. HAMILTON, of the Massachusetts Institute of Technology, dealt with the conductivity and electrolysis of cuprous chloride solution.

The only advantage that could be expected in refining copper from a cuprous chloride solution is that 96,600 coulombs of electricity would deposit 63.6 grams of copper, assuming 100 per cent efficiency, in place of half this quantity, which is obtained in the acid copper sulphate solution universally employed in copper refining. If the resistance of the cuprous solution were the same as the sulphate solution, and if the former worked otherwise satisfactorily, it is evident that the power now consumed for refining copper could be cut in half.

In order to show that cuprous chloride could be used in place of the sulphate solution, it is first necessary that the conductivity of cuprous chloride solutions should be equally satisfactory.

This salt is only slightly soluble in water, but dissolves in a solution of another chloride. Sodium chloride was the salt used for dissolving the cuprous chloride. Table I shows the solubility of cuprous chloride in sodium chloride.

TABLE I.

Concentration of Sodium Chloride.	Quantity of Cu_2Cl_2 Dissolved. At 90°.	At 40°.	
Saturated solution	16.9 per cent	11.9 per cent	8.9 per cent at 11°
15 per cent solution ...	10.3 "	6.0 "	3.6 " at 14°
5 per cent solution	2.6 "	1.1 "	.....

The conductivity was measured of saturated solutions of cuprous chloride in solutions containing 15, 25 and 30 grams of NaCl in 100 grams water. The conductivities in table 2 are given in reciprocal ohms per centimeter cube.

TABLE 2.

	Saturated Solutions of Cuprous Chloride in Solutions of		
	I	II	III
Temperature.	15 Grams NaCl in 100 Grams Water.	25 Grams NaCl in 100 Grams Water.	30 Grams NaCl in 100 Grams Water.
25°	0.145	0.193	0.200
40°	0.188	0.245	0.258
50°	0.217	0.281	0.298

As compared with the conductivity of pure sodium chloride, the saturation with cuprous chloride lowers the conductivity of the solution. For example, the conductivity at 25° of a solution made by dissolving 25 grams of sodium chloride in 100 grams of water is 0.225 reciprocal ohms, while after saturating it with cuprous chloride it falls to 0.145. This can be counteracted by the addition of a small amount of hydrochloric acid, which at the same time clears up the turbidity. The measurements of table 3 were made of the conductivity of a solution made by dissolving 25 grams of sodium chloride in 100 grams of water, saturating with cuprous chloride and adding the indicated amount of hydrochloric acid of specific gravity 1.188 at 21.7°.

TABLE 3.

Cubic Centimeters of HCl added to 100 c.c. of Solution.	Conductivity.
1.115	0.195
1.595	0.204
2.08	0.206
2.52	0.215
3.26	0.224
3.66	0.230
4.32	0.237
4.81	0.242
5.55	0.251

For the sake of comparison measurements were also made on the conductivity of acid copper sulphate solutions. In electrolytic copper refining the solutions used in practice generally contain between 12 and 20 per cent of crystallized copper sulphate, and between 4 and 10 per cent free sulphuric acid, and are employed between 50° C. and 60° C. Measurements were made of two solutions: "A," containing 3.75 per cent free acid and 12.5 per cent copper sulphate; "B," 9.3 per cent free acid and 18.3 per cent copper sulphate. The specific gravity of A at 22.2° was 1.077 that of B at 21.2° was 1.199. The results of the conductivity measurements are given in table 4.

TABLE 4.

Conductivity of Acid Copper Sulphate Solutions.		
Temperature.	Solution A.	Solution B.
25° C.	0.1573	0.3260
40° C.	0.1752	0.3754
60° C.	0.1865	0.4252

As it thus appears that the conductivity of the cuprous chloride solutions can be made by the addition of free hydrochloric acid as satisfactory as that of copper sulphate solutions, the next point to be investigated was the amp-hour efficiency attainable in the electrolysis of cuprous chloride solutions.

Before studying the effect of impurities a few electrolyses were carried out in a pure solution, such as was used in the conductivity determinations. Two small cells containing a solution of 25 grams of sodium chloride in 100 grams of water saturated with cuprous chloride, and copper electrodes, were connected in series with a copper coulometer. The cuprous chloride cells were covered with a layer of liquid paraffine. In one cell the solution had turned to a deep brown, while in the second it remained nearly white.

On electrolyzing with a current density of 2 amp per square decimeter at room temperature the efficiency in the dark colored cell was 78.2 per cent, and in the other 95.9 per cent, 100 per cent being twice taken as twice the amount of copper deposited in the coulometer. The anodes were eaten away badly at the surface of the liquid. This always took place in neutral solutions where there was no stirring. There was also a corresponding heavier deposit on the upper part of the cathode, as though the liquid conducted better at the top than at the bottom of the cell.

There was a great tendency to form crystalline trees of copper, so that where the current density was not uniform the tree would grow to the anode and short-circuit the cell. Stirring the solution prevented uneven solution of the anode and produced an even deposit on the cathode of a darker color than obtained with a sulphate solution. During the electrolysis the solutions tended to oxidize and form a green scum on the surface. The scum and the turbidity of the solutions can be removed by the addition of a little hydrochloric acid.

In order to see what effect the acid has on the electrolysis, to one portion of a solution of 25 grams of sodium chloride in 100 grams of water saturated with cuprous chloride just enough hydrochloric acid was added to clear the turbidity; to a second portion hydrochloric acid of specific gravity 1.188 was added in the proportion by volume of 4.6 of acid to 100 of solution. These were placed in cells with copper electrodes, and were connected in series with a coulometer and electrolyzed with current density of 0.75 amp per square decimeter. The cuprous chloride solutions were stirred by passing in hydrogen.

The results are given in table 5.

TABLE 5.

Electrolysis of Acid Cuprous Chloride Solutions	Cuprous Chloride.		Copper Sulphate.
	No. 1.	No. 2.	
Voltage at start	2.0	0.18	0.36
Voltage after 2 hours	3.0	0.50	0.46
Voltage after 3 hours	0.84	0.20	0.35
Gain in weight	2.68 grams	3.05 grams	1.66 grams
Current efficiency	80.6 per cent	91.6 per cent	100 per cent

The deposits in cells 1 and 2 were both smooth and adhered firmly. After the electrolysis the solutions stood in contact with the anodes for 48 hours. After 24 hours the first solution had a bright green scum on the surface, and had become turbid, while at the end of 48 hours the second solution had only become a little darker in color. The copper was badly corroded at the surface of the liquid in Cell Number 1, while in Cell Number 2 it was scarcely corroded at all.

The general results of the investigations are summed up as follows: The conductivity of cuprous chloride solution can be made as high as that of the solutions ordinarily used in copper refining, by the addition of hydrochloric acid. At room temperature a current efficiency of 90 per cent of the theoretical can be obtained when the solutions are acidified. The presence of free acid is therefore beneficial in every respect.

Electrolytic Reduction of Nitric Acid.

A third paper "on the electrolytic reduction of nitric acid" was presented by Dr. Harrison E. Patten and Mr. W. J. McCaughey, of the U. S. Department of Agriculture.

The results of their research are summed up as follows:

It is shown again that hydroxylamine may be reduced to ammonia at a copper cathode in presence of copper sulphate, sulphuric acid and hydrochloric acid. Nitrogen is liberated at both anode and cathode.

The rate of this liberation was studied quantitatively. Nitrogen is evolved at the anode at a fairly constant rate, ammonia being left in solution; at the cathode, nitrogen is set free rapidly at first, and in such quantity as to suggest the possibility of the formation of hydrazine as an intermediate step, and its subsequent oxidation by hydroxylamine, thus producing free nitrogen. As electrolysis proceeds, this formation of nitrogen gas decreases.

This initial evolution of free nitrogen in quantity is also met in the electrolytic reduction of nitrate under the same conditions, and suggests that the hydroxylamine formed in the reduction of nitrate may in that case, too, be reduced to a lower state of oxidation, such as hydrazine.

Conductivity of Soils and Corrosion of Steel.

"The Effect of Moisture and of Solutions on the Electric Conductivity of Soils" was the subject of a paper by Dr. R. O. E. Davis, of the Department of Agriculture. This is of importance in connection with the problem of the corrosion of

iron and steel pipes in soil. For if corrosion is an electrochemical phenomenon, the soil furnishes the electrolyte.

From previous work it is known that the condition of the soil has a marked influence on corrosion, and from a study of the soil conditions it has been observed that soil containing much water adds to the corrosive influence. Again, since corrosion is electrolytic, the solution in the soil facilitates corrosive action.

The measurements of soil resistivity show:

1. That in the dry condition, the soil offers a very high resistance to the passage of the current.

2. That at a depth of 2 ft. or more for a given soil and area, the conductivity remains roughly constant.

3. That the conductivity of moist soil increases almost directly as the percentage of moisture increases, the amount of increase depending upon the type of soil.

4. That the conductivity of soils saturated with water increases directly as the amount of salt in solution increases.

5. That below saturation, the resistivity increases almost in proportion to the surface area of the soil; at saturation and beyond, the surface area does not exert so great an influence.

6. That sodium carbonate has an effect of greatly increasing the conductivity.

7. That humus decreases the conductivity of a soil.

8. Sandy soil will probably afford least electrolyte and clay soil most, due mostly to the state of physical division of the soil.

From the data given in the paper it would be possible to calculate roughly the current produced by a given potential difference between two points in the soil, the cross-section of the soil column involved being known. In such a calculation the texture, content of organic matter, water content and saturation point of the soil must be approximately known, as well as the content of soluble salts present in the soil solution. Such calculations, while rough, would be of practical value.

Aluminium-Calcium Alloys.

A paper by Mr. JAMES M. BRECKENRIDGE, of the University of Wisconsin, dealt with "electrochemical potentials and electrical conductivity of aluminium-calcium alloys." It is an interesting example of metallographic methods in physico-chemical and metallurgical research.

Modern research has shown that combinations of metal may take the form of simple mixtures, solid solutions, or chemical compounds. In the two former, the physical and chemical properties of the alloy formed must result from a blending of the characteristic of the original metals. With the compounds, on the other hand, the union of these metals in atomic proportions may result in a substance with properties entirely at variance with those of the original constituents.

Since the alloys of the calcium-aluminium series present physical and chemical properties which were different from the original metals it was thought that this marked change of properties might be due to the presence of one or more chemical compounds, and for this reason a study of the constitution of the alloys of this series was undertaken.

The electrochemical potentials of the aluminium-calcium alloys were measured in a solution of calcium chloride in ethyl alcohol, since aqueous solutions would have reacted on the alloy directly. The potentials were measured against the standard calomel electrode, and a special connection was worked out between the ethyl alcohol solution and the calomel electrode. The electric conductivity of the alloys was simultaneously determined.

The curves of potential and of conductivity when plotted as functions of the calcium content of the alloy are quite smooth and regular with the exception of a distinct break in both curves between 32 and 34 per cent calcium.

The single potential rises suddenly from 0.4810 to 32.8 per cent Ca to 0.9120 at 34.3 Ca, while the electric conductivity curve which was falling all the time with increasing calcium content reaches between 32 and 34 per cent Ca a minimum, rises again slightly to a maximum, and then continues to fall. In the latter curve the break is less evident than in the potential curve.

The composition of the alloy, where the break in the two curves occurs, corresponds to the formula Al_2Ca. This is further in agreement with the results obtained by Donski (*Zeit. f. anorgan. Chemie*, Vol. 57, p. 201) in the study of the cooling curves for this particular series.

Rotating Graphite Anode.

A paper by Dr. J. W. TURRENTINE, of Wesleyan University, describes a rotating graphite anode designed to accompany his graphite cathode dish. The author hopes that the graphite anode may be found capable of supplanting the more expensive platinum anode in electrochemical analysis and other electrochemical processes where the rotating anode is used. The anode is placed above the graphite cathode dish; this offers certain advantages.

The anode used was fashioned from a platinum dish by cutting 10 radial slits in its sides and by soldering, or otherwise fastening, across its top a stout platinum wire, to the center of which was attached, T-wise, a perpendicular platinum rod to serve as the axis of rotation.

The graphite anode is constructed of an especially hard grade of Acheson graphite. It is dish-shaped. It is cut from stick graphite, and is turned down on a lathe to the desired size and thickness. To overcome the porosity of the graphite it is treated with paraffine or ceresine.

Rapid Electro-Analysis.

A first paper, by Prof. FRANCIS C. FRARY and Mr. ANDREW P. PETERSON, of the University of Minnesota, on a "comparison of rapid electro-analytical methods," dealt with the determination of copper. Rapid electro-analytical methods depend on some process of stirring the electrolyte and bringing continually free electrolyte to the cathode; they differ by the special method of stirring employed.

The following three methods were compared: Magnetic rotation of the electrolyte with a ganze cathode (Frary); rotating spiral anode and dish cathode (Smith); simple use of platinum-ganze cathode without rotation (Stoddard), depending on evolved gases and convection currents for the necessary agitation. They used as electrolyte a copper sulphate solution with addition of as small a quantity of nitric gas as possible.

From their determinations, more than 150 in all, the authors conclude that, "in the case of copper, precipitation by the rotating anode is complete and satisfactory in a few minutes less than with magnetic rotation. Precipitation with magnetic rotation is complete and satisfactory in a little shorter time than when no rotation is used. In the last case, however, the character of the precipitate is much more variable, and often for no apparent reason the deposit becomes unweighable. Possibly with the use of somewhat more nitric acid the character of this deposit might have been improved, but the time required would naturally have been longer and the certainty of complete precipitation much less. In all cases the solution should be heated to 70° before electrolysis, the heat evolved by the current maintaining this temperature during the process.

"It is to be noted, however, that the comparatively favorable results obtained in the deposition of copper on the gauze cathode alone do not show that stirring or rotation of the solution may be generally dispensed with. Copper is well known to be one of the easiest metals to deposit under widely varying conditions, and it remains to be seen what results can be obtained with other metals."

Electrodeposition of Platinum.

A paper by Mr. W. J. MCCAUGHEY and Dr. HARRISON E. PATTEN, of the U. S. Department of Agriculture, gives a report of their continued researches on the electrodeposition of plati-

num and on the conditions under which the different forms of cathodic platinum deposits are obtained.

They employed aqueous solutions of potassium chlorplatinate and of ammonium chlorplatinate without or with citric acid. The conditions of producing the different forms of platinum deposit were studied and in addition anode and cathode discharge potentials were taken to assist in the interpretation of the differences observed in the deposits of platinum and the result of citric acid.

The results are given in detail in numerous tables.

Lead Plating and Refining.

A paper by Prof. Frank C. Mathers, of the University of Indiana, describes experiments with the lead perchlorate ($Pb[ClO_4]_2 . 3H_2O$) plating and refining bath. The properties of lead perchlorate, which are of special value in a plating or refining solution, are: Great solubility; smoothness and denseness of cathode deposits and freedom from "trees"; approximately theoretical corrosion of the anode and deposition upon the cathode; absolute stability under all conditions to which it is subjected in a plating or refining bath; no polarization from the formation of lead peroxide on the anode; and very high electrical conductivity.

Sodium perchlorate was used as the starting material for the preparation of the perchloric acid. The solid dry sodium perchlorate was treated with an excess of concentrated hydrochloric acid. After thorough stirring to break up lumps, the mixture was filtered through asbestos and the residue of sodium chloride was washed with successive small portions of concentrated hydrochloric acid. The filtrate, consisting of a mixture of aqueous perchloric acid, hydrochloric acid, and small amounts of sodium perchlorate, was heated on the hot plate until the temperature reached 135°. This completely volatilized the hydrochloric acid with only an insignificant loss of perchloric acid. This aqueous perchloric acid is stable, and is no more dangerous to handle than nitric acid. The lead perchlorate was made by neutralizing this aqueous perchloric acid with litharge.

The electroplating or refining bath should contain about 5 per cent of lead, 2 to 5 per cent of perchloric acid, and 0.05 per cent of peptone. The latter was found the most suitable "addition substance." (Without any addition substance the lead perchlorate bath does not give good lead deposits, except at very low current densities.) A current density of from 2 amp to 3 amp per square decimeter (18 amp to 27 amp per square foot) may be used. The peptone is gradually used up, and after about four days a quantity equal to the original amount should be added. The free acid, which is very slowly neutralized by chemical solution of the lead, must be restored by the treatment of a suitable portion of the solution with the right amount of sulphuric acid. This precipitates lead sulphate and leaves perchloric acid in solution. The filtrate is to be returned to the bath.

The bath gives excellent purification, the cathode being about 99.98 per cent pure. The deposit is smooth, coherent, and has a density of 11.36. As a plating bath, the lead perchlorite solution works exceptionally well on account of the absence of "trees" or loose crystals on the edges even when the deposits reach a thickness of an inch. The bath shows no deterioration with use, and gives as good deposits after two months as at the beginning, if the concentration, acidity, and the required amount of peptone are maintained.

The author sees no reason why the perchlorate bath cannot be applied on a commercial scale, especially for plating of lead, since the deposits are of exceptional smoothness and entirely free from trees or loose crystals on the edges or points.

In opening the discussion Dr. Baekeland said that the use of peptone as a nourishment of an electrolytic bath was a novel idea. Dr. Richards referred to the paper as a striking example of successful research on the use of addition agents. In this field we have not yet reached finality. Without giving further details he referred to a plating process in which the addition of a small amount of a new organic substance not only gives a better deposit, but permits to increase the current density to thus reduce the time.

New Mercury Cell for Alkali Chloride Electrolysis.

A paper by Mr. Jasper Whiting, of Boston, Mass., described his new cell for the production of caustic alkali and chlorine by electrolysis of alkali chlorides. The Whiting cell is a mercury cathode cell.

The author thinks that the mercury type of cell, from the purely theoretical standpoint, is the best thus far conceived, chiefly because it is free from diaphragm troubles and produces in one operation pure caustic alkali. When one considers, however, that the first mercury cell was invented more than 20 years ago, that the patent office is full of patents covering cells of this kind, and that there has been up to the present time only one successful process, the Castner, in commercial operation in this country, it is easy to believe that while the idea is simple, there must be innumerable difficulties in the way of the successful operation of mercury cells.

Moreover, while certain mercury cells seem to work well on a small scale, the results are difficult to duplicate in a plant of commercial size. The cells of Rhodin, Bell, and Baker, are mentioned as examples. Moreover, a well-seasoned apparatus in the hands of men inexperienced in the idiosyncrasies of mercury and sodium amalgam has never been a success. This is instanced by the fact that the Japanese, though possessed of all the published knowledge of the Castner Company, and with all the rights to use the Castner patents, lost $100,000 before the unsuccessful attempt to make bleach and caustic soda by the Castner process in Japan was abandoned. On the other hand, the Castner process is one of the most striking successes of the whole electrochemical field. The company has a very large plant at Niagara Falls and other plants in Europe, and has been paying dividends on its capital for very many years. The fact is that sodium amalgam is a most elusive and contrary material. In the language of a man well qualified to judge, it defies all the laws of nature.

The chief stumbling block in the way of the successful operation of mercury cells is the difficulty of removing the sodium amalgam in a uniform and continuous manner from the decomposing chamber where it is formed to the oxidizing chamber. If this amalgam is not so removed, it tends to accumulate in the decomposing chamber until, when a certain concentration is attained, violent secondary reactions are set up with consequent lowering of the efficiency of the cell. In the ordinary type of apparatus the mercury is caused to flow in a wide and shallow channel from one compartment to another. Hydraulic engineers have long recognized the fact that when a stream of water flows through such a channel the liquid travels much more rapidly in the center than on the sides because the friction of the walls of the channel holds back the material adjacent thereto. If this is a serious difficulty with such a simple fluid as water it is doubly so with a complex fluid such as mercury, and its overlying strata of sodium amalgam. As a result, the sodium amalgam tends to accumulate to an excessive degree with consequent lowering of the current efficiency.

In the Whiting cell a principle, new to this type of apparatus, is employed for the purpose of overcoming the above difficulty. This cell operates intermittently. A body of mercury is placed in the decomposing compartment and held there in a stationary position, as regards the anodes, for a given length of time, during which sodium amalgam of the desired concentration is produced. This amalgam is then completely drawn off from the decomposing compartment, and when removed its place is taken by another body of mercury free from sodium. By placing a number of these compartments in parallel, and operating them successively, the cell, while intermittent in principle, is continuous in its action.

The Whiting cell is a massive concrete structure, supported

on four concrete pedestals, but insulated therefrom. It consists of a shallow receptacle, or box, divided into two compartments, "A" and "B," by a concrete partition. The bottom of the larger or decomposing compartment, "A," is divided by low glass partitions, "C," into a number of sections. The sections have V-shaped bottoms sloping at a slight angle toward central slots "D," these slots being carried through the dividing concrete partitions into the smaller, or oxidizing compartment, "B," where they turn upward and are closed by poppet valves "E," operated by cams, "F," attached to a slowly revolving shaft, "G."

The other ends of these slots are connected one with another by means of a common channel, "H," called the distributing level, and this channel connects in turn with a secondary channel, or run, "I," which leads through one of the side walls of the cell to a pump, "J," at the extrme end of the oxidizing compartment. Mercury covers the bottom of the decomposing compartment, filling to a common level the several sections above described.

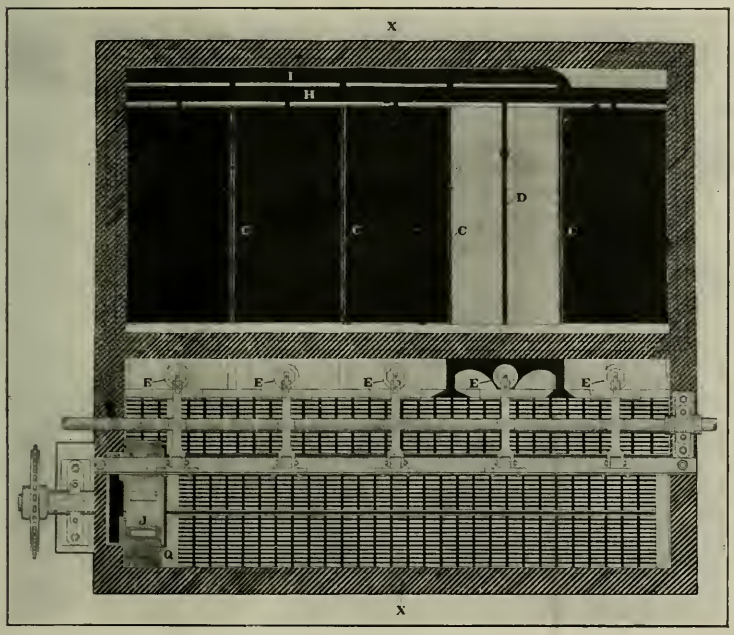


FIG. 1.—WHITING ELECTROLYTIC CELL PLAN.

The anodes, "K," consisting of slabs of Acheson graphite, perforated to permit the free discharge of chlorine, rest upon ledges, "L," placed at the ends of the sections in such a way that the distance between the electrodes shall be a minimum. To these anodes are fastened graphite leads, "M," which extend upward through the cover, "N," connecting with the busbar.

The oxidizing chamber is divided into three channels, "P," lined with graphite of special design, and sloping successively downward to form a zigzag path to the pump pit "Q," where a stoneware rotary pump, "J," also of special design, is placed and operated by auxiliary means. A common body of electrolyte, brine, fills the decomposing chamber, and water or caustic the oxidizing compartment.

The action of the cell is as follows: The floor of the several sections of the decomposing chamber is covered with mercury, which is maintained at a common level by means of the distributing level. The current, supplied to the anodes from the busbars, flows through the brine solution to the mercury and out through the bottom of the cell through iron rods, "R," partly embedded in the concrete and serving to connect the mercury in the cell with the cathode busbar.

The current passing through the electrolyte decomposes the brine, chlorine is given off at the anodes, escaping upward, and sodium is separated at the surface of the mercury, amalgamating therewith. When this action has proceeded for a predetermined period of time, approximately two minutes, the poppet valve at the point of exit of one of the sections is opened by the action of the cam, and the entire mass of sodium amalgam contained in that section sinks rapidly into the slot and down through the connecting pipe into the oxidizing chamber.

The period required for the emptying of a section having been accurately determined, the design of the cam is such that when the mercury is all out of the section the valve closes. Mercury free from sodium then flows into the empty section by way of the distributing level, and continues to so flow until the common level is established. In the meantime, the sodium amalgam in the oxidizing compartment flows by gravity successively over the graphite plates and in so doing it is freed from the sodium it contained, only pure mercury reaching the pump pit, this pure mercury being raised then by the rotary pump into the wall pipe of the decomposing department, through which it flows by gravity to the distributing level. Thus is the cycle completed.

By operating the sections successively, the action of the cell is, in effect, continuous, though the action of each individual section is intermittent.

It is easy to see the advantages of the electrical conditions in the decomposing compartment. A body of mercury is placed in a stationary position as regards the anodes, and at such a distance from them as is theoretically the best; therefore, the distribution of the current is uniform and the resistance of the electrolyte a minimum. Moreover, as there is no agitation of the mercury, during the period of electrolysis, secondary reactions are not facilitated, and as the initial primary conditions are re-established at frequent intervals, uniformity of operation is maintained.

In order to operate the cell satisfactorily it is necessary to feed brine of great strength uniformly between the electrodes. As the electrodes are very close together, and as the area of the cell is considerable, this appeared at first to be a serious difficulty. The problem was solved by utilizing the intermittent action of the mercury in the sections to control the operation of a device for feeding the brine.

Cup-shaped receptacles, "S," equal in number to the sections of the decomposing chamber, are formed in the cover, "N," of the decomposing compartment, these receptacles being connected by a common channel, "T," at their upper extremities. Through the bottom of these cups glass tubes "U" extend downward between the anodes and end of each tube terminating below the surface of the mercury, at a point near the middle of each section.

Brine is fed into one of these cups from an outside source and when the cup is full the brine overflows into the channel, filling each of the cups in turn. As long as the sections are filled with mercury, the bottoms of the tubes are sealed, so that no brine can flow into the cell, but as soon as the mercury is withdrawn from one of the sections the seal is broken on the corresponding tube and a definite quantity of concentrated brine corresponding to the size of the cup flows into that individual section.

As this brine has a greater specific gravity than the partly electrolyzed brine in the cell, it sinks to the bottom and spreads laterally, forming a layer of about 1/16 in. thick over the glass plates. When the purified mercury again fills the section, the new brine, free from chlorine, acts as a protective layer to it, and is in its proper position for being electrolyzed. Moreover, as the brine enters the cell at a time when the section is empty, it causes no agitation of the mercury, and consequently facilitates no secondary reactions.

The action of the oxidizing chamber of the cell could hardly be more simple, and it has the distinct advantage of being automatic. The graphite slabs are so designed as to form a large number of channels through which the mercury flows. The sides of these channels extend upward into the caustic solution, and form with the amalgam a galvanic couple, which serves to decompose the amalgam.

The principle of this action is by no means new, but Mr. Whiting thinks that it has never been successfully employed in an electrolytic cell until now. The difficulty lay in maintaining a proper contact between the amalgam and the graphite. It is easy to secure contact at first, but after the action has proceeded for a varying length of time, the graphite tends to become polarized by the hydrogen given off at its surface, until the necessary contact between it and the amalgam is destroyed.

Mr. Whiting has overcome this trouble by means of a very simple device. Holes about ⅛ in. in diameter and ½ in. deep are bored at frequent intervals in the graphite slabs, and these holes are filled with pure mercury before starting to operate the cell. This pure mercury makes, of course, an effective contact with the graphite walls, of the holes, especially as it is under a relatively high head, and, as it is also in contact with the sodium amalgam which flows above it, it forms a connecting link between the sodium amalgam and the graphite.

Moreover, as sodium amalgam is lighter than mercury, and as the holes are narrow and deep, there is little tendency for the sodium to diffuse downward; consequently the mercury in the holes remains pure, and, as there is no action between it and the graphite, there is no serious polarization of the walls of the holes, and, as a result, the original contact between the mercury and the graphite is not destroyed.

In practice, this galvanic action proceeds at a very vigorous and uniform rate until all the sodium in the amalgam is oxidized, when it automatically stops, pure mercury being delivered to the rotary pump to be returned to the decomposing chamber.

This method of oxidizing amalgam is thought to be superior to that employed in any other cell operating on a commercial scale. It requires no attention, being at all times automatic and is free from mercury losses. Caustic of almost any desired concentration may be produced. It is only necessary to regulate the amount of water fed into the chamber to produce the strength desired. The caustic formed is, as already stated, practically pure.

One of the features of the Whiting cell is its accessibility. This is largely by reason of the construction of the cover of the decomposing compartment. This cover is made of reinforced concrete, with edges extending downward to loosely fit into channels made in the top of the wall of the decomposing compartment, thus forming a gas-tight, liquid seal. The anode leads extend upward through the cover and, while removable, are so arranged that when the cover is raised, the anodes are raised also. To the cover, likewise, are fastened the busbar and the system for feeding the brine, consequently, when the cover is lifted, all the auxiliary apparatus in connection with the decomposing chamber of the cell is removed in one operation, permitting a rapid inspection and cleaning of the cell.

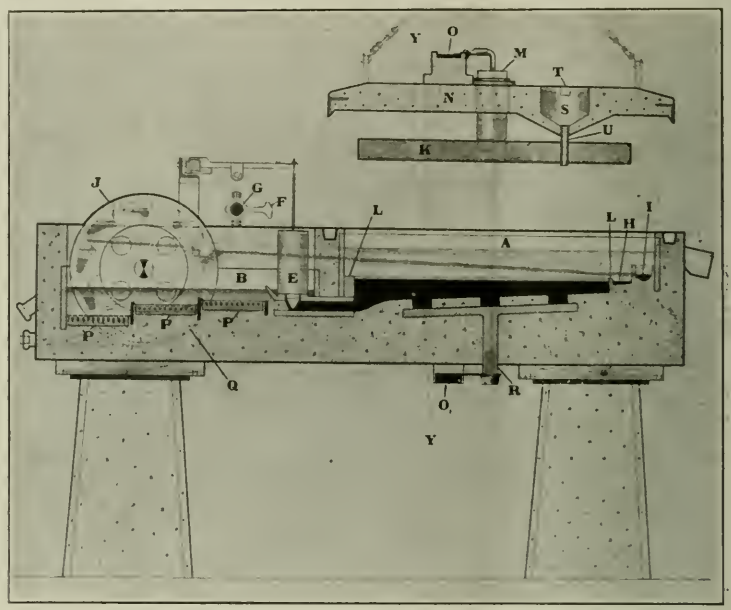


FIG. 2.—WHITING ELECTROLYTIC CELL. CROSS-SECTION.

There is, of course, no fixed limit to the size of the Whiting cell. As each section works independently, one from another, the sections may be multiplied in number as desired, until a cell of capacity suited to the conditions is produced. The cell which has been developed at the plant of the Oxford Paper Company, at Rumford, Maine, is about 6 ft. square over all. It contains five sections, and is operated with a current of from 1,200 amp to 1,400 amp, equivalent to a current density of over 100 amp per square foot of active anode surface. The voltage across the terminals is approximately 4, and the cells operate at a current efficiency of from 90 per cent to 95 per cent.

Mr. Whiting thinks his cell is the only one which can be shut down without noticeably affecting its efficiency or without injury to the apparatus. To illustrate this in a practical way, during a test run of four cells the current was shut off for various reasons 11 different times for periods varying from a few minutes to 12 hours, but in spite of these many shut-downs the plant maintained an average current efficiency of over 95 per cent.

In case of shut-down it is only necessary to stop the operation of the pump, when all the mercury is automatically drawn off from the decomposing to the oxidizing compartment, where it remains until it is again desired to start the cell, which is accomplished by the starting of the pump. As the mercury is all out of the decomposing compartment while the current is off, there is no loss of mercury due to the corrosive action of the chlorinated brine, as is the case with other types of mercury cells.

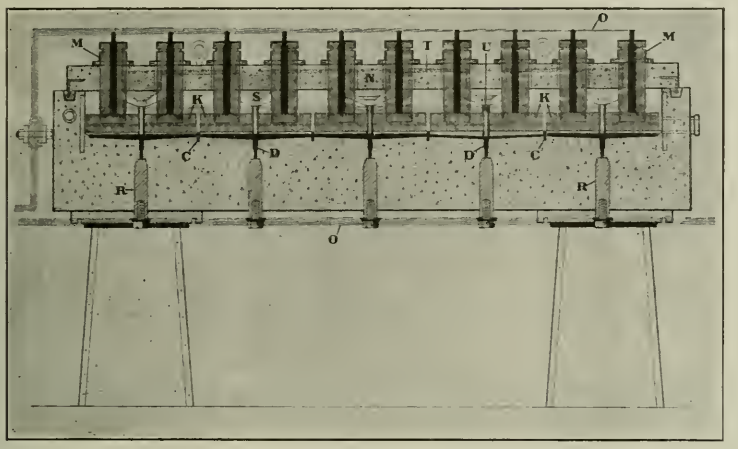


FIG. 3.—WHITING ELECTROLYTIC CELL. LONGITUDINAL SECTION.

Moreover, the Whiting cell is singularly free from injury on account of accidents. Should there be an unexpected stoppage of the pump, the mercury leaves the decomposing chamber as above described. Should one of the valves leak, the level of the mercury in the decomposing chamber will be slightly affected, with a corresponding increase in voltage, but no serious damage will be done. Should one of the valves fail to operate sodium would accumulate in the mercury of that section, but the remainder of the cell would operate satisfactorily.

Occasionally the cell has to be washed, to free it from precipitated graphite dust and from slime which forms on the glass plates. The operation of washing a cell is, however, an extremely simple one and requires the labor of but one man for about half an hour.

As the cell is operated at a temperature below 40° C., the tendency to form chlorates is very small and, consequently, the deterioration of the graphite anodes is reduced to a minimum. There is, of course, no depreciation of the graphite slabs in the oxidizing chamber of the cell.

As before stated, the mercury type of cell produces pure caustic. Consequently the cell efficiency, as far as the caustic is concerned, is the plant efficiency. There are no losses incident to subsequent treatment. What flows out of the cell is the finished product, assuming, of course, that liquid caustic is sold. The chlorine gas escaping from the cell is 98 per cent pure, the remaining 2 per cent being hydrogen, and as the chlorine leaves the cell at a relatively low temperature, it is unnecessary to cool it further before using it to form bleaching liquor.

The Whiting electrolytic cell is not an expensive cell to install. It is generally believed that mercury cells are very costly, for the reason that large quantities of mercury are required in their operation, and mercury is, of course, a high-priced material. When one considers, however, that the use of mercury eliminates the necessity of installing very expensive and rapidly deteriorating multiple-effect evaporating machinery and auxiliary apparatus for the purification of the caustic produced in diaphragm cells, it is easy to see that one expense largely offsets the other. Moreover, mercury is a quick asset convertible into cash at a high proportion of its first cost.

The present plant, erected for the Oxford Paper Company, has a capacity in bleach-liquor equivalent to about 5 tons of bleaching powder per day, and produces, in addition, about 2 tons of pure caustic soda. This plant has been operating since early in January, and the results obtained from it have been so satisfactory that its capacity is now being increased five times.

In the discussion which followed Prof. C. F. Burgess pointed out the great difference which exists between laboratory apparatus and commercial apparatus. In reply to several questions by Dr. Baekeland, Mr. Whiting stated that the loss of the mercury has been found to be less than 6 per cent per year, but the estimates of the cost had been based on an assumption of 10 per cent loss of mercury per year. The strength of the cathode liquid is usually 20 per cent, but caustic has also been made 40 per cent strong. There is no further concentration then necessary. The cell is washed about once a month. In reply to a question of Dr. Richards as to possible evaporation of mercury and its effect on the workingmen, Mr. Whiting said that the mercury could not evaporate, since it was always covered.

FIG. 4.—WHITING ELECTROLYTIC CELL. INSTALLATION OF OXFORD PAPER COMPANY.

With respect to the formation of hypochlorites and chlorates in chloride electrolytic cells, Dr. Baekeland said that when the chlorates and hypochlorites begin to participate in the electrolytic action they are decomposed and oxygen is evolved or oxygen compounds, including sulphates, are formed at the anode. This causes a destruction of the anodes, even of graphite anodes, which are practically indestructible by chlorine, but are corroded in sulphate. This is not only a serious item of expense, but the internal resistance of the cell is increased and the whole operation is disturbed. Treatment of the anode with paraffine would help nothing, since paraffine absorbs chlorine. Cast-iron oxide anodes are used in Germany to a certain extent, but they also involve some disadvantages. Dr. Baekeland finally pointed out that the proper design of the contacts was a very important problem for an electrolytic cell.

Townsend Cell.

A paper by Messrs. H. K. Richardson and R. L. Patterson, of Cornell University, gave an account of experiments made with an experimental Townsend cell. The object was to determine the effect of current density (varied from 18 amp to 500 amp per square foot) on efficiency and output. The watt-hour efficiency and amp-hour efficiency drop, of course, with increasing current density. The numerical results were given in a table. The authors also determined the effect of variation of the flow of anode liquor on the results.

In the discussion Mr. Hooker, of the Hooker Electrochemical Company, pointed out that the commercial Townsend cell, as operated by this company, is giving far more favorable results, the difference of the numerical results being due to the use of dimensions in the authors' experimental cell, quite different from those used in the commercial cell.

Dr. Backeland remarked incidentally that running a Townsend cell at a high current density is advantageous on account of the scrubbing action of the hydrogen gas.

Electrolytic Winning and Refining of Metals.

A paper by Mr. Edward M. Chance, of Pottsville, Pa., describes "a new method for the electrolytic winning and refining of metals." He suggests a process of treating mattes and speisses by electrolysis in the fused state.

Robert H. Aiken has patented such a process (U. S. patent 873,648, Dec. 10, 1907). Mr. Aiken's method consists in fusing a sulphide ore containing nickel, copper, and iron, such as pyrrhotite, or the matte from such an ore, and directly electrolyzing the same. He claims that the copper and nickel may be readily separated as metals, from the fused sulphide, while the iron sulphide will remain undecomposed if the difference in potential between the poles be properly controlled. If, after the removal of copper and nickel, the voltage be increased, the iron will be deposited. In both cases sulphur is liberated at the anode.

Mr. Chance comments on this as follows: "It would seem that insuperable obstacles would lie in the way of the commercial application of such a process. First of all I believe that there would be considerable loss of electrical energy, as I understand that such a bath would conduct current directly, without the current doing electrolytic work. The separation of the metals would not be sharp. Mr Aiken himself acknowledges that the copper nickel deposit is largely contaminated with iron, which he suggests may be removed by subsequent refining It seems reasonable, also, that metals so deposited would retain considerable sulphur."

Mr Chance proposes a process in which a fused electrolyte is superimposed upon the fused matte. The most troublesome elements, arsenic, antimony, and sulphur, would volatilize at the temperature required, thus keeping the electrolyte clean, and ensuring uncontaminated deposits on the cathode.

The author made experiments on a small scale in a crucible filled with the fused matte, which was electrically connected to the outside circuit by means of an iron wire in the bottom of the crucible. This formed the anode. The electrolyte was then added and a carbon or steel rod, forming the upper electrode (cathode) and dipping into the electrolyte, was inserted through a hole in the top. This cell was inserted in a gas furnace. Current was obtained from a battery of four cells.

Experiments were made with the electrolysis of copper-iron mattes of about the following composition: Copper, 50 per cent; iron, 24 per cent; sulphur, 26 per cent. The electrolyte found most efficacious was a sodium metasilicate. It was found that by controlling the difference in potential across the poles, a very pure copper could be deposited on the cathode, iron remaining in the electrolyte as silicate. When the e.m.f. was increased, both iron and copper were deposited apparently as alloy, on the cathode. The sulphur appeared to volatilize entirely at the anode, none remaining in the electrolyte or upon the cathode.

With a nickel-copper-iron matte or speiss, the copper could be deposited selectively, and when all the copper had been removed from the electrolyte, the nickel could be removed free from iron, by increasing the e.m.f. It is understood, of course, that the voltage must be kept below the point at which iron silicate is decomposed. In the case of speisses the arsenic and antimony appeared to volatilize quantitatively.

Attempts were made to use halogen salts in place of silicate in the treatment of mattes. It was found, however, that at the temperatures used, chlorides of the alkalies and alkaline earths were both volatile and unstable, while calcium fluoride was not sufficiently fluid.

Chloride of sodium, however, was found very useful in the electrolysis of alloys with low fusion points, such as silver-lead.

The author mentions that by this process a very sharp separation of silver from lead in a silver-lead bullion was made. This alloy was attacked with a sodium lead silicate electrolyte, the lead being carried over to the cathode, while the silver remained unattacked at the anode.

It is stated that the metals deposited on the cathode by these processes were marked by great adherence as well as purity. Copper when plated upon the 9 mm steel-rod cathode could be removed only by filing. It was possible to hammer, bend and otherwise deform the steel, without loosening the copper coating.

In the discussion Dr. Richards remarked that he had repeated in his laboratory Mr. Chance's experiments and found the method to work, but he was not sure whether the deposition of the copper was really electrolytic.

A motion of thanks to Pittsburgh for its splendid hospitality was then unanimously passed.

Public Meeting.

In the afternoon an open meeting was held in the Carnegie Music Hall, to which the general public had been invited. It was well attended. Mr. Charles F. Scott presided. He said that Pittsburgh's pride in the past had been tonnage, but times are changing, and Pittsburgh which has profited so much by science in the past will profit still more by science in the future, with respect to quality and efficiency as well as to tonnage

Electrochemical Industries.

Dr. Joseph W. Richards delivered an address on "the present status of the electrochemical industries." If all the electrochemical industries were put together on one spot, they would equal the industries of Pittsburgh. But they are scattered all over the world. At present there are only two electric steel refining furnaces in regular operation in the Pittsburgh district, but in other places of the world there are almost 100. But when Pittsburgh takes up a thing it is done with elementary energy. Dr. Richards predicted a great future for the electric steel furnace in Pittsburgh.

A concise sketch was then given of the various classes of electrochemical industries—electrolytic processes, electric furnace processes, and processes employing the electric arc and electric discharges in general. Dr Richards then showed in a

long series of lantern slides typical installations of such processes, to carry home his main point, that electrochemical processes do not simply exist on paper, but that electrochemical industries are a reality, an accomplished fact of considerable commercial importance to-day. But the future is even brighter.

There is now a demand and a necessity for cheaper power, and the developments along this line change the commercial geography of the world.

Conservation of Natural Resources.

Mr. JOHN H. FINNEY, secretary of the Southern Appalachian Conservation Commission, then delivered an address on "the conservation and utilization of natural resources of power."

The conservation policy got its mighty impetus in the Governor's meeting called by Roosevelt to the White House; it was the first gathering of its kind in the world.

Conservation is a moral issue. The speaker paid a tribute to Mr. Pinchot. The result of the propaganda has been an aroused national conscience.

Mr. Finney then classified our resources as exhaustible and inexhaustible. Exhaustible are coal, natural gas, oil, peat, wood. Inexhaustible are manual labor, animal labor, wind power, sun power, water-power. All of these resources were discussed with reference to the waste now going on. With respect to coal, the speaker dealt especially on by-product coke ovens and the utilization of waste gases.

As to water-power the speaker said that not every water-power is a gold mine, yet the question of regulating the status of water-powers in this country is of immense industrial importance. The question of ownership and rights of water-power in this country is exceedingly complicated. We have individual ownership, State ownership and joint ownership. It is a complex and serious problem.

Complete individual independence is only possible where man is isolated. Otherwise regulation by the State becomes necessary. Development of a water-power is no longer to be considered as a private investment of an individual. Since electric power transmission has become a reality, a water-power is now a public utility.

Mr. Finney gave a review of water-power legislation in Europe, especially in Switzerland, Italy, and France. He outlined the fundamental principles of desirable legislation for this country and urged that the question should be fearlessly taken up, or what are still free and equal and open opportunities will be closed forever.

Concluding Speech.

The final speech of the meeting was made by Dr. JOHN A. BRASHEAR, the astronomer, who had been introduced by Mr. Scott as one of "Pittsburgh's civic assets."

Dr. Brashear gave in a delightful and humorous speech some of "the beautiful reminiscences of a lifetime," of his coming in contact in years past with many eminent scientists and engineers, and especially reminiscences of the early work of Dr. E. G. Acheson, who began his brilliant career on a small scale in Pittsburgh. "Don't believe he started to do it for the money that might be in it. He did it because it was in him. He had to do it."

The meeting was finally closed with a few felicitous words by the retiring president, Dr. L. H. Baekeland.

Social Functions and Excursions.

As stated before, the society was royally entertained. On Wednesday evening there was an informal reception in the English room of the Fort Pitt Hotel by the Pittsburgh local committee and the officers of the society to visiting members, guests and ladies. On Friday evening a subscription dinner was held at the Fort Pitt Hotel, Mr. C. F. Scott acting as toastmaster.

A special program had been arranged for the entertainment of ladies. Various excursions, among them one to the Heinz factory, were made on Thursday afternoon, while a theater party had been arranged for the evening. On Friday a visit was paid to the Margaret Morrison School of the Carnegie Technical Schools, followed by lunch at the Hotel Schenley, an automobile trip through Pittsburgh's parks and boulevards in the afternoon, and the dinner in the evening.

The most delightful feature of the whole convention was, however, the eminently successful excursion program. This had been arranged with utmost care for the minutest details. Pittsburgh's hospitality could indeed not be better shown than by the wide-open doors of its industrial establishments, and the utmost efforts were made to keep every visitor fully informed on all the features of the plants visited, which were specially worth seeing.

In this respect the handbook prepared by the local committee and containing concise, illustrated descriptions of the engineering features of the various plants, was a novelty so successful that we would wish that similar handbooks might be prepared for all future meetings, if we did not know that this would be practically impossible. The preparation of the Pittsburgh handbook must have required an enormous amount of work and the highest credit is due to the editors. The book is, moreover, exceedingly well printed and neatly illustrated. One of the illustrations of the handbook is herewith reproduced, that is, the view of the crucible furnace of the Crucible Steel Company of America—the largest in the world. For the photograph from which the cut was made we are obliged to Mr. P. Kemery, of the Crucible Steel Company of America.

A 60-POT CRUCIBLE FURNACE. THE LARGEST IN THE WORLD. CRUCIBLE STEEL COMPANY OF AMERICA.

The following gentlemen were in charge of the various excursions. Wednesday excursion: Messrs P. Kemery and W. O. Snelling. Thursday excursion: Mr. H. Rodman, general chairman; Mr. Bruce Warner, Isabella Furnaces; Mr. George S. Page, Crucible Steel Co.; Mr. H. H. Laughlin, Jones & Laughlin Steel Co.; Prof. F. Crabtree, Pennsylvania Smelting & Refining Co.; Mr. William Kinney, Universal Portland Cement Co.; Mr. H. K. Hitchcock, Brazing, and Mr. Max Harris, Nernst Lamp Co.

Friday excursion: Mr. P. Kemery, general chairman; Mr. G. S. Page, train arrangements; Mr. H. K. Hitchcock, at Allegheny Plate Glass Co.; Mr. H. Rodman, at Westinghouse Machine Co.; M. B. Wiley, at Westinghouse Electric & Mfg. Co.; Mr. E. B. Clark, at Firth-Stirling Steel Co.; Mr. L. T. Watson at Homestead Steel works.

U. S. Geological Survey Testing Station.

On Wednesday afternoon the Testing Station of the Technological Branch of the U. S. Geological Survey was visited. A number of very interesting and instructive tests had been specially arranged for this occasion.

Two tests were made to determine the effect of a charge of an explosion when fired into 120 pounds of bituminous

coal dust from the Pittsburgh seam, placed on horse and on side shelves, in a dust gallery. In another test the unit disruptive charge of blasting powder, tamped with two pounds of dry fire clay, was fired at a ballistic pendulum to illustrate the method of determining the unit disruptive charge of an explosion.

Two other demonstrations had to do with explosions due to lamps. In one test the effect of an explosive air and gas mixture on various "safety lamps" was shown with different velocities of the gas. In the other demonstration the globe of an incandescent lamp was broken mechanically in an explosible atmosphere to see under which conditions an explosion would ensue. The comparative thickness of the filament was shown to be very important in this respect. When the globe of a lamp with a comparatively thick carbon filament is broken, usually an explosion results; when the globe of a lamp with a frail tungsten filament breaks, an explosion is very unlikely. Different types of mine rescue apparatus were also demonstrated.

Besides the investigation of prevention of mine accidents the Technological Branch of the U. S. Geological Survey has two other objects; the testing of structural materials (especially at present in connection with the Panama Canal and fuels.

The testing of firebrick under load conditions at 1300° C. was demonstrated and test kilns shown in operation. A test of a cable was also made on a 600,000-pound machine.

Thursday Excursion.

For Thursday afternoon six different excursions had been arranged, all of which were well attended.

The *Isabella furnaces* of the *Carnegie Steel Co.* are of special interest, as this is the plant in which the dry-blast process of James Gayley was first applied. The estimated saving by the use of dry blast is 360 pounds of coke per ton of iron, an increase of tonnage of 11½ per cent.

The *Park Works* of the *Crucible Steel Co. of America* is the largest individual works belonging to the Crucible Steel Company, and, when in full operation, employs about 2500 men. Its average production of finished steel amounts to 140,000 to 150,000 tons per year.

Natural gas is used for fuel in melting both crucible and open-hearth steel and also largely for re-heating purposes, while coal is used for balance of the re-heating and for steam power. Natural gas is piped from West Virginia. Coal is received at the works by both rail and river, about one-half the supply coming each way.

Raw materials so far as possible are received in yards served by overhead electric traveling cranes where electromagnets are used when convenient for handling.

The crucible melting department consists of 12 Siemens regenerative furnaces ranging from 24 pots to 48 pots each.

EXCURSION OF AMERICAN ELECTROCHEMICAL SOCIETY TO THE

The maximum melting capacity in crucible steel ingots is about 3,000 pots of 100 pounds each in 24 hours, or about 150 net tons. In tonnage output, this crucible melting department is the largest in America and is only rivaled by that of Krupp abroad.

The open-hearth department consists of 8 furnaces, 3 "acid" and 5 "basic," ranging in capacity from 18 to 50 tons each. In this department effort has been directed toward the production of the finer grades of open-hearth steel and to specialties, rather than toward large tonnage. About 30 steam hammers and 20 trains of rolls are used in the production of the finished product. The company manufactures all their graphite crucibles from graphite brought from the island of Ceylon and from clay brought from Germany. At this works are made all the celebrated brands of "black diamond" tool steel, some of which have been on the market for nearly fifty years. The Park "extra," "special" and "silver" brands are also made here. Among the high-speed steels the "rex A" and "rex AA" brands are produced at this works. Protective shields for field guns for the United States Army, which require plates 15-inch thick to stand impact of cupro-nickel jacketed bullets from an army rifle with 2,140 ft. per second muzzle-velocity at range of 50 yards, without the plate showing any signs of crack, and only a limited amount of "bulge," are also made at this works. Alloy steels in which tungsten, chromium, nickel, vanadium, silicon, manganese, titanium and other metals are used in varying combinations and proportions, are produced extensively at this plant, as are also the composite steels made up of layers of hard and soft steel, such as 5-ply safe steel, jail bar, soft center and hard center.

The *Jones & Laughlin Steel Co.* is the largest independent steel company outside of the U. S. Steel Corporation. The works are metallurgically of special interest on account of the nine 200-ton Talbot continuous furnaces, which the South Side open-hearth plant contains, besides seven stationary furnaces of approximately 50 tons capacity each.

The Talbot furnaces are called continuous because they are never entirely empty during a run. They may be termed basic-lined tilting continuous furnaces.

When starting the furnaces the first of the week, the charge consists of cold pig iron of low silicon content, steel scrap, burnt lime or limestone; the same as used in a stationary furnace. When the bath is in the liquid state, molten pig iron of low silicon content is taken from the mixer and added to the bath, an oxidizing slag being made by the addition of roll scale before the above metal addition. These different operations of adding roll scale, lime and metal are continuous until the bath contains 200 tons or more, when the slag and bath are made in condition for tapping; after which 60 to 80 tons are poured off into a ladle where the proper deoxidizing alloys are added and the steel is teemed into moulds.

The important part of the operation is the addition of roll scale and lime, as all the metalloids in the iron are to be oxidized chiefly by the oxygen brought in by the addition of roll scale. Experience has shown that the required amount of scale to add is about 21 to 24 per cent of the metal charged; and the lime from 6 to 8 per cent, according to the composition of the lime and scale.

After the scale has been charged in the furnace for about one hour, 30 to 60 tons of mixer-iron is poured through a trough into the furnace. The reaction takes place immediately and is somewhat like a Bessemer blow during the elimination of carbon, inasmuch as a large volume of carbonic oxide gas is driven off. When the iron oxide of the slag reacts upon the bath containing silicon, phosphorus and manganese, they are in turn oxidized, to silica, phosphoric acid, and manganese oxide, which enter the slag. The carbon is oxidized to carbonic oxide.

The *Pennsylvania Smelting Co.* at Carnegie, Pa., contains two lead blast furnaces for smelting the ore, reverberatory matte roasting furnaces, reverberatory softening (refining) furnaces and desilvering kettles for separating silver and gold form the lead bullion; zinc distillation furnaces for separating zinc from zinc-silver crusts, and cupelling furnaces for refining the silver and gold; also, beehive ovens for making coke required by the blast furnaces.

A considerable variety of ores and of products containing lead and silver are smelted here—ores from the Cobalt district in Canada, the Coeur D'Alene district in Idaho, the Middle West and other districts; and among the products are high-

HOMESTEAD WORKS OF THE CARNEGIE STEEL COMPANY.

grade litharge, refining lead, antimonial (hard) lead, silver, matte and speiss.

At the *Universal Portland Cement Co.* at Universal, Pa., Portland cement is made from limestone and granulated blast furnace slag, the latter being the product of the Bessemer furnaces of the Carnegie Steel Company. Large bins approached by elevated tracks at one end of the raw material mill provide the storage room for the raw material. From these bins the limestone is fed directly into gyratory crushers which prepare it for the dryers into which the stone then enters. The slag is also fed into separate dryers, all of which are maintained at a sufficiently high temperature to drive off all the moisture. After passing through the dryers each of the raw materials is conveyed into separate Ball and Williams mills, where rolling and impact reduces them to extreme fineness. The two materials are then carried to separate hoppers from which they are drawn off and properly proportioned by electrically operated weighing devices. The mixture of slag and limestone, already a powder, is then passed into tube mills where flint pebbles rolling in a steel drum thoroughly mix it and complete the process of pulverization.

The next step is the calcination of this raw mixture in rotary kilns. Leaving the rotary kilns the clinker, produced by the burning, is dropped into storage pits where it is cooled and cured. Electrically-operated cranes then carry the clinker to the finishing mill, where it is passed through jaw crushers to Kent mills and then over Newyago separators. After the addition of a certain quantity of gypsum, which is done by means of an automatic device, the material again passes through tube mills similar to those used in finishing the raw

material. Being now ground to a specific degree of fineness, the cement passes on to a belt conveyor and is distributed into the various bins of the storage house. The cement in passing from the finishing mill into the storage house is automatically sampled, from which samples the various tests for quality are made in the laboratory.

At the works of the *Nernst Lamp Co.* the processes of manufacture of the glower and the heater were shown.

The *Pittsburgh Reinforced Brazing & Machine Co.* was also visited. The methods of repairs of this company were described in an article on page 300 of our last issue.

Friday Excursion.

For Friday a delightful all-day excursion had been arranged, by special train.

The *Allegheny Plate Glass Co.'s* works at Glassmere were first visited. The various successive operations of casting, cooling, trimming, grinding, and finishing were most interesting. The output of this modern, up-to-date plant is 4,000,000 square feet of polished plate glass per year.

The immense works of the *Westinghouse Electric & Mfg. Co.* and of the *Westinghouse Machine Co.* were next visited. Of special interest were the large electric locomotives for the New Jersey-New York-Long Island tunnels of the Pennsylvania Railroad. For the repair of castings the Bernardo system of electric arc welding is employed on account of the simplicity of the apparatus; it is stated to give very good results and to save considerable money. At the works of the Westinghouse Machine Co. the Melville-Macalpine reduction gear, with floating frame for equalization of two pressures was of special interest. It permits reductions up to a ratio of 12 to 1 and capacity as high as 25,000 hp. It interlinks turbine of most efficient speed with propeller of most efficient speed for marine work. It is also especially applicable to direct-current work in connecting a turbine which is of inherently high speed with a generator that must be operated at low speed.

The *Firth-Stirling Steel Co.*, at McKeesport, was then visited. Only fine crucible steel, for tool purposes, is made at the McKeesport plant. They have installed a large number of crucible furnaces, one 2½-ton Heroult furnace and one resistance furnace. The remaining equipment is the same as found in any crucible steel works, and consists of a number of steam-driven rolls, steam hammers, and electric-driven shears, etc. Power for the motors is obtained from 1-250-kw and 1-150-kw, d.-c. generators, belted to steam engine units. The boiler house contains a bank of six boilers, aggregating some 1800 hp. Of special interest is the 2½-ton Heroult electric arc furnace. In its general design it is just like the 15-ton Heroult furnace, at South Chicago, described in our April issue, page 179, but smaller and operated by single-phase instead of three-phase. It is operated with cold charge. Three heats can be refined, working from cold scrap for 24 hours, and probably 15 heats when working with hot material from the open-hearth furnace.

There is also a simple 120-kw resistance furnace for ferro-alloy manufacture at these works.

The famous and magnificent *Homestead Works of the Carnegie Steel Co.* was the last plant visited. They are so well known that no description is necessary.

It should be mentioned, however, that an electric furnace is now running at these works of 25 tons capacity Its feature is that it can be used either as open-hearth or electric furnace.

It is first used as an open-hearth and when the heat is finished the gas supply is closed, the electrodes are lowered and the electric refining process begins. The advantage of this arrangement is that the transfer of molten metal from an open hearth to an electric furnace is avoided. The disadvantage is the complication of the roof construction.

The party then returned to Pittsburgh

In the following we give a complete alphabetical list of the names of members and guests in the last printed registry list. Where no city is stated, Pittsburgh is meant:

G. P. Adamson, Easton, Pa.; I. C. Allen; R. Amberg, Chicago, Ill.; R. Anderson, Philadelphia, Pa.; W. C. Andrews, New York; R. B. Anthony; J. C. Armor Ingram, Pa.; Mr. and Mrs. W. C. Arsem, Schenectady, N. Y.; J. Aston, Madison, Wis.; R. W. Atkinson; G. H. Auperle, Avonmore, Pa.; J. A. Aupperle, Middletown, Ohio; J. E. Babb; H. Bacharach; A. E. Bachert, Tyrone, Pa.; Dr. and Mrs. Leo H. Baekeland, Yonkers, N. Y.; Charles Bailey; F. D. Barbour; H. Barbour; W. E. Barnes, Creighton, Pa.; A. M. Barrett; A. M. Barron, Cleveland, Ohio; C. Bauer; J. W. Beckman, Niagara Falls, Ont., Canada; J. Bechtel, Tarentum, Pa.; A. W. Belden; H. Beltz, Perry, N. Y.; J. C. Bennie, New York City; P. McN. Bennie, Niagara Falls, N. Y.; G. H. Blaxter, Beaver Falls, Pa.; A. D. Blemenger; E. Blough; P. A. Boeck, Worcester, Mass.; G. Boericke, Primos, Pa.; J. J. Boyle, Ambridge, Pa.; Wm. Brady, Chicago, Ill.; J. D. Brenemau; A. Brill; W. H. Bristol, Waterbury, Conn.; Mr. and Mrs. R. E. Broadwell, Cleveland, Ohio; Mr. and Mrs. W. B. Brookfield, Syracuse, N. Y.; G. H. Brown; J. T. Brown; J. W. Brown, Cleveland, Ohio; Jos. Brown, R. C. Brown, Greensburg, Pa.; Mr. and Mrs. C. L. Bryden, Scranton, Pa.; N. M. Buch, West Pittsburgh, Pa.; R. V. Buigay; H. R. Bulmer, Alliance, Ohio; C. F. Burgess, Madison, Wis.; W. R. Burrows, Harrison, N. J.; J. R. Cain, Washington, D. C.; A. R. Calder, Steelton, Pa.; J. D. Callery; C. B. Callomon; H. R. Carveth, Niagara Falls, N. Y.; C. R. Cary, Philadelphia, Pa.; A. Casselberry, Altoona, Pa.; Adolph Challas, London, England; G. D. Chamberlain; W. P. Champney, jr., Cleveland, Ohio; D. H. Childs, Alfred, N. Y.; R. B. Chillas, jr., Cleveland, Ohio; G. B. Chorpening, Clarksburg, W. Va.; E. C. Chrisown; C. C. Cito, Newark, N. J.; Mr. and Mrs. E. B. Clarke, McKeesport, Pa.; J. K. Clement; Mr. and Mrs. W. R. Clymer, Cleveland, Ohio; W. C. Coffin; W. P. Cohoe, Toronto, Canada; E. A. Colby, Newark, N. J.; F. Collins; Wm. F. Collman; A. L. Colton; H. R. Connell; Sidney Cornell, Duquesne, Pa.; Mr. and Mrs. J. S. Coye, Boston, Mass.; Prof. and Mrs. F. Crahtree; H. H. Craver; F. S. Crawford; J. S. Crider, Cleveland, Ohio; F. B. Crosby, Oakmont, Pa.; W. B. Crowe; A. W. Crowhover, Greensburg, Pa.; C. W. Danforth, Sharon, Pa.; G. H. Danforth; H. E. Davis; W. B. Davis; S. A. Davis; S. Davis, Jr.; A. H. Dean; B. Dewey, Vandergrift, Pa.; Colby Dill, Perth Amboy, N. J.; H. E. Diller, Schenectady, N. Y.; W. A. Donkin; C. A. Doremus, New York; J. W. Dougherty, Steelton, Pa.; H. Dripps; H. DuBois, New York; S. N. Dunbar; L. H. Duschak, Corning, N. Y.; Sam Dushman, San Vito, Switzerland; J. J. Eckfeldt, Latrobe, Pa.; D. Edge, Swissvale, Pa.; H. R. Edvecomb; A. D. Edwards; H. F. Engelbrecht; H. W. Erhing; F. F. Farnham, McKeesport, Pa.; G. Fesserkes; J. J. Flannery; G. H. Follows; C. S. Foster, Rochester, N. Y.; J. L. Foster; M. W. Franklin, Schenectady, N. Y.; J. W. C. Fraser; E. Friedlaender, Braddock, Pa.; H. Frodsham, Schenectady, N. Y.; H. A. Field; A. C. Fieldner; C. G. Fink, Schenectady, N. Y.; C. G. Fisher; Mr. and Mrs. H. W. Fisher; H. L. Fullerton; H. A. Gardner, Philadelphia, Pa.; Mrs. Gass; André Gerard, Liége, Belgium; J. A. Gerhart; C. L. Gerwig; W. H. Gibb; A. E. Gibbs, Wyandotte, Mich.; Miss Gibbs, Wyandotte, Mich.; Mr. and Mrs. W. T. Gibbs, Buckingham, Quebec; C. B. Gibson, Wilkinsburg, Pa.; H. W. Gillett, Ithaca, N. Y.; J. S. Goldhaum, Philadelphia, Pa.; S. L. Goodale; G. M. Goodspeed, McKeesport, Pa.; J. H. Goodwin, Fremont, Ohio; S. P. Grace; C. M. Graves, Fostoria, Ohio; J. M. Graves; W. G. Graves, Cleveland, Ohio; A. E. Greene, Chicago, Ill.; H. J. Grove, Homestead, Pa.; R. S. Guerber; H. R. Hadfield; H. Halcomb; J. Hall, Tarentum, Pa.; H. Hambuechen, Madison, Wis.; J. O. Handy; C. A. Hansen, Schenectady, N. Y.; Mr. and Mrs. R. H. Hartley; N. Hecht; H. W. Heinz; C. T. Henderson, Milwaukee, Wis.; E. Hensen, McKeesport, Pa.; Carl Hering, Philadelphia, Pa.; Dr. P. L. T. Heroult, La Praz, France; P. H. Heroult, Jr., New York; B. F. Hessom, Jr., Aspinwall, Pa.; H. D. Hibbard, Plainfield, N. J.; E. F. Hicks, Philadelphia, Pa.; H. O. Hill, Allegheny, Pa.; H. K. Hitchcock, Tarentum, Pa.; Mrs. E. P. Hitchcock, Tarentum, Pa.; C. R. Hodgdon; M. Hokanson; A. H. Hooker, Niagara Falls, N. Y.; H. Howard, Brookline, Mass.; L. E. Howard, Chicago, Ill.; S. P. Howell; W. P. Hurley, Wilkinsburg, Pa.; R. H. Hursh; E. J. Hutchinson, Cincinnati, Ohio; W. G. Ireland; A. S. Isaacs; H. Isenberg; K. Iwai, Tokyo, Japan; J. H. James; R. James, Munhall, Pa.; W. P. Jend, Mr. and Mrs. C. F. Jenkins, Washington, D. C.; F. R. Jennings; H. R. Jessy; E. B. Jewett, Clarksburg, W. Va.; R. W. Johnson, Butler, Pa.; J. S. Johnston; J. L. Jones, Oakmont, Pa.; W. P. Jones; Paul Jorstad, Wilkinsburg, Pa.; E. T. Joyce; F. B. Kelley, Rochester, N. Y.; J. A. Kelly; H. W. Kellogg, Niagara Falls, N. Y.; C. L. Kemery; Mr. and Mrs. P. Kemery; E. F. Kenney, Johnstown, Pa.; S. M. Kier, A. Kingsbury; F. Kjeim; L. L. Knox; F. L. Koethen; M. M. Kohn, New York; C. R. Krueger, Schenectady, N. Y.; Dr. R. Krulla, Vienna, Austria; E. F. Lake, New York; A. B. Lamb, New York City; H. G. Lamker, H. M. Lane, Cleveland, Ohio; J. Langton, New York; Mr. and Mrs. H. C. Laudebeck, Wilkinsburg, Pa.; Mr. and Mrs. H. H. Laughlin; E. J. Lavino, Philadelphia, Pa.; G. H. Leeds; Robert B. Leighton, Northumberland, Pa.; J. D. Lewis, Braddock, Pa.; F. A. Lidbury, Niagara Falls, N. Y.; Mr. and Mrs. P. M. Lincoln, W. B. Lindall, Toronto, Canada; D. H. Littell; M. G. Lloyd, Chicago, Ill.; Chas. Longanecker; F. H. Loveridge, Chicago, Ill.; A. Lowenstein, Chicago, Ill.; C. W. Lytle, Sharon, Pa.; J. K. Lyons; W. L. Macinskey, Gordon F. Marble, Schenectady, N. Y.; S. J. Marchan; C. W. Marsh, New York City; J. F. Martin; E. K. Mason; H. A. Mason; J. F. Mateu; P. L. May Steubenville, Ohio; W. A. McCaffey; E. H. McClelland, W. M. McConahey, W. H. McCune, Vandergrift, Pa.; R. A. McDonald; L. H. McFeely, J. N. McIlvaine; F. F. McIntosh, Sewickley, Pa.; F. McIntosh, W. S. McKee, Chicago, Ill.; Jos. McKinley; J. F. McKirdy; W. N. McKnight; D. J. McLaughlin, Munhall, Pa.; C. A. Menk, Munhall, Pa.; F. S. Merriam, Marietta, Ohio; J. L. Merrill; K. S. Miller; H. S. Miner, Gloucester City, Pa.; H. B. Mitchell; Miss R. I. Mitchell; J. D. Mooney; L. A. Moran; W. P. Morgan; C. H. Moritz, Niagara Falls, N. Y.; E. K. Morse; W. R. Mott, Cleveland Ohio, J. M. Muir, New York; Mr. and Mrs. H. N. Muller; D. H. Murphy; S. Myasaki; H. A. Neel, Dravosburg, Pa.; George Neilson, Oakmont, Pa.; A. B. Norton, Jr.; Mr. and Mrs. I. E. Ober; J. A. O'Callaghan, D. L. Ordway, Cleveland, Ohio; R. S. Orr; L. A. Osborne; G. S. Page, L. R. Palmer; F. L. Parker, Philadelphia, Pa.; C. W. Parkhurst, Johnstown, Pa.; L. A. Parry, Gettysburg, Pa.; A. G. Parsons, Buffalo, N. Y.; Wm. C. Parsons, Braddock, Pa.; R. S. Perry, Philadelphia, Pa.; N. Petunot, Niagara Falls, N. Y.; A. Pinkerton; F. B. Porch; H. C. Porter; F. P. Poste, A. L. Quereau, Philadelphia, Pa.; K. C. Randall; H. S. Randolph, J. C. Reed, Steelton, Pa.; R. C. Reed, Duquesne, Pa.; W. E. Reed; P. P. Reese, Munhall, Pa.; C. R. Reicher; C. R. Rhodes; C. H. Rich, Clearton, Pa.; Jos. W. Richards, South Bethlehem Pa.; H. K. Richardson, State College, Pa.; H. K. Rigdon; C. D. Robb, New York; J. C. Roberts, H. Rodman, East Pittsburgh, Pa.; E. F. Roeber, New York; A. G. Rogers, Niagara Falls, N. Y.; F. M. Roller, New York; F. Rose, W. J. Rouch; T. Rowland, Niagara Falls, N. Y.; H. H. Rudd; Louis Ruhl, New York, D. J. Rusin, B. Rutherford; S. S. Sadtler, Philadelphia, Pa.; B. D. Saklatwalla, L. E. Saunders, Niagara Falls, N. Y.; W. M. Saunders, Provi

dence, R. I.; A. C. J. Schabacker; C. G. Schluederberg; C. H. Schultz, Ravenna, Ohio; R. C. Schwarz, Rochester, N. Y.; A. P. Scott, Breckenridge, Pa.; Mr. and Mrs. Chas. F. Scott; Ralph L. Seabury, Cleveland, Ohio; R. W. Seabury, Boonton, N. J.; J. O. Seede, Schenectady, N. Y.; H. W. Seldon, Verona, Pa.; F. M. Sickler, Los Angeles, Cal.; H. F. Sill; A. Silverman; A. T. Simonds, Chicago, Ill.; Mr. and Mrs. C. E. Skinner, Wilkinsburg, Pa.; R. H. Skinner; V. Skillman, Ithaca, N. Y.; H. Slocum; F. L. Slocum; A. W. Smith, Cleveland, Ohio; E. S. Smith, Niagara Falls, N. Y.; M. A. Smith; R. G. Smith, Irwin, Pa.; Wm. A. Smith, Niagara Falls, N. Y.; J. A. Smitsmans; E. H. Smythe, Chicago, Ill.; W. O. Snelling; J. L. K. Snyder, Cleveland, Ohio; R. A. L. Snyder; A. W. Soderberg; A. L. Sonnhalter, Sharon, Pa.; W. L. Spaulding, Buffalo, N. Y.; E. C. Speiden, Niagara Falls, N. Y.; Mr. and Mrs. F. N. Speller; E. R. Spencer; E. A. Sperry, New York City; E. C. Sprague, Niagara Falls, N. Y.; Mr. and Mrs. S. G. Stafford, Coraopolis, Pa.; R. H. Stevens, Munhall, Pa.; C. G. Storm; W. J. Strassbergh; F. M. Sturgess, Swissvale, Pa.; Haakon Stym, Christiania, Norway; M. Tattersfield, Schenectady, N. Y.; C. E. Taylor, Penn Yan, N. Y.; Mr. and Mrs. E. R. Taylor, Penn Yan, N. Y.; Miss E. B. Taylor, Penn Yan, N. Y.; H. G. Texter; B. F. Thomas, Chattanooga, Tenn.; R. D. Thomas, Oakmont, Pa.; E. C. Troutman, Woodsfield, Ohio; S. A. Tucker, New York; C. P. Turner, Harrisburg, Pa.; M. C. Turpin, Wilkinsburg, Pa.; F. Uhlenhaut; F. Uhlenhaut, Jr.; J. S. Unger, Duquesne, Pa.; G. N. Vardy; C. C. Vogt; R. M. Von Hofen, Sewickley, Pa.; Mr. and Mrs. A. von Isakovics, Monticello, N. Y.; L. D. Vorce, Detroit, Mich.; M. Waddell, New York; F. L. O. Wadsworth, Sewickley, Pa.; R. E. Waggoner; A. A. Wagner; W. H. Wagner; W. G. Waitt, Fremont, Ohio; Mr. and Mrs. S. S. Wales, Munhall, Pa.; W. H. Walker, Boston, Mass.; H. E. Walters; J. A. Ward, Wilkesbarre, Pa.; G. S. Warren, Sharon, Pa.; M. M. Waterman, New Castle, Pa.; R. H. Watson; O. P. Watts, Madison, Wis.; G. E. Webster; Wm. S. Weedon, Wilmington, Del.; G. Werlin, Philadelphia, Pa.; G. A. White, Vandergrift, Pa.; R. H. White, Natrona, Pa.; W. Whited; Mr. and Mrs. J. Whiting, Boston, Mass.; M. A. Whiting, Schenectady, N. Y.; C. S. Wickes, Merchantville, N. J.; R. S. Wile; B. Wiley; W. B. Wilkinson; W. A. Williams, Buckingham, Quebec; J. R. Witherow, Columbus, Ohio; T. A. Witherspoon, Washington, D. C.; W. J. Woolridge, Pittsfield, Mass.; T. A. Worcester, Schenectady, N. Y.; Wm. R. Work; E. Yawger; R. L. Young; S. E. Young; Wm. Yost, Creighton, Pa.; Mr. and Mrs. J. A. Yunck, South Orange, N. J.

Notes on Chemistry and Metallurgy in Great Britain.

The Rubber Outlook.

The synthetic production of rubber is being keenly discussed in London amid the incentives of phenomenally high prices (300 to 400 per cent above those of 10 years ago), an avalanche of flotations of rubber companies (many of the wildcat order), to say nothing of inflated share prices and wildly reckless gambling. The elements of danger to the industry are numerous—synthetic rubber not least of all. Everybody is talking of it, and quite a number of people have actually in their possession strips of rubber alleged to have been produced in the laboratory. Some samples which the writer has examined are so similar in their visual appearance to plantation rubber as to call to mind the way in which a prominent financial magnate was hoodwinked over the artificial production of diamonds in Paris a year or two ago.

The daily talk about synthetic rubber is becoming keen, and one hears of ounces being produced. The cost of production is mentioned as ranging from 1 to 10 shillings per pound for materials alone. Nothing is yet stated to have reached the practical stage, practical being defined as the making of 25 lb. or 100 lb. lots. There is, therefore, the danger of rubber planting being hit as hard as indigo planting has been hit by the success of the chemist. All this is being ridiculed by the officials of the various companies.

Other dangers lie in insect pests or fungoid diseases, for nature seems to be quite capable of producing the equivalent of the cotton boll weevil.

Overproduction must also be regarded as the certain outcome of a few years hence. It is even possible at an earlier date, as the present high prices are an incentive to the extensive collection of large amounts of rubber from the wild trees. If this be on the reckless scale which has reduced the West African output through overtapping, this will ultimately bring more harm than good. The tapping of immature trees is also a likely possibility on a number of the plantations.

Up to the time of writing these words there are no signs of a retrogression in the price of rubber itself. There are fortunately signs that the investing mania has reached its zenith.

Faraday Society.

The visit of Professors Walden and Guye to the meeting of the Faraday Society, to read papers giving their latest investigations into the "Constitution of Water," was the occasion for a very pleasant gathering at the Trocadero Restaurant on April 26. The distinguished foreign guests were entertained to dinner by the society, this being the first dinner ever held by this body. About 45 members were present, and the guests included Sir Wm. Ramsay, K.C.B., F.R.S.; Sir Wm. Jilden, F.R.S.; Dr. Wilsmore and Sir Joseph Larmor.

As Mr. Blount remarked, in proposing the health of Sir Wm. Ramsay, it was difficult to say who were guests and who hosts, so thoroughly was the whole company in sympathy. In returning thanks for the toast of his health Professor Walden, who spoke first in English and then in German, after a modest disclaimer of his ability to render his meaning clear in English, the reason for which was not apparent, made feeling reference to the good work effected by science in overcoming national differences, and providing a common meeting ground for the natives of all civilized countries. He attributed the introduction of chemical knowledge into Russia to an Englishman, Pegram, and cited the many well-known names illustrious in the annals of chemistry which have belonged to natives of this country.

Professor Guye, who spoke in French, referred to the early years of Faraday, who had studied under a Swiss, and may be said to have first had his genius appreciated by a member of that nation. The toasts of the "Associated Societies" was responded to by Dr. N. T. M. Wilsmore, president of the Chemical Society, who referred to the growth of the Faraday Society and made appreciative remarks on the papers of the evening before.

The president of the society, Mr. James Swinburne, F.R.S., past-president of the Institution of Electrical Engineers, referred to the work done by Mr. Sherard Cowper-Coles in founding the society, which was intended more particularly for the study of electrochemistry.

Nitrogen in Mild Steel.

Mr. C. E. STROMEYER, in a paper presented at the spring meeting of the Institution of Naval Architects on "The Brittleness of Mild Steel Due to Nitrogen," said that until quite recently nothing was known with regard to the effect of nitrogen on steel in causing brittleness. For phosphorus 0.060 per cent was considered to be a reasonable limit, and steel plates containing more were open to suspicion. This view had now to be modified. One sample which the author had tested, which was undoubtedly of very bad quality, contained only 0.047 per cent of phosphorus, but it also contained 0.0123 per cent of nitrogen, which was more than twice as much as that contained in any good steels. Another sample, from a boiler plate which burst under hydraulic test, contained only 0.052 per cent of phosphorus, but 0.0200 per cent of nitrogen, or four times as much as is contained in good steels.

Nitrogen had 10 times the effect of phosphorus in raising the tenacity of steel: an addition of 0.01 per cent increasing the tenacity by nearly three tons. A safe rule was that the sum of the percentage of phosphorus *plus* five times the percentage of nitrogen should not exceed 0.080 per cent.

No investigation into the cause of failure of a mild steel plate could be considered complete unless both the phosphorus and nitrogen had been determined. The nitrogen determination was simple and rapid, but precaution had to be taken to guard against absorption of ammonia. Some continental angle irons intended for small portable locomotive boilers failed in the boiler shop; they contained 0.0146 per cent of nitrogen, and portions of a damaged furnace plate proved to be quite brittle. The nitrogen was 0.0180 per cent. Fractured steel plates from a ship being built in a continental port, which plates had been tested and passed, contained 0.023 per cent of nitrogen. These percentages were from three to five times as great as the maximum found in any good steels.

As to how the nitrogen got in the steel, it had not been found possible to combine nitrogen with steel by heating the two together; and the only known means of effecting combina-

tion was to heat steel in an atmosphere of ammonia. The nitrogen probably entered the pig iron in the blast furnace, and could not be removed by subsequent heating. The addition of ferrotitanium alloy to steel had not demonstrated that titanium affected the removal of nitrogen. Nevertheless, a strong argument in favor of the belief that titanium could reduce the percentage of nitrogen was recently brought to the author's notice by the discovery inside the dismantled blast furnaces of the Farnley Iron Company of some crystals of titanium nitride, which were now in the Leeds Museum.

Switch and Transformer Oils.

At the meeting of the Manchester Local Section of the Institute of Electrical Engineers, on March 22, a paper on the properties of "Switch and Transformer Oils," by Mr. W. Pollard Digby and Mr. D. B. Mellis, was read. The authors considered that oil for switch and transformer use had not received sufficient care in its selection nor due investigation during its use. The electrical engineer relied on the literature of the firms supplying oils of this character, and it was by no means certain that the whole of a consignment would be equal to the special sample submitted.

The properties they regarded as requisite were absence of moisture; good dielectric strength; a low viscosity, to facilitate the transfer of heat from the core and windings to the case; a flash-point at a temperature two or three times that of the designed limit of the transformer; a perfectly neutral reaction; absence of vegetable oil or resinoid matter; and free from metallic compounds.

The authors had found that specific resistance was a very good index of purity. They determined the resistance between a pair of metallic disks, the distance between which was regulated by a micrometer screw, taking readings on a "megger" or other kind of ohmmeter. Of a series of six typical oils, the variation was from about 600,000 megohms to 6.5 million megohms per c.c.

For dielectric tests electrodes consisting of a point and a disk were found to answer best; and the authors described at considerable length their tests of oils as received and after baking. Baking was found to increase the dielectric strength where this was less than 10,000 volts. The specific resistance for oils of high resistance was not increased, but where a lowered specific gravity indicated evaporation of moisture there was an increase. Generally there was increase in specific gravity and viscosity, with some raising of the flashing point.

The authors gave detailed descriptions of all the physical and chemical tests which they considered necessary for the complete examination of oils. Results of tests undertaken to investigate changes produced in oil switches by 200 breaks of a definitely known current went to show that a switch could be tripped a very great number of times before the properties of the oil perceptibly deteriorated. The amount of carbon produced by prolonged arcing was also determined in several samples, and was found to vary with the physical and chemical properties. The authors also suggested that specific resistance measurements might be taken in a series of heating and cooling curves to determine the point at which certain molecular regrouping of the hydrocarbons seem to occur.

The Classification of Steels.

The proposals of the committee of the International Association for the Testing of Materials with regard to the definitions of iron and steel have not been favorably received in Sheffield, where the prevalent opinion is that those proposals, if adopted, will be prejudicial to the status of the best class of Sheffield products and will open the door to malpractice.

The committee recommended that the term "blister steel," which is restricted to cemented Swedish wrought iron or wrought iron of equal quality, should in future include "steel made by carburizing wrought iron by heating it in contact with carbonaceous matter, which may also be made by carburizing a low carbon steel," and an important meeting of Sheffield manufacturers was held on April 19 to consider the question.

Professor Arnold indicated the effect of the adoption of the proposal in facilitating the introduction of inferior metal into the trade, and demonstrated that ultimately, not only would the quality of blister steel be involved, but both sheer and crucible steel would be degraded. A committee was appointed to protect the interests of the crucible-steel trade and to communicate with the International Committee.

The Action of Sewage on Concrete.

Mr. Sidney H. Chambers, at the last (April) meeting of the Concrete Institute, stated that his experience at Hampton sewage works did not support the widely accepted opinion that no marked disintegration need be anticipated in sound concrete either from sewage or its emanations. The concrete in question consisted of one part Portland cement and six parts of clean ballast. A few months after the tank was put into use the concrete roof and walls of the hydrolyzing chambers, above the level of the sewage liquid, became covered with a chalky-looking efflorescence which was easily removed and was found to be mainly sulphur. Subsequently the walls and roof of the second part of the tank exhibited signs of peeling and disintegration.

The first part of the tank—the hydrolytic chamber—had only the slight deposit of sulphur and a slight erosion at the liquid level; but in the second part—the hydrolysing chambers—the action was much more marked, and on removing the eroded surface the underlying concrete was found to be moist, soft and gritty. The greatest effect was at the varying liquid level and depended upon the quantity of sulphuretted hydrogen in solution. There was comparatively little of this gas at the intake or in the hydrolytic tank; but its amount increased in the liquid passing through the hydrolysing chambers.

With a varying level the concrete was left moistened with a film of liquid containing sulphuretted hydrogen which was oxidized by the air, and thus sulphur was liberated and sulphuric acid produced, the result being that the lime in the cement was finally converted into calcium sulphate. The intermediate product of the oxidation of the sulphuretted hydrogen was probably sulphurous acid. As the liquid level rose the decomposed concrete was washed away and a fresh surface was exposed to the action when the level fell.

The investigation indicated that the gases dissolved in sewage and those emitted from it acted injuriously on Portland cement when there were (1) a high degree of putrescence of the sewage; (2) a moistened surface retaining or absorbing the gases resulting from putrefaction; and (3) a free supply of air; but that in the absence of any one of these conditions serious erosion was not to be feared.

Plating Powders.

A preparation which is being extensively advertised and was the subject of a paper read before the Society of Arts on Feb. 2, 1910, exemplifies the truth of the saying "there is nothing new under the sun." An old book of recipes, dated 1823, gives a formula for silvering by a mixture of silver chloride, salt and cream of tartar, to be rubbed on the surface of the article to be plated.

In the *Chemist and Druggist* of July 15, 1881, appears the following formula for an "instantaneous silvering powder": Silver chloride, three parts; cream of tartar, 20 parts; sodium chloride, 75 parts; mix. Moisten a portion with water, and with a piece of blotting paper rub it on the clean metallic object, which is to be rubbed with a piece of cotton on which precipitated chalk is dusted; wash with water, and polish with a dry cloth.

In Cowley's "Encyclopedia of Practical Recipes," sixth edition, Vol. 2, page 1496, appears a similar formula with the addition of whiting (purified chalk). It is within the writer's experience that these recipes easily and quickly produce a film

of metallic silver on brass and copper articles, but the thickness of the coating is so slight that the process would be of little use for articles exposed to any degree of wear.

The material recently put on the market differs from the powders referred to above by the addition of magnesium powder to act as a positive element; and this difference appears to constitute the essential feature of the patents which have been secured.

Market Prices.

April, 1910.—Copper has shown a downward tendency throughout the month. Starting at £58 it fell after the 7th, reaching £55.5.0 on the 26th, thence recovered to £56.3.9.

Tin has been irregular, dropping away from the opening £152, and touching £147.15.0 on the 13th. Closes at £148.

English lead slight decline during the first week, thence steady, closing at £13.

Hematite opened at 68/7½, rose in the course of a week to 68/9, thence declining to a final price of 67/1½.

Iron, Cleveland warrants, rather jerky, commencing 51/6, dropped by the 13th to 51/-, and by the 25th to 50/6. Present price (29th) 50/3.

Scotch pig, following the same line as Cleveland, opened at 57/6 and closed at 56/3. Downward tendency throughout the month.

	£. s. d.
Aluminium ingots in ton lots	80. 0. 0
Ammonia—alum, lump, loose	5.12. 6
Antimony black sulphide powder, per ton	20. 0. 0
Borax, best refined crystal, per ton	16. 0. 0
Sulphate of ammonia, f.o.b. Liverpool	12. 2. 6
Sal-ammoniac, lump, first delivered U. K.	42. 0. 0
Copper sulphate	18. 0. 0
Camphor, 1 oz. tablets	1. 7
Caustic soda, 48 per cent ordinary	5.10. 0
Bleaching powder, 35 per cent	4. 5. 0
Shellac, Standard, T. N. Orange Spot, per cwt.	3.10. 0
Sulphur, recovered, per ton	5. 0. 0
Carbolic acid, liquid, 97/99 per cent, per gal.	1. 0
Hydrochloric acid, per cwt.	5. 0
Creosote, ordinary good liquid, per gal.	2¼
Naphtha solvent, 90 per cent at 160° C., per gal.	1. 4½
Rubber, Para, fine, per lb.	11.10
Platinum, per oz.	6. 0. 0
Zinc, sheet (Vieille Montague brand)	26.15. 0

Rubber is still engrossing the attention of the Stock Exchange. Fine Para, which opened at 10/5½ per pound, rose steadily to 11/4½, then to 11/8, and touched at mid-month 12/3. It subsequently declined to 11/10.

There is an obvious shortage of this material, which appears to have been coming on for nearly two years. Extensive planting in the Malay Peninsula has been encouraged by present prices, and over 100 companies interested in rubber have issued prospectuses since Jan. 15. Very large prices are at present being given for shares. A fall in the price of these securities would not necessarily entail cheaper rubber, although possibly the reaction might check the demand for a time. Speculation can have little influence on the actual commodity itself. Of the other prices it will be noted that aluminium has risen, while the other metals, with the exception of platinum (always a rather vague price), are inclined to fall off.

Naphtha solvent again moves upward, presumably in sympathy with oil.

London, April, 1910.

Ductile Tungsten.—At an A. I. E. E. meeting on May 27, Dr. W. D. Coolidge, of the Research Laboratory of the General Electric Company, read a paper on ductile tungsten, which is printed in the June issue of the *Proceedings.*

SYNOPSIS OF METALLURGICAL AND CHEMICAL LITERATURE.

Iron and Steel.

Case-Hardening.—A paper by Mr. SYDNEY A. GRAYSON, presented at the May meeting of the Iron and Steel Institute, gives an account of "some recent investigations on case-hardening." The author shows that it is necessary to classify case hardening compositions, both by the carbon percentage obtained in the "case" and also by the graduation of the carbon diffusion, which is best shown graphically. This classification is necessary on account of one composition being more suitable for certain kinds of work than another. A high carbon "case," such as 1.10 per cent centigrade would be very efficient for the kind of work where the pressure was fairly constant, such as a plain bearing, but it would be very unsuitable and inefficient for parts which had to resist repeated shocks, because of the strong tendency of the high carbon "case" to chip or even to peel off. Consequently, it is advisable, where all kinds of case-hardening have to be done, that two compositions be used—one of them to produce a high-carbon "wearing surface" and the other to produce a medium-carbon "wearing surface." On the other hand, for a composition to be used for "general" purposes, it should produce a "wearing surface" containing a percentage of carbon equal to that of the eutectoid 0.9 per cent in order to give the most efficient results.

The evidence obtained from the author's results respecting the carburizing temperature shows that the best temperature is 950° C. to 1000° C., although the high temperature 1000° to 1050° can be used with advantage in some cases. The results obtained at 900 C. to 950° C. tend to show that although there is an active carburizing atmosphere produced in the box the temperature of the steel is only high enough to allow of a slow diffusion of the carbon, consequently supersaturation of carbon takes place at the surface of the steel.

Diffusion of the sulphur seems to proceed in a similar way to that of the carbon at the temperature of 900° C. to 950° C. and 950° C. to 1000° C. All charred leather material does not contain this active amount of sulphur, although the sample used by the author was obtained from a well-established firm, and was used in this investigation in order to find out, if possible, the cause of a soft skin, which was always produced under certain conditions by this material, of which this sulphur diffusion is the most probable explanation. The brown scintilla material contains a fairly large amount of sulphur, and which exists chiefly in the sulphide form, although it does not seem to act by any means as keenly as the sulphur in the charred leather material. This is most probably due to Scintilla material having a high percentage of volatile hydrocarbons, which carries off the effective part of the sulphur with it. (From advance sheets.)

Steel for Gears.—Mr. LOUIS P. M. REVILLON has tested the leading types of steel offered by French manufacturers for the construction of gears (especially speed-change gears for motor vehicles) and also products supplied from two German steel works. The results are given in the *Iron and Steel Institute Carnegie Scholarship Memoirs,* Vol I, pages 161 to 218. In the whole 26 steels were tested, which are divided into four classes. I, steels which contained neither nickel nor chromium; they were, generally speaking, manganese-silicon steels; II, nickel-chromium steels with low percentages of nickel and low and medium percentages of carbon, quenching in water or oil; III, nickel-chromium steels with low percentages of nickel and higher percentages of carbon, quenching in oil and sometimes even in air; IV, steels with a medium percentage of nickel, either with or without chromium and a variable percentage of carbon; they were frequently capable of undergoing air quenching. "Some of these steels were made by the open-hearth process, in large quantities; others were made in crucibles alone, while finally the electric furnace, being now not infrequently in

use for obtaining high-class steels, several of the samples employed in the experiments came from firms which employ this method of manufacture."

The following conclusions are reached: It is possible to find among either class of sample a steel more or less suitable for use in making gears. Steels not containing nickel are commercial products which can be obtained at low cost, and can be made, after treatment, to give hard gears, but have relatively a high degree of brittleness. They have been much in favor, but they require complicated heat treatment and manipulation closely adjusted to their composition.

Soft nickel-steels quenched in water are insufficiently hard, but they are susceptible of being case-hardened and then furnish products of superior quality.

With an increasing percentage of carbon it is possible to find steels quenching in oil or in air, the properties of which as regards resistance to shock and the elongations of which are increased by the presence of nickel.

When the percentage of nickel is increased it is possible to obtain, with a well-adjusted carbon percentage, a steel which, with a simple treatment, will yield the best results. Annealing should, however, be very carefully conducted in order to soften the metal well for the purpose of machining, while after having completely finished the piece air quenching will suffice to insure good hardness and good resistance to shock. The great defect of these steels is their extreme sensitiveness to the smallest variations in composition. They require the greatest watchfulness in the metallurgical works in order to yield the results one is justified in expecting to obtain.

Apart from the steels of the first class, for which a complicated heat treatment is necessary, it is always possible to find, among nickel-steels, a steel corresponding in properties to all the requirements of a specific manufacture and requiring but simple treatment; that is to say, capable of being used after simple quenching without subsequent annealing—annealing being from the industrial point of view impossible to define scientifically, and difficult to adjust uniformly from one piece to another when the volume of the piece is altered.

Special Ternary Steels; Shearing Test; Electrical Resistance.—An extended investigation of two problems of the theory of special ternary steels, carried out by Mr. Albert M. Portevin, is reported in the *Iron and Steel Institute Carnegie Scholarship Memoirs*, Vol. I, pages 230 to 364. The first problem discussed is the application of shearing tests to the special steels with the object of ascertaining the influence of the elements added on the results of these tests and of examining their relation to tensile tests. Nickel-steels, manganese-steels, chrominum-steels, tungsten-steels, aluminium-steels, molybdenum-steels, vanadium-steels, titanium-steels, boron-steels and tantalum-steels were tested, and the principal conclusions reached are as follows: The results on shearing are, like those of the tensile tests, influenced to a considerable degree by the structure, and may serve to distinguish the different types of steel, e.g., pearlitic, martensitic, polyhedral, etc. The shearing test may, therefore, serve like the tensile test to classify these steels. If it is asked whether the shearing test is capable of replacing the tensile test, the answer will be that it would be necessary that there should be in existence formulæ correlating for all these steels the characteristic results of the tensile tests with those of shearing. To establish a relation between the tensile and shearing tests of special steels, it is necessary to take into account (1) The nature of the element added to the steel, (2) the structure of the steel, and (3) the limits of tensile strength within which the formula becomes applicable.

It is beyond contest and the general appearance of the curves demonstrate the fact that the characteristics of the tensile tests and those of shearing depend upon variables common to both, but the formula becomes complicated by reason of the intervention of other variables, which differ in each case, and the influence of which may render any simple relation, applicable to a wide range of steels, impossible. Although the shearing tests does not admit of the supersession of tensile testing, it does not follow that it is inferior to the latter. On the contrary, it brings out other properties, and, in addition to the advantages resulting from the case with which the test piece can be prepared, it possesses the superior advantage, allows of the homogeneity of the metal being ascertained, owing to the small size of the test pieces and the possibility of renewing the test on sections close together, thus eliminating the influence of local defects, whereas the tensile test only yields an average result. This is evidently an important consideration from the point of view of the application to be made of the steel, but a disadvantage from the point of view of research because of the intervention, pell mell, of a host of factors; the inequality in chemical composition and in the heat treatment of the sample, and all the special accidents, cracks, hollows, etc., which help to falsify the results. The test has nevertheless its own particular application, which is both old-established and widespread.

The second problem discussed by the author is the electrical resistance. The same steels as mentioned above were tested and the most significant fact which is brought out is that the laws as to the variation in the specific electrical resistance of the ternary steels, with the amount of the added element, undergo fairly sudden changes corresponding with alternations in the microstructure, and that if these laws be plotted as curves they can, generally speaking, be separated into a certain number of rectilinear portions, the weak curvature of each of which is characteristic of a definite micrographic condition of the steel, pearlitic, martensitic, polyhedral, or accompanied by the appearance of a double carbide. There are two appendices following the paper dealing with the influence of vanadium and titanium on the transformation points of steel.

Gold and Silver.

Cyaniding Sulphide Concentrates.—In the March 19, 1910, issue of *Min. & Sci. Press*, A. E. Drucker describes his method and device for cyaniding sulphide concentrates as worked out in Korea. He uses a combined air-agitation and continuous leaching vat which embodies the good points of other apparatus which the author has seen in various countries. His original feature consists in the plow used to keep the bottom filter free from slime-cake so that the leaching and filtering process can be continuous. The dimensions of the vat are, for 10 tons, 6 ft. x 15 ft.; 20 tons, 7 ft. x 25 ft.; 40 tons, 8 ft. x 35 ft. The smaller dimensions are diameters and the larger are the heights of the tanks.

The sulphide concentrates are slimed in tube mills, and the pulp fed to the vats is about 1 to 1 by weight. The agitation and plowing device is in the bottom of the vat and the operating mechanism located beneath and outside. The filter cloth is about 6 in. above the bottom of the tank and is supported by a bottom of 3-in. boards bored with 1-in. holes. Over the filter cloth is placed a ⅛-in. sheet-iron cover, perforated to correspond with the wooden filter bottom. The revolving plows can be raised or lowered and traverse the sheet-iron plate, keeping it clear of slime, which cements very strongly in the case of sulphides. Compressed air for agitation is delivered through each plow shoe and is thus evenly distributed at the bottom of the vat and keeps the charge agitated. Filtration proceeds constantly, the filtrate going to process for clarifying. When the pulp has been reduced to about 30 per cent moisture, barren cyanide washes are introduced through the same orifices that carried air and the charge is thus thoroughly washed. Pressure filtering then proceeds as before until the pulp has about 30 per cent moisture, when it is dicharged to filter presses and given a weak wash. The following figures are given for a 40-ton vat and a 4-day treatment

Forty-Ton Charge.	Horse-power.
For driving short arms (intermittent power).	1
Air agitation (15 cu. ft. air per minute at 26-lb. pressure; intermittent power)..	1½

Total intermittent power	2½
Continuous power	2
Continuous power per ton treated	0.05

The operations can be divided as follows after all-sliming the sulphide within a tube mill in cyanide solution, taking for example a four-day treatment:

1. Collecting a full charge in agitator (undergoing an air agitation and leaching while filling), 24 hours.

2. Air agitation and continuous leaching (cyanide solution carrying gold and silver, continuously flowing to the zinc boxes; pulp finally pressure-filtered down to 30 per cent moisture before washing), 36 hours.

3. Intermediate barren cyanide wash (mixing and finally pressure-filtering down to 30 per cent moisture before sending to filter press), 12 hours.

4. Filter-pressing, weak wash, discharging, 24 hours.

Total, 4 days.

ANALYSIS OF SULPHIDE SLIME TREATED IN KOREA.

No. 1 Sulphide.	Per Cent.
Iron pyrites (FeS_2)	56
Galena (PbS)	36
Sphalerite (ZnS)	6
Arsenopyrite (FeAsS)	2
	100

No. 2 Sulphide.	Per Cent.
Arsenopyrite	72
Marcasite	27
Galena and Sphalerite	1
Copper pyrites	traces
	100

No. 1 sulphide carries from $40 to $50 in gold and yields from 92 to 94 per cent extraction in four days. The old method of leaching for 18 days yielded an extraction of 84 per cent. No. 2 sulphide carries $20 to $30 in gold and yields 92 per cent extraction in four days, as against 70 to 75 per cent in 16 days by the old treatment of leaching in shallow vats.

The author gives five points to be observed in the cyanidation of sulphide concentrates: (1) To be treated when fresh and reground in tube mills with strong KCN solution. (2) To be air-agitated and aerated in a combined agitator and leaching vat for 20 to 36 hours. (3) To provide for continuous filtration so that at the end of the treatment the bulk of the cyanide solution has been precipitated in the zinc boxes. (4) To preserve the protective alkalinity of the solution and keep the cyanide strength up to 0.25 per cent KCN. Use KCN in preference to NaCN where air agitation is employed. (5) Do not allow the cyanide solution to become so foul as to become inactive.

Copper.

Methods of Analysis.—In the *Mining World* for April 2, 1910, Mr. EVANS W. BUSKETT gives details of the methods of analysis used in the laboratories of the Cananea Consolidated Copper Company. Methods are given for furnace slags, settler slags, ores, chrome bricks, copper bullion, coal and coke. There is nothing novel about any of the methods used, but it is always of interest to note the practice at different plants. Many chemists will differ with the author in some of his methods. Both electrolytic and permanganate-sulphocyanate methods are used for copper, practically as devised by Guess. The use of the chromate method for lead is interesting in view of the attempt of Western chemists to work out a satisfactory method along this line. Fusion methods are used for arsenic and antimony.

Lead.

Bismuth Lead Smelting.—Data on the blast-furnace smelting of bismuth ore are rather meagre, and consequently any information on the subject is of interest. A record of the results obtained in a 26-day run on a mixture of lead and bismuth ores is communicated to the *Eng. and Min. Jour.* for April 9, 1910, by S. E. BRETHERTON. The Bismuth King Mine in Sinaloa, Mexico, produced a bismuth ore of the following composition: Bismuth, 2 per cent; iron, 33 per cent; silica, 31 per cent; alumina, 12.5 per cent; magnesia, 2 per cent; zinc, 1 per cent; copper, 9 per cent, and a trace of lime, arsenic and sulphur. The principal value is gold, and there is some silver. The lead ore produced carried: Lead, 50.7 per cent; copper, 1.4 per cent; zinc, 8.4 per cent; sulphur, 18 per cent; iron, 6 per cent; silica, 11 per cent, and alumina, 1 per cent. At the time the author made the test under consideration, there was also on hand from a previous campaign some lead bullion containing no bismuth and some silver. The furnace was 30 x 60 in. at the tuyeres and 14 ft. 10 in. from crucible to feed floor. A forehearth was improvised from an abandoned boiler. The first charge used carried 9.75 per cent lead and 1 per cent bismuth. The slag from this charge assayed 1.2 per cent lead and 1.1 oz. silver, with bismuth and gold present in quantities too small to estimate. This indicated a good saving of all the metals, so in order to curtail the lead ore and enrich the bullion in bismuth and gold, the amount of lead on the charge was reduced one-half, and the bullion was fed back into the furnace until it assayed over 17 per cent bismuth. About this time the bismuth losses began to rise, no doubt due to insufficient lead in the charge. Increasing the lead increased the saving. As soon as this feature was demonstrated it was decided to install a cupel furnace in order to secure the following advantages: (1) Freight and treatment saved by shipping richer bullion. (2) Less burden to be smelted in the furnace to get the necessary lead. (3) Litharge from the cupel furnace mixes well with the blast furnace charge, and (4) enables the metallurgist to carry a higher percentage of lead in the charge where lead ore is limited. In the cupeling process, bismuth is concentrated with the silver up to a certain point, say 50 to 60 per cent silver, when it begins to go off with the litharge. The author concludes that 80 per cent of the bismuth contained in a 2 per cent ore can be recovered by lead smelting in the blast furnace by using not less than 10 per cent lead in the charge and suitable dust chamber and bag house connections. The author also refers to his experiences with the use of green hardwood instead of coke for blast furnace fuel. He believes that green hardwood aided by the hot blast should be effective in copper matte smelting on a small scale.

Miscellaneous.

Thoulet Solution in Ore Testing.—The use of the Thoulet heavy solution is described in *Mines and Minerals* for April, 1910, by Mr. HENRY B. HALLOWELL. This solution is made by dissolving four parts of weight of potassium iodide in six parts of warm water and then adding five parts of mercuric iodide, stirring constantly. Evaporate until a slight crystalline scum forms on the surface and then filter. The clear solution has a specific gravity of about 3.2 and may be brought to any desired degree of dilution with distilled water. The author uses the solution for separating constituents of sand mixtures, crushed ore, jig middlings, etc., using an ordinary separatory funnel. He finds that valuable information can be obtained by this simple means, such as the character of middlings and tailings in concentrators, the control of jig middlings, where regrinding is practiced and where the free gangue must be kept as low as possible, and in the estimation of percentages of mineral in an ore.

Matte Refining.—On account of the presence of some galena in the concentrates at the cyanide plant of the Mono Mining Company, Yreka, Cal., the zinc box precipitates contain considerable lead, which forms a matte in the process of melting the precipitates. In the *Pacific Miner* for March, 1910, Mr. J. E. COLLIE describes the method used to recover gold and silver from this very rich lead matte. The matte is broken into coarse pieces and melted with scrap iron. Metallic lead, carrying the gold and silver, is precipitated, and a barren iron matte is formed. The amount of iron used is calculated, assuming the

lead matte to be pure PbS and the resulting iron products to be FeS. Two-thirds of the necessary quantity is added as nails and small scrap, and when this is consumed, long iron spikes are added in the same manner as nails are used in assaying sulphide ores in the crucible.

RECENT METALLURGICAL AND CHEMICAL PATENTS

Gold and Silver.

Filters.—Modifications of the well-known Moore process of vacuum filtration of fine pulp have been made the subjects of letters patent granted to Mr. George Moore, of Salt Lake City. The principal change from the present system of immersing the filter leaves successively into pulp, barren solution, and wash water, consists in having the filters enclosed in a pressure tank into which pulp, barren solution, and wash water may be successively forced by gravity pressure. The vacuum may be applied to the tank to aid in filtration or the plant may be so arranged that gravity pressure alone will be used to effect the

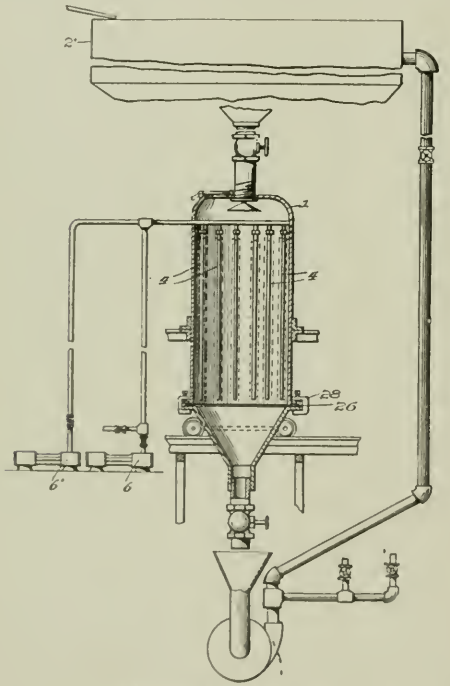


FIG. 1.—PRESSURE TANK SYSTEM.

filtration and subsequent washing of the cake. Two different modifications of the pressure-tank system are shown in the accompanying illustrations. Fig. 1 shows a vertical pressure tank (1) connected with a source of pulp supply (2) and with vacuum and pressure pumps (6 and 6'). Within the tank are filters (4) in the form of leaves or pipes so disposed as to give the maximum filtering surface. The bottom of the pressure tank is a conical member which is clamped to the pressure tank at 26 by a clamp 28. The conical bottom is on wheels and can be removed from the position shown. In operation pulp is admitted to the top of the tank and filtration commences at once. Uniformity in distributing is effected by the spreader 3. Filtration is allowed to continue until a sufficiently thick cake forms, as indicated by the dotted lines outside the filter leaves. The vacuum pump is then placed in operation, and the balance of the unfiltered pulp withdrawn from the conical bottom. Barren solution and wash water are then successively applied in the same manner. When the cake is to be discharged, the conical bottom of the tank is unclamped, wheeled out of position, and the vacuum is then released and the cake allowed to fall of its own weight or by the assistance of air pressure if needed.

Fig. 2 represents the application of the same idea in another form. In this case the pressure tank is stationary and the head with the filter is removable for discharge to the position shown

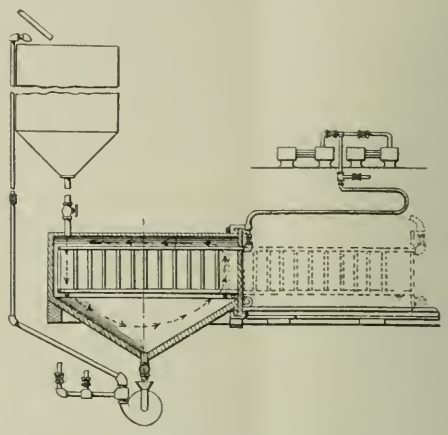


FIG. 2.—MODIFICATION OF FIG. 1.

by the dotted lines. It will also be noticed that the pulp in coming into the chamber will take the direction indicated by the arrows, thus maintaining the particles in uniform suspension during filtration. In still another form, not illustrated, provision is made for moving the tank away from the filter, which is stationary. (955,660-955,836, April 19, 1910.)

Improvements in the Cyanide Process.—To increase the efficiency and extend the scope of the cyanide process for gold and silver extraction is the object of certain methods of chemical procedure recently patented by Mr. John Collins Clancy, of New York. The invention is claimed to be unusually efficacious in the case of sulphide or other ores which have hitherto required roasting preliminary to cyanidation. The inventor claims that by the use of his methods this preliminary roasting will not be necessary. In addition to the use of potassium cyanide, Mr. Clancy uses potassium iodide and some strong oxidizing agent such as ammonium persulphate, ozone, or nitrogen peroxide. The effect is to form a cyanogen-iodine compound to dissolve the precious metals, the iodine being regenerated for use by the oxidizing agent used. The following proportions are said to be advantageous: 1 lb. potassium cyanide, ½ lb. potassium iodide, 3 lb. sodium persulphate, and ½ lb. lime, dissolved in 2000 lb. of water and used in the proportion of two parts of solution to one part of ore. If ozone is used, the quantity necessary must be determined in each case, but a proportion equivalent to 200 grams of ozone to 2000 lb. of solution may be used to advantage. The solution must be kept up the proper strength in cyanide and oxidizing agent, but the only additional iodide added is that required to compensate for ordinary mechanical losses. (955,318-9, April 19, 1910.)

Continuous Vacuum Filtration.—The latest suggestion for the continuous filtration of slimes is found in a machine recently patented by Cyrus Robinson, Mount Vernon, N. Y.

Briefly it consists of cylinders revolving in tanks of pulp and carrying over their surface a traveling belt filter medium, which acquires a thin coating of slime as it passes through the pulp tank and delivers a clean solution for precipitations. More specific description can be given by referring to the accompanying figures. Fig. 3 is a sectional elevation of the apparatus, in which 1 is a tank containing the slimes solution and 2 a tank containing water for washing the slime cake formed on the traveling filter. The drums 3 and 4 have a peculiar peripheral construction best shown in Fig. 4. Perforated wooden blocks 11 are fastened to the drum so that there are recesses 12 on the under side. Pipes 13 communicate with these recesses and with a valve chamber 14 which contains a stationary valve 15. It will be seen that this valve communicates with only that part of the traveling filter which is submerged in the pulp, and that as the drum revolves around the stationary valve, progressive communication is established by the pipes 13 between the chambers 12 and the valve port. If the valve chamber is then in connection with a vacuum pump, the solution from the tank will filter into this chamber, while a slime cake will be deposited on the traveling filter 7. The traveling filter has chain sprockets 8 on its edges which engage with the sprocket wheels on the edges of the drums. Provision is made to so adjust these sprocket wheels that the filter medium will always be tight and smooth.

The drum in tank 2 is similar to the one just described, except that there is a double valve whereby two successive washes may be applied to displace the solution held in the cake. The first wash would be strong enough to gold cyanide solution to be conducted to the precipitation boxes while the second wash would be weaker and could be used in such a manner as second best. The washed slime cake now travels, as indicated in the diagram, to a point where it is scraped off the filter by a brush 23, or other means so that the filter medium is again ready to receive another cake. In this manner the inventor seeks to continuously filter and wash a slime solution in a rapid and efficient manner. Suitable means of agitation are, of course, employed in the first tank. (954,466, April 12, 1910.)

Arsenical Ores.—Since the discovery of the Timiskaming district of Canada, metallurgists have given considerable attention to the metallurgy of the arsenical ores which have been

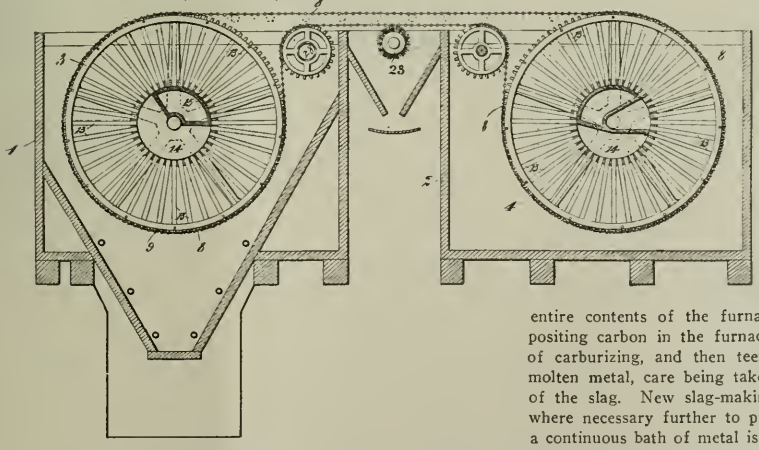


FIG. 3.—APPARATUS FOR CONTINUOUS FILTRATION OF SLIMES.

developed there. It is well known that when the silver ores carrying arsenic are roasted in the ordinary manner there is a heavy loss of silver due to volatilization with the arsenious acid (As_2O_3) produced in the roasting process and that this loss is especially heavy when the arsenic content is as high as 50 or 60 per cent. A patent has been granted to FREDERIC P. DEWEY, Washington, D. C., for a proces especialy designed to roast these ores without loss of silver. In the ordinary roasting process comparatively little arsenic acid (As_2O_5) is formed and but little attempt has been made to produce a large amount of arsenic acid in the roast. It is the object of the inventor's process to produce as much arsenic acid as possible and to avoid as far as possible the production of arsenious acid. The essential element of the process consists in maintaining a very low temperature during the roast. Arsenic begins to oxidize considerably below the melting point of lead and when this

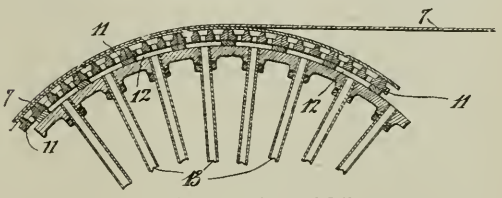


FIG. 4.—DRUM CONSTRUCTION.

oxidation sets in there is great danger of its developing too much heat for the successful conduct of the operation and it may be necessary to reduce the external heat in order to prevent the too rapid rise in temperature. The inventor gives a record of an experimental roast of an ore carrying 2700 ounces of silver, 58 per cent arsenic, 11 per cent cobalt and 4 per cent nickel. Starting with a cold furnace and a low gas flame arsenious acid began to come off in 17 minutes; in about an hour lead melted near the back of the charge and red spots began to appear in the ore. In order to keep the temperature down at this stage of the process the ore was stirred vigorously. At the end of about an hour and a quarter, the charge was very red and in another three-quarters of an hour the process was completed and the charge removed. At the close of the roast the heat was just high enough to cause zinc to thoroughly melt in front of the charge. By this process there was practically no loss of silver by volatilization, the assays showing a loss of less than 1 ounce per ton. On treating the roasted ore with sulphuric acid the arsenates and other compounds pass into solution, leaving the silver with the residue, which is practically free from cobalt and nickel, and can be recovered by various ordinary means. (954,263, April 5, 1910.)

Iron and Steel.

Purifying Steel.—A new process of manipulating steel in a basic open-hearth furnace is patented by Mr. REGINALD H. BULLEY (assigned to the Halcomb Steel Company, of Syracuse). The process consists essentially in pouring out the entire contents of the furnace, particularly all the slag, depositing carbon in the furnace to produce the desired degree of carburizing, and then teeming back into the furnace the molten metal, care being taken to prevent the return of any of the slag. New slag-making materials may then be added where necessary further to purify the metal. By this process a continuous bath of metal is maintained in the furnace, which is both kept at the lowest point in phosphorus and sulphur, and recarburized to any desired extent. The great difficulties and disadvantages of the old processes are overcome or avoided, particularly the danger of the metal getting into a superoxidized condition, with the attendant waste and difficulty of working; the tendency of the phosphorus to return from the slag into the metal; the difficulty of introducing carbon or

carbon compounds (carburizing materials) through the slag.

This process is stated to be particularly valuable where an open-hearth furnace is used in connection with an electric furnace: a continuous bath may be maintained in the open-hearth furnace and the electric furnace supplied at frequent intervals with comparatively small amounts of molten metal (of high carbon if desired). The open-hearth furnace is essentially an oxidizing furnace, where it is difficult to maintain a bath containing a high degree of carbon for any length of time and there is danger of the metal getting into what is known as the superoxidized condition. In that condition the carbon is, practically, entirely removed and an excess of oxygen is present; while a metal very low in phosphorus can be obtained in that superoxidized condition (for transfer to an electric furnace), yet there are certain serious objections thereto. The superoxidized metal becomes of a creamy, semi-viscous consistency, so that it is difficult to work and very difficult to teem out of the bottom of the ladle by means of the usual stopper and nozzle, which is desirable, to avoid the return of any slag; there is considerable loss occasioned by a portion of the metal adhering to the sides of the ladle. Another serious objection to the use of metal in this condition in the open-hearth furnace is the great loss, for a considerable portion of the metal oxidizes and passes into the slag. Also when the metal in the open-hearth furnace has reached this superoxidized condition it is very difficult, by the ordinary process, to recarburize the bath.

With Mr. Bulley's new process the transfer of the purified metal from the open-hearth into the electric furnace is easy, since the metal can readily be teemed out of a ladle by the use of the stopper and nozzle in the ordinary way, and by taking over in the ladle a greater quantity than is needed and stopping the teeming operation when the desired quantity is in the electric furnace, no slag will be introduced into the electric furnace. This is very important, because, if any slag is introduced into the electric furnace, any phosphorus in said slag will pass into the metal in that furnace. The excess steel remaining in the ladle can then be returned to the open-hearth furnace and the loss of metal in the operation is very slight. (955,378, April 19, 1910.)

Electric Furnaces.

Induction Furnace.—Messrs. E. A. A. Grönwall, A. R. Lindblad and O. Stalhane, of Ludvika, Sweden, patent the construction of an induction furnace shown in Fig. 5. 1 is the transformer core, 3 is the primary coil, and 4 the crucible or furnace groove, which is provided with a comparatively long projection. The following advantages are claimed for this arrangement: The smelter bath can be placed as near to the iron core as with regard to the heat, from a technical point of view, is possible without any risk of damage to the insulation of the primary coil. Further, the space through which the leaking lines of force can unite can be reduced to the least imaginable, as the primary coil and that part of the smelting bath that lies around the iron core receive diameters of about the same size. (953,030, March 29, 1910.)

Another similar design of the same inventors is shown in Fig. 6, which shows the connection of the induction furnace with a blast furnace. The material heated and reduced in the blast furnace drops in molten state directly into the secondary channel 5 of the induction furnace. 4 is the primary coil and 3 is the transformer coil. (953,029, March 29, 1910.)

Resistor Material.—The following resistor material for rheostats and the like is patented by Mr John T. H. Dempster, of the General Electric Company: 91.6 per cent iron, 2 tellurium, 6 silicon, and 0.4 manganese. This alloy is relatively cheap and has a sufficient resistance for use in rheostats and the like. It has high tensile strength and can be rolled and drawn without difficulty. (953,412, March 29, 1910.)

Cover for Electric Furnace.—To protect the outer portions of electric furnace electrodes against destruction by the hot furnace gases, Mr. Paul Girod, of Ugine, France, constructs the cover of the furnace as follows: The cover is formed of a metal ring, seated in refractory bricks and supporting one or more inner pieces of cast iron, which give passage to one or more electrodes, insulating refractory bricks

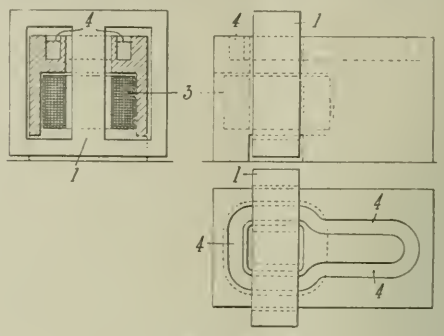


FIG. 5.—INDUCTION FURNACE.

being interposed between the iron pieces and the ring. When the electrodes are not of the same polarity the metal pieces, through which they pass, are independent of each other and insulated from each other by masonry. The electrodes are accurately fitted at their passage through the cast iron, so that the hot air and gases cannot pass through the electrode holes; for the exit of the gases special outlet holes are provided. In order to avoid fusion of the cast iron it is cooled by water circulation. But to prevent any danger of an explosion in case of a breakdown of the cast iron, the cooling water is passed

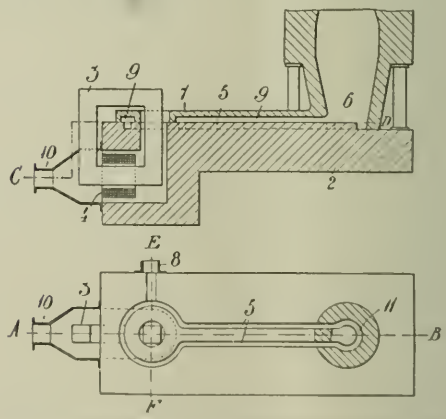


FIG. 6.—COMBINATION OF INDUCTION FURNACE WITH BLAST FURNACE.

through pipes which are mechanically independent of the cast iron. To make such a cover iron pipes are placed in a mold and iron is cast around the pipes, which then form a portion of the water circulation. Even should the cast iron burst, the pipes remain intact. (956,699, May 3, 1910.)

Furnace Construction.—A patent of Mr. Alois Helfenstein, of Vienna, Austria, relates to the furnace construction described and illustrated on page 46 of our January issue. The first claim refers to a "dome mounted over the furnace chamber, said dome carrying the electrode suspended into said furnace chamber, and means to prevent the escape of gases from said furnace, and a cone at the lower end of said dome." (949,895, Feb. 22, 1910.)

Electroplating.

Production of Conductive Layers on Ceramic Wares.—According to Mr. S. HELLER, of Teplitz, Austria, a conductive layer on ceramic wares (suitable for subsequent electrodeposition) is produced from a mixture of a solution of celluloid in an alcohol with tar, unsaturated hydrocarbons and gold-graphite. The mixture is applied to the ware and is burned in it in a muffle at a temperature of about 200° C. The gold-graphite is made by changing 3 grams of pure metallic gold to gold-protochloride, adding 500 grams of water and 10 grams of potassium cyanide, mixing this solution with 1 kg of chemically pure graphite, and burning the mixture in a muffle furnace at a temperature of 600° C. A mixture of 5 parts by weight of unsaturated hydrocarbons and 12 parts by weight of gold-graphite finely ground together in a color-grinding machine is particularly suitable. The layer formed is a good conductor and firmly adherent. (952,901, March 22, 1910.)

Plating Steel Tubes.—Mr. WILLIAM H. WINSLOW, of Chicago, Ill., patents a method of electric steel tubes, especially for small boilers (like automobile boilers), in order to produce tubes strong enough to stand the torsional strains and at the same time so thoroughly protected by the plating as not to be subject to corrosion. Copper is a particularly effective coating against corrosion. The resulting tube is practically a double tube, a steel tube surrounded by a copper tube, the two being practically one piece. The ends of the steel tubes are first closed by plugs of wood or other non-conducting material; conductors lead through the centers of the tubes and are bent within the tube. The tubes are then subjected to sand-blasting and then dipped into a dilute solution of copper sulphate, whereby a thin film of copper is deposited on the steel surface. The tubes are then electroplated in an alkaline plating solution. (The conductors through the end plugs forming the electric connection with the steel tubes, which are the cathode.) The electroplating is finally continued and concluded in an acid plating bath. (950,777, March 1, 1910.)

Wire-Plating Machine.—The plating of continuous lengths of such material as chain and barb wire offers some difficulties which Mr. GEORGE L. WALLACE, of Bridgeport, Conn., seeks to overcome in a machine which he has recently patented. It comprises a series of tanks, cleaners and polishers, through which several continuous lengths of material of the class mentioned can be automatically passed for plating. The metal is fed successively through a sand blast, a electro-cleaner tank, rinsing solution, plating solution, polishing apparatus and winding drums. All parts of a strip chain or barb wire are thus plated so that no ends or edges are left exposed to the action of the elements which would set up a galvanic action at those points, causing rapid deterioration. (954,833, April 12, 1910.)

Miscellaneous.

Down-Draft Furnaces.—With a view to economy in the process of smelting ores of ocopper and other metals, BERNARD DE ST. SEINE, of Paris, has patented a down-draft furnace with a single row of twyers placed above the smelting zone. The blast passes into the charge of mixed ore and fuel and the gases and the molten products issue below. This manner of operation is supposed to prevent the formation of undesirable deposits inside the furnace and to avoid the dust losses which ordinarily occur in an up-draft furnace. It is claimed, further, that hotter slags are obtained from which a separation of copper matte and such products can be more readily effected. Any form of solid fuel can be efficiently employed. (952,284, March 15, 1910.)

Hard Copper Alloy.—Messrs. JAMES NAULTY and JOHN SCANLIN, of Philadelphia, produce a copper-chromium alloy which is "approximately as hard as steel, thereby permitting edged tools to be made therefrom and adapting the metal for rolling hot or cold, welding and forge work." The copper and chromium are mixed in the proportion of nine to one. An alloy consisting of 45 per cent copper, 5 chromium and 50 zinc is also metioned in the patent. (959,156, May 24, 1910.)

Copper Alloys.—To produce tungstate of copper, Mr. GIDEON BOERICKE, of Philadelphia, Pa., lets sodium tungstate react with copper sulphate, forming sodium sulphate and a precipitate of tungstate of copper. In an analogous way chromate of copper is produced by the reaction between sodium chromate and copper sulphate, etc. (959,048, May 24, 1910.)

Reduction of Refractory Ores.—For the reduction of refractory ores, particularly oxides and sulphides, Mr. GIDEON BOERICKE, of Philadelphia, Pa., proceeds as follows: He takes a quantity of the refractory oxide to be reduced and places with it some manganese compound, as, for example, ferro-manganese, together with another metallic reducing agent, such, for example, as ferrosilicon—the said other reducing agent referred to being of such a character that the two reducing agents together in their oxidized form will serve the purpose of a flux. In some cases he adds carbon. The mixture is subjected to heat. (958,851, May 24, 1910.)

Graphic Recorder for Cooling Curves.

BY C. B. THWING, PH.D.

There are various methods in use for studying the transformations in materials in which heat is absorbed or liberated at various stages of the heating or cooling process. The instrument about to be described was designed especially for studying the cooling curves of iron and iron alloys. It can be modified in various details to make it more suitable for special problems.

The method consists in comparing the rate of cooling of an adjacent neutral body which may be assumed to follow the changes of temperature of the furnace quite closely. The instrument consists of two galvanometers mounted side by side, and recording on a single chart, which is carried forward by one clock. The needles are simultaneously depressed on the chart once in 10 seconds. The chart has two or more speeds, as may be desired.

One galvanometer is attached to a thermo-couple imbedded in close contact with the metal of the neutral body. The other galvanometer measures the e.m.f. of a series of short thermo-couples, one set of the junctions of which is imbedded in the neutral body; the other set of junctions similarly imbedded in the test piece.

Fig. 1 shows the thermo-couple in position for test. The

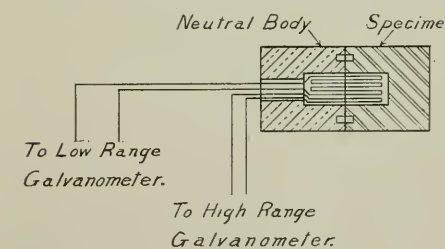


FIG. 1.—ARRANGEMENT OF THERMO-COUPLES.

size of the test piece need not exceed 1 in. in length and the same in diameter, with a cavity ⅜ in. in diameter and ½ in. deep. The galvanometers are sufficiently sensitive so that the one measuring the temperature of the test piece may be made to give full deflection for a range of a few hundred degrees only, while the galvanometer attached to the differential couple will give the full scale deflection for a difference of 20° below 400°, and for a difference of 15° between 400° and 1000° with three thermo-couples in series.

The specimen cooling curve shown in Fig. 2 was made with a specimen of tool steel as the test piece and copper as the neutral body. By placing resistance in series with the

galvanometer the sensibility may be reduced when the temperature differences are so large as to make it necessary. The

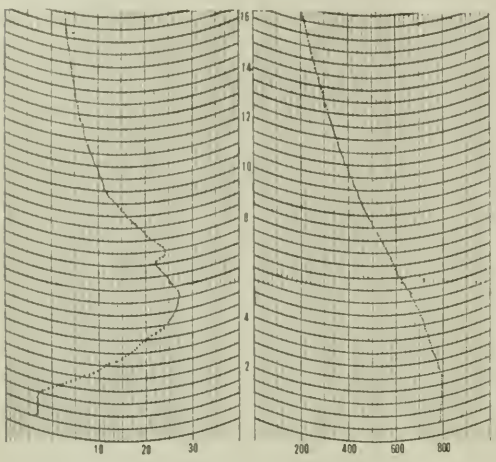


FIG. 2.—SPECIMEN COOLING CURVE.

zero of the galvanometer may be easily set to any part of the scale for convenience in taking heating or cooling curves, as may be desired.

A New Electric Steel Furnace Plant.

The Simonds Manufacturing Company, of Fitchburg, Mass., with branch factories at Chicago, Ill., and Montreal, Canada, makers of the well-known "Simonds" line of saws and machine knives, are installing in the crucible-steel department of their Chicago plant a Girod electric furnace.

The experimental plant of the Simonds Manufacturing Company is of special interest just at this time, as they have just acquired a tract of 60 odd acres of land at Lockport, N. Y., with excellent railroad and canal connections and ample electric power from Niagara Falls, which will make desirable and practical the melting of at least a portion of their steel by electric furnaces, should the rather rigid requirements of their quality be met by the electric process and furnace.

It is not the intention of the Simonds Manufacturing Company to do much, if any, refining of their higher grades of steel, but to employ the same raw materials, consisting largely of the best brands of Swedish melting bar, which they are now using, and are expecting their success in the application of electric melting to be entirely on the lines of uniform product, elimination of crucible troubles, etc., and a reduction in labor costs.

The new steel works at Lockport represent an increased investment of $250,000 to $300,000, not including their present very complete equipment, and the production from this new mill will be from two and one half to three times the capacity of the present mill at Chicago. Provision is being made at Lockport for the installing of more electric furnaces of a larger size than the one which they are now installing, should the expectations of the company in the success of the experimental Girod furnace be realized. This will probably be the first Girod electric furnace to be operated in the United States.

Pyrometer Laboratory.

At the recent Pittsburgh convention of the American Electrochemical Society numerous members used the opportunity to visit the laboratory of Mr. S. H. Stupakoff, which is particularly well equipped for the commercial testing of pyrometers and electrical measuring instruments.

One section of the laboratory is devoted especially to pyrometer work and the principal types of thermoelectric, resistance, and radiation instruments of 10 or more well-known manufacturers may be seen in operation. Another portion of the equipment is adapted for adjustment and repair of electrical measuring instruments for commercial and laboratory service.

FIG. 1.—MOTOR-GENERATOR SET

The sections for different classes of work are each centered about a table, and might be divided as follows:

Power supply, thermoelectric testing table, Fery radiation pyrometer, recording and indicating instruments, protection tubes, ammeter testing set, voltmeter table and potentiometer table.

Electric power is supplied from the mains of the Allegheny County Light Company at 220 volts single-phase and two-phase and direct current from the building power station at 110 volts.

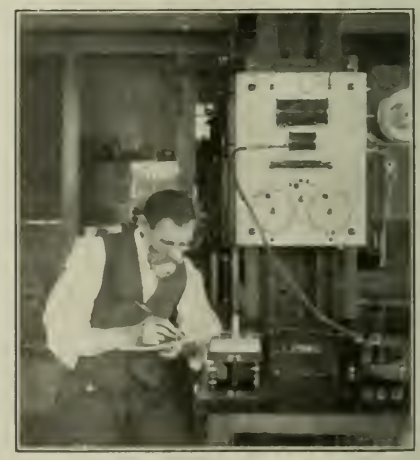


FIG. 2.—THERMO-ELECTRIC TESTING TABLE.

Motor-generator sets in the laboratory furnish direct current of high intensity at low voltage, and small voltage transformers step the alternating-current voltage down to 110 volts and up to 660 volts. One of the motor-generator sets is shown in Fig. 1. A Zucker & Levett & Loeb generator gives 10 amp to 720 amp at 6 volts and is belt driven by a 3-hp Westinghouse single-phase motor. Current for this is taken at 110 volts from the 220-volt line through a small transformer.

Another table with motor generator set supplies direct cur-

rents at varying voltages from 0 to 500 for the voltmeter testing outfit.

Fig. 2 shows the thermoelectric testing table. Above is a switchboard with three electric furnaces on front and one at top on the right-hand side. Two thermocouples can be seen inserted in the middle furnace. They are connected on the table to a Siemens & Halske precision galvanometer on the left and in the center to a Robert Paul duplex indication, which is provided with three scales, namely, a millivolt scale, one

FIG. 3.—FERY RADIATION PYROMETER.

temperature scale for platinum-rhodium thermo-couple, and one for base-metal couples. To the right stands a Meylan-D'Arsonval suspension galvanometer used in connection with the Fery pyrometer.

This arrangement is used to compare the needle deflections of instruments on test with the precision standard. The S. & H. galvanometer, which is of the suspension type, is mostly used as the standard of comparison. It is at frequent intervals checked by potentiometer.

At the lower portion of the marble switchboard are three

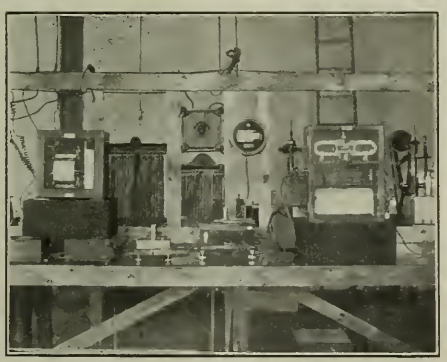


FIG. 4.—INDICATING AND RECORDING INSTRUMENTS.

sets of contacts, of which the one to the left is connected to an electric furnace in series with suitable resistance. This enables either one of the furnaces to be heated to any degree of temperature up to about 2500° Fahr. by shifting the center contact to its respective terminal. The lower, small tubular furnace, however, is used only at exceedingly low temperatures, never above 200°-250° Fahr. It carries in its interior a large number of thermo-couples connected up in series to the switchpoints arranged in a circle at the right side of the board. With this combination it is possible to tap thermoelectric currents in 20 different steps from, say, 1 to 20 millivolts, varying by 1 millivolt each, or any higher range up to perhaps 150 millivolts, in correspondingly larger steps.

This seems to be a very convenient way of commercially testing instruments of various makes.

To the extreme left one of three electric furnaces can be seen, which are mostly used for determining cooling curves, etc. The two larger furnaces are hidden behind Mr. George Schindler, who is in charge of the laboratory.

Just above his head hangs an experimental hot plate constructed of a high-resistance metallic ribbon. This apparatus is conveniently used for demonstrating the action of radiation pyrometers. In the condition as shown in the photograph it would give the radiation of a hot body in the open air. By placing it between two long muffles with both ends open a close approximation to black-body temperatures can be obtained. By this means it can be shown in an amazingly short space of time why radiation pyrometers should be used under conditions which approach as nearly as possible black-body conditions.

Most instruments of this nature are calibrated for black-body temperatures, and naturally their reading can be correct only when thus used. By simply removing the muffle mentioned and leaving the telescope of the pyrometer pointed to the heated plate an immediate drop of the indicator needle to 50 per cent or more can be observed. A simple explanation of this difference in the reading is that when a hot body is ex-

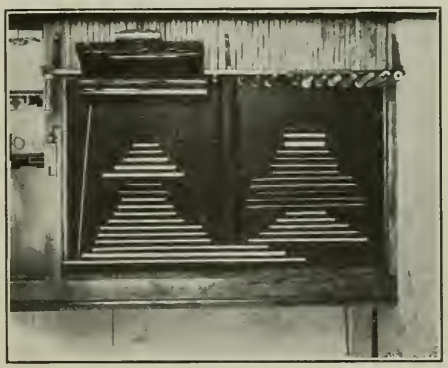


FIG. 5.—PROTECTION TUBES FOR PYROMETERS.

posed in the open air a portion of the heat which it radiates is lost into space and only a small portion of it can be collected by the mirror and concentrated upon the sensitive thermo-couple which forms the active portion of the instrument.

The telescope of the Fery radiation pyrometer mounted on its tripod is shown in Fig. 3. The radiation type of pyrometer has been designed for temperatures above those that can be measured by a thermoelectric couple on account of the deterioration of the latter at high temperatures. Base-metal couples are entirely reliable to about 1000 Fahr. and platinum-rhodium couples to about 3000° Fahr. Radiation pyrometers are suitable for any temperature up to the highest limit, but should preferably be used above red heat.

The gentleman seated at the table explaining the characteristics of the Marquardt-mass protection tube in his hand is Mr. S. S. Stupakoff, whose work as a consulting engineer on heat-measuring problems and apparatus has been well known for the past 15 years.

The Taylor Instrument Companies, of Rochester, N. Y., of which Mr. Stupakoff is the Pittsburgh representative, manufacture the Fery pyrometer, as well as the Foster fixed-focus pyrometer (which is described in detail elsewhere in this issue), and the standard line of "Tycos" thermoelectric pyrometers, of which a large selection is on exhibit in the laboratory. One

of the most commendable features of the exhibit is the arrangement which permits any one of the instruments to be shown in active operation on a few minutes' notice.

In Fig. 4 is shown a portion of the collection of various indicating and recording instruments. To the left, elevated, stands a thread recorded, and below two "Tycos" indicators, all made by the Taylor Instrument Companies, in Rochester, N. Y.

Immediately to the right is a Siemens & Halske precision galvanometer. Continuing, a S. & H. recording instrument, directly above a S. & H. wall-type pivoted galvanometer, and at the right side of the table a Leeds & Northrup recording instrument, which has been fully described in our columns.

Behind the latter instrument is to be seen a portion of a Whipple indicator. These last two instruments are exclusively used in connection with platinum-resistance pyrometers, as is also a Callender recording instrument, which is suspended on the wall to the right, but not shown in the cuts. The other instruments mentioned above are for use with electric pyrometers or radiation pyrometers.

Behind this table just described is a similar one extending the full length of the former shown in the cut, and at the extreme right hand of the same may be seen a Pellin thermo-electric recorder. Among several instruments upon this table,

FIG. 6.—AMMETER TESTING TABLE.

but not visible in the cut, are a selenarium, a number of various types of rare-metal and base-metal pyrometers and several Foster fixed-focus pyrometers.

While it is unnecessary to go here into a detailed description of the instruments it should be mentioned that all means were provided for testing them and seeing them in actual operation under conditions closely approximating those found in practice.

Fig. 5 shows the exhibit of protection tubes for the electric and resistance pyrometers, including the following types:

Marquardt-mass, Royal Berlin porcelain, electro-quartz, transparent silica, pure magnesia, nickel, and refractory fire-clays.

On the shelf are various types, lengths, and shapes of fire-clay, graphite and plumbago tubes and tips.

To the right below is a case containing a standard inspector's outfit Le Chatelier pyrometer, while above is a sample case of miscellaneous protection tubes.

A few of the superior class of Hohmann & Maurer "Tycos" industrial thermometers are fastened to the right of the blackboard. Below this on the floor are some tubes of electro-quartz and refractory clays of exceptionally large size. One of these is used in connection with the Fery pyrometer to obtain true black body radiation.

The sections of the laboratory devoted more particularly to testing of electrical measuring instruments, such as commercial service wattmeters, stationary instruments and laboratory apparatus, are shown in the remaining views.

Fig. 6 is the ammeter testing table. The meter on the small table to the left is a Weston laboratory standard with seven shunts ranging from 1 amp for full-scale deflection to 1000 amp. One side of the line is so arranged that the plus sides of all shunts are on the busbar and the current goes through the overhead rack, which forms the adjustable "load." The

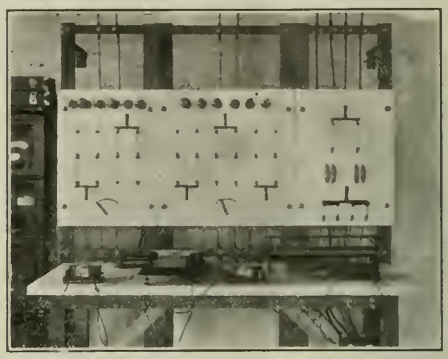


FIG. 7.—SWITCHBOARD OF VOLTMETER TABLE.

other sides terminate in a plug which can be inserted in the receptacles on the lower slab of the board for the necessary full-scale readings on the instrument when using any one of the shunts. The upper sets of receptacles serve to change the board from alternating current to direct current, or vice versa.

The small transformer under the table is a step-up from 5 amp to 100 amp. The two instruments on the table are undergoing repairs.

The switchboard of the voltmeter table, shown in Fig. 7, is so arranged that by using the circular switch with its contacts any voltage from 0 to 650 can be obtained, the finer graduations being secured by moving the sliding resistances seen on the table. The switching connections permit this voltage to be put upon the ammeter board when desired.

The second and fourth instruments on the table are Weston

FIG. 8.—POTENTIOMETER TABLE

laboratory standard voltmeters, direct current and alternating current, respectively, while the two smaller meters are in for test and calibration. The 1-to-6 step-up "Pittsburgh" voltage transformer is shown under the table at the right.

In the cases to the left of the voltmeter table are a large number of various instruments, mostly for use in connection with pyrometric measurements. At the corner of the marble table is to be seen a portion of a "Chauvin & Arnoux" record-

ing instrument. Above is a thermoelectric battery giving about 2½ volts by simply lighting the Bunsen burner, which is a part of the apparatus. On the top shelf sets a Maloney thermopile and behind it a Hartmann & Braun voltmeter, while on top of the glass cases are some carrying cases containing inspectors' outfits of Mr. Stupakoff's make and design. The balance of the shelves of these cases carry a collection of interesting apparatus, among others a variety of platinum-wound electric furnace tubes. It might here be mentioned that all of the electric furnaces described above are platinum wound, some of the Heræus type with platinum foil and others with platinum wire.

The potentiometer table with a Leeds & Northrup potentiometer and its resistance to the right are shown in Fig. 8. In the left foreground is a Kohlrausch bridge used for making resistance measurements. A Weston standard cell supplies the potential and a Leeds & Northrup D'Arsonval suspension reflecting galvanometer (rear view shown) completes the outfit. The meter to the right is on test.

The main switchboard is not shown. It has the line switches and fuses with the wattmeters for single and two-phase lines.

The laboratory, as at present described, is of comparatively recent date, and is even yet incomplete, several additional pyrometers and other instruments having been ordered, many of them from foreign lands.

A New Gas Producer Design.

The development of gas producers has been in the direction of increasing the amount of coal gasified per unit of producing capacity, and the reduction of the labor required in operating. Various devices have been introduced to accomplish this, such as mechanical poking and stirring of the fuel bed from the top, and continuous and even distribution of the coal in feeding, but little attention has been paid to devices by means of which the descent of the fuel bed in the producer is regulated and controlled, and to devices by means of which ashes are removed from the ash bed upon which the fuel bed rests, automatically and regularly.

All of these devices are embodied in the design and operation of the Forter-Trump gas producer, shown in Figs. 1 and 2.

The novel features of this design are the means of feeding the coal and removing the ashes continuously. The coal feed is accomplished by using the Trump knife mounted on a table, each half of which is of spiral shape. The knife and water-cooled table revolve at different rates of speed, thus delivering the coal onto the fuel bed, uniformly from the center to the circumference. In this design, the feeding knife is of sufficient depth and strength to prevent any clogging of the feed in case a large lump finds its way into the hopper.

The ash removal is accomplished by a revolving deflector or Trump knife, so designed as to give a radial thrust to the ashes. In operation, the table is stationary, and the knife is slowly revolved, cutting a section of the ashes out from the ash bed under the fire, and dropping the whole bed of fuel and ashes in pie-shaped segments onto the table, thus repeatedly dropping the bed, regularly breaking up the whole mass of incandescent fuel, preventing caking of the fuel bed and the formation of openings or holes through the fuel bed during operation.

The knives are tapered; that is, the thickness of the knife at the toe is smaller, and is larger at the heel; the knives can be used either singly or doubly, and operate to sweep out from under the column of ashes and coal a uniform section of the ashes, which may be varied by the taper of the knife from toe to heel. The movement of the knife causes the column of ashes and coal above to drop down uniformly over the whole area, and thus keeps the whole bed of coal in movement, leaving it porous and free from coke or cinder masses and without holes.

The fuel is fed continuously, and the ash knife may be run continuously at a slow speed of 20 or 40 minutes each hour; in this way, the fuel bed is kept level all around.

The air and steam necessary for gasification and partial combustion is delivered by two steam blowers into a circular wind box, which contains the Forter circumferential grates—these blowers are located on opposite sides of the box; a third steam blower delivers air and steam through a centrally located vertical tuyere with cone-shaped hood, to the center of the fuel bed. The Forter circumferential grate is inclined at a steep angle to the horizontal, thus allowing the incandescent fuel and the ashes to easily slide down on it.

A perfectly uniform distribution of air and steam over the whole area of the fuel bed can be effected by reason of this arrangement. The gas maker has the supply of air completely under his control, as he can force more or less air either to the center or circumference of the bosh, as the nature of the fuel or the condition of the gas may require.

The grate area is very large. As it does not depend on the inside dimensions of the producer, it can be made larger than on other producers which have the inlet of air only in the center. Access to the ash knife and table is easily obtained by doors placed conveniently on the shell, as shown on cut, Fig. 1, or by lifting the water-sealed shell, as shown in Fig. 2.

A 10-ft. producer of the Forter-Trump design will gasify 30 tons per day bituminous coal, nut size, and 25 tons per day of nut and pea coal mixed. With fine anthracite coal and with a 7-ft. diameter producer, the producer will gasify 7 tons of such coal per 24 hours.

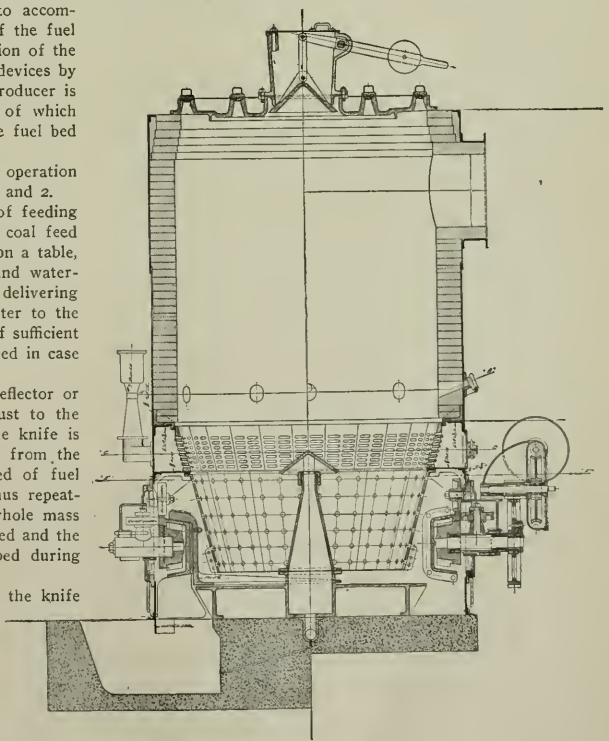


FIG. 1*a*.—HAND-FED PRODUCER, WITH AUTOMATIC ASH EXTRACTOR. VERTICAL SECTION.

When the gas is used hot for metallurgical purposes, the calorific efficiency of the producer plant would be 85 per cent, and if the gas is cooled to 75° Fahr., or 80° Fahr., the efficiency would be 80 per cent. The high efficiency is due to continuous coal feed and removal of the ashes which is clearly demonstrated by the analysis of the ashes which for long periods show only 35 per cent carbon, and are always either red or white, depending upon the kind of fuel in use. Tests of averages of the ash for a week at a time, show in one case 3.21 per cent carbon, and in another case 3.5 per cent, which, with producers of any other type, run from 7 to 13 per cent.

On account of the continuous removal of the ashes and the constant feed, the gas produced by the Trump producer is very uniform in quality and under the control of the operator. The maximum volume is obtained from any given coal on account of the operation of the producer, which fact is evidenced by the low carbon in the ash.

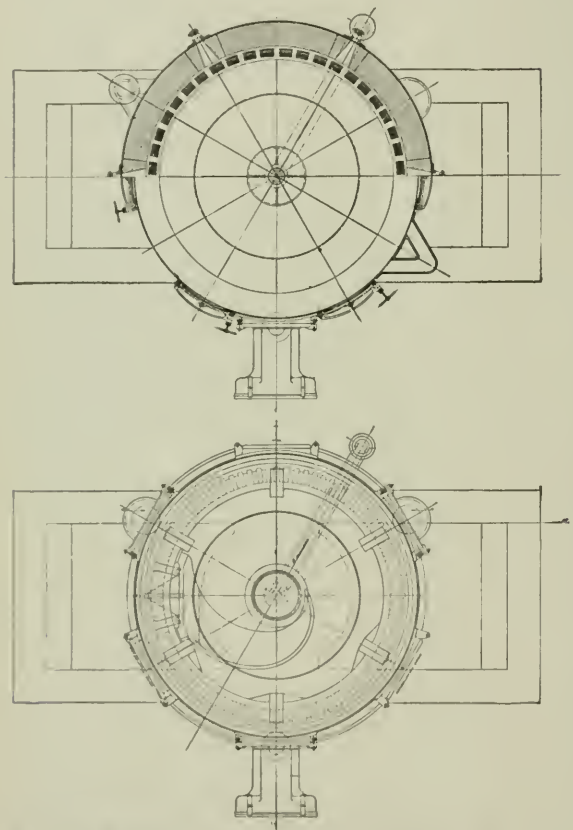


FIG. 1b.—HAND-FED PRODUCER, WITH AUTOMATIC ASH EXTRACTOR. CROSS-SECTION.

Eight producers of this type require the services of one man to supervise the operation on top and one man to take care of the removal of the ashes. These, with a foreman on each turn complete the force, where coal is delivered mechanically into the feeding apparatus.

The power required to revolve the knife has been measured and found to be for one revolution in two hours 2 hp as the knife is usually run, when run continuously, at one revolution in six hours, the power would be cut down to less than 1 hp.

Referring to the illustrations, Fig. 1 represents the Forter-Trump gas producer with ash pans, similar to the ordinary water-seal gas producer. The ash knife is attached to a cast-iron knife ring, geared on the inside. The knife ring is held in position by guide rolls on top, inside and the outside, and is placed above the table. The ashes are thrown off the table into a groove around the table, and from there are removed into the water-seal ash pans by two scoops attached to the revolving knife ring.

Fig. 2 represents the Forter-Trump producer with automatic feed, ash knife, ash hopper and water seal. The ash knife is attached to a knife ring with ball bearings below the table. The ashes are drawn off once a day into a bucket or suitable ash car. The shape of the ash knife is shown on the plans.

That this producer has passed the experimental stage is shown by the fact that several of them have been in use for over two years at two large industrial plants in this country. Its great advantages are to be found, as already stated, in the mechanical feed, which will handle any size of coal; and in the continuous removal of the ashes, which is accomplished by the Trump knife.

These producers are now being put on the market by the Forter-Miller Engineering Company, Hartje Building, Pittsburgh, Pa., who are prepared to contract for any size of installation required, and also contract for remodelling the ordinary water-seal gas producer.

Obituary.

Mr. **Townsend Wolcott**, for the past ten years the electrical inspector and electrical engineer at New York for the Signal Corps of the United States Army, died suddenly on April 29 of pneumonia, after an illness of but a few days. Mr. Wolcott, who was born in New York City on July 18, 1857, had won many sincere friends by his untiring activity for the best interests of his profession. Mr. Wolcott was an active member of the American Institute of Electrical Engineers, which he served in various capacities and by arduous committee work. He was a charter member of the American Electrochemical Society, and used to attend regularly the meetings of the New York section and to participate in the discussions. Aside from his engineering work, he was a mathematician and theoretical physicist of no mean ability. But his many friends will like to remember him best as a man of highest professional ideals.

Mr. **Walter Craig Kerr**, president of the Westinghouse, Church, Kerr & Company, died at Rochester, Minn., on May 8. He was born at St. Peter, Minn., on Nov. 8, 1858. He was an assistant professor at Cornell University until 1883, when he joined the Westinghouse Machine Company. In 1884 he became one of the organizers of the firm of Westinghouse, Church, Kerr & Company.

Pittsburgh Section American Chemical Society.

For the next months the following program of papers has been arranged by the Pittsburgh Section of the American Chemical Society:

June 16, 1910.—"The Chemistry of the Clay Industries," A. V. Bleininger, chief chemist, Ceramic Division, United States Geological Survey.

Sept. 15, 1910.—"Water-Softening Problems," J. C. W. Greth, chemist, W. B. Scaife & Sons Company.

Sept. 15, 1910.—"The Reduction of Iron by Carbon Monoxide," G. D. Chamberlain, chemist, Carnegie Steel Company

Oct. 20, 1910.—"Steel Manufacture," Prof. F. W. Crabtree, Carnegie Technical Schools.

Oct. 20, 1910.—"Chemistry and Chemical Technology of the Industry," D. M. Buck, chemist, American Sheet & Tin Plate Company.

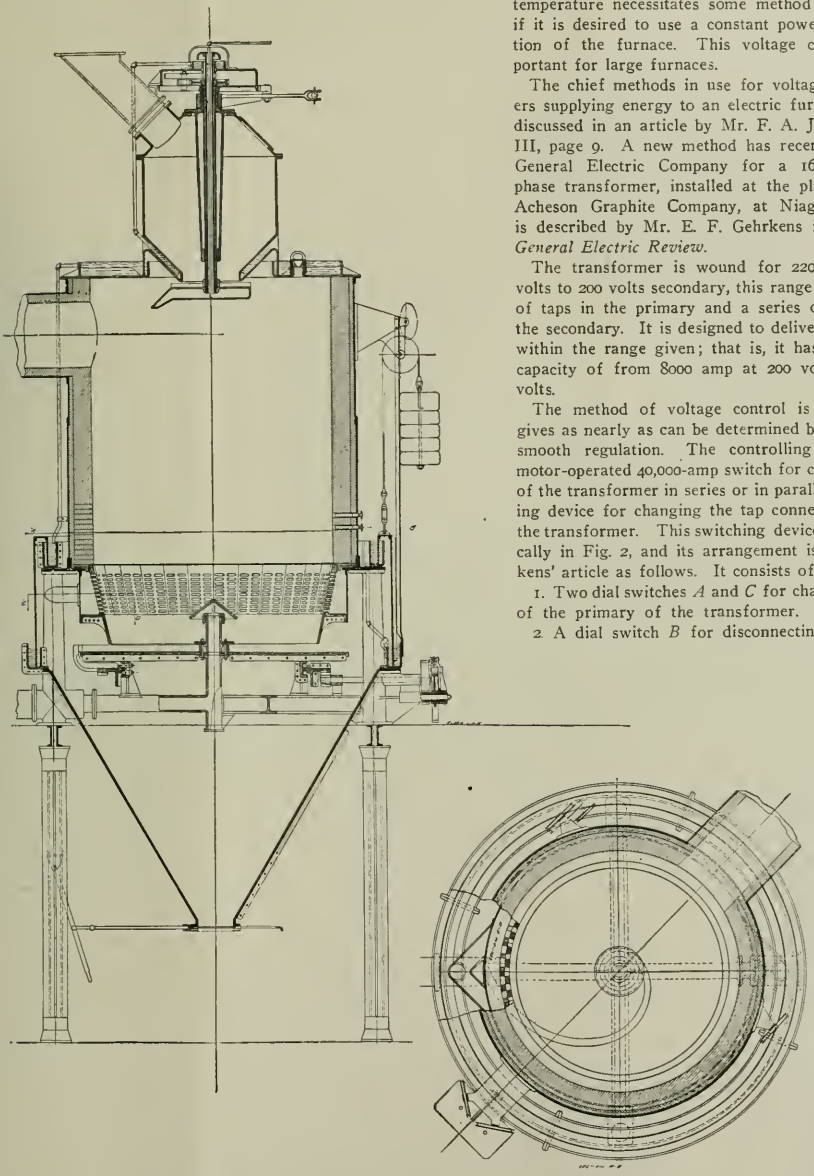


FIG. 2.—AUTOMATIC FEED, WITH AUTOMATIC ASH EXTRACTOR AND ASH HOPPER.

Nov. 17, 1910.—"The Analysis of Aluminium and Its Ores," E. Blough, chemist, Aluminum Company of America.

Dec. 15, 1910.—"Operation and Enforcement of the Food and Drugs Act of June 30, 1906," M. C. Albrech, chief of the United States Food and Drug Laboratory, Pittsburgh.

Voltage Control of Transformers for Electric Furnaces.

In most resistance furnaces, like the graphite or the carborundum furnace, the variation of the resistance with change of temperature necessitates some method of varying the voltage, if it is desired to use a constant power throughout the operation of the furnace. This voltage control is especially important for large furnaces.

The chief methods in use for voltage control of transformers supplying energy to an electric furnace were described and discussed in an article by Mr. F. A. J. FitzGerald in our Vol. III, page 9. A new method has recently been devised by the General Electric Company for a 1600-kw, 25-cycle, single-phase transformer, installed at the plant of the International Acheson Graphite Company, at Niagara Falls; this method is described by Mr. E. F. Gehrkens in a recent issue of the *General Electric Review*.

The transformer is wound for 2200 volts primary and 40 volts to 200 volts secondary, this range being obtained by means of taps in the primary and a series or parallel connection of the secondary. It is designed to deliver 1600 kw at any voltage within the range given; that is, it has a range in the current capacity of from 8000 amp at 200 volts to 40,000 amp at 40 volts.

The method of voltage control is exceedingly simple and gives as nearly as can be determined by a voltmeter a perfectly smooth regulation. The controlling apparatus comprises a motor-operated 40,000-amp switch for connecting the secondaries of the transformer in series or in parallel (Fig. 1) and a switching device for changing the tap connections of the primary of the transformer. This switching device is shown diagrammatically in Fig. 2, and its arrangement is described in Mr. Gehrkens' article as follows. It consists of the following parts:

1. Two dial switches *A* and *C* for changing the tap connections of the primary of the transformer.

2. A dial switch *B* for disconnecting the successive sections of the primary winding that have been cut out of circuit by dial switches *A* and *C*, so as to prevent excessive voltage across the transformer terminals which might occur as soon as the number of active turns of the primary windings are decreased.

3. A compensator, the winding of which is subdivided into a number of equal sections and arranged so that its terminals are consecutively connected across two adjacent taps of the transformer by means of the dial switches *A* and *C*; the terminals of this compensator being permanently connected to the moving arms of the two dial switches, respectively.

4. A cylinder switch with two sets of stationary collectors consisting of heavy bars, the two sets being arranged at right angles to each other and having the taps and the terminals of the compensators connected to one set, and individual resist-

ances to the other set. The moving member of the cylinder switch consists of a multiplication of fingers, individually insulated, by means of which the tap connection of the compensator is changed.

5. A two-part commutator controlling a contactor switch, by means of which the compensator circuit is opened as soon as the line is connected to a transformer tap direct and while the transformer tap connections are being changed by either dial switch *A* or *C*.

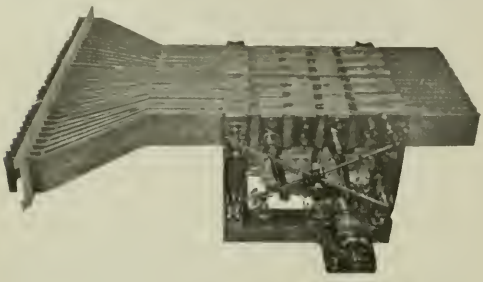


FIG. 1.—40,000-AMP MOTOR-OPERATED SWITCH FOR SECONDARY TRANSFORMER.

6. A voltmeter which is controlled by a rheostat and a potential transformer, and which indicates the voltage which will be obtained at the terminals of the furnace before the transformer is connected to the line.

7. An operating motor, shafts and gearing, by means of which the switches are operated; the motor being furnished with a reversing switch—so as to obtain a boosting or lowering of the transformer voltage — and a limit switch to prevent it operating the mechanism beyond the range of control.

The operating motor is geared to the shaft which, by intermittent gearing, actuates the dial switches *A* and *C* and the disconnecting switch *B*, and is connected to the cylinder switch by a crank The odd numbered taps of the transformer are connected to the dial switch *A*, and the even numbered ones to the dial switch *C*; the ratio of the intermittent gearing is such that the moving member of the cylinder switch (designated as *P*) passes over the entire set of compensator collectors from top to bottom, or vice versa, for every change of connection of either dial switch *A* or *C*. The blades of these dial switches—which are shown as *X* and *Y*, respectively—are of sufficient width, and the spacing of the stationary clips *a*, *c*, *e*, etc., on switch *A* and *b*, *d*, *f*, etc., on switch *C* is such that each blade remains in contact with each clip for an entire revolution of the crankshaft; that is, during the time necessary for the moving element *P* of the cylinder switch to travel over all of the contacts in both directions.

The crank operating the cylinder switch has an angular adjustment with reference to the blades *X* and *Y* of the dial switches, such that the change in the transformer tap connections of the dial switches takes place only when the crankshaft is on its centers, in which position the movement of *P* is exceedingly small. To compensate for this small movement both end collectors of the cylinder switch are made sufficiently wide to insure that all of the fingers of the moving element will remain on either end collector during the time required for changing the adjustment of either *X* or *Y* on switch *A* or *B*.

When the 2200-volt circuit is connected across the entire winding of the transformer, the minimum voltage will be secured across the secondary. This obtained with *X* on *a* and *P* connected to tap No. 1 of the compensator. With *X* on *a* *Y* is also connected to *b*, but practically no current flows through switch *C* with *P* on tap No. 1 of the compensator. The intermittent gear driving the dial-switch blades is inoperative with the blades in the position shown, with reference to the stationary clips *a* and *c* on switch *A*, *b* and *d* on switch *C*. The crank is just over the center and *P* is on tap No. 1 of the compensator. As the motor rotates, the contacts *X* and *Y* remain stationary, but *P* is moved along the horizontal collectors connected to the tap and terminals of the compensator, until it reaches the further extreme and the line is then, in effect, connected directly to the tap *b* of the main transformer. During this operation, the voltage of the secondary of the transformer has been gradually changed, because of the multiplication of the fingers on *P* and the large number of the taps on the compensator; the fingers *P* being arranged so that with the center of the top finger on the center of collector No. 3 of the cylinder switch, the center of the bottom finger is on the center

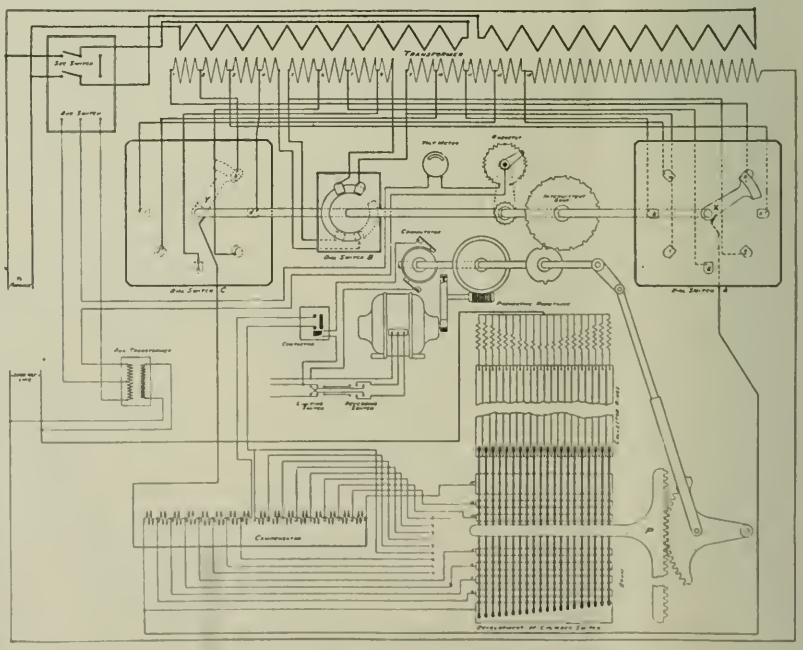


FIG. 2.—DIAGRAM OF CONNECTIONS FOR REGULATING DEVICE.

of collector No. 2 of the switch. As soon as P has reached the top collector of the cylinder switch, and the line is, therefore, connected direct to tap b of the transformer, the intermittent gearing engages and X is moved to c, Y still remaining on b. This changes the polarity of the compensator and again leaves P in a position such that by a continued movement of the crank the cylinder switch is again gradually changed between the next two transformer taps; that is, from b to c, thus again gradually increasing the secondary voltage of the main transformer. This alternating movement of the cylinder switch is continued until the entire range has been obtained.

With the secondary coils in multiple, the range in the secondary voltage of the transformer for a complete range of operation of the primary switch is from 40 volts to 100 volts. To obtain a higher voltage it is necessary to open the main primary switch of the transformer and entirely disconnect it from the line; then to connect the secondary of the transformer in series by means of the motor-operated switch, shown in Fig. 1. The operation of the primary switch must then be reversed until the voltage across the secondary is again 100, after which the motor must be again reversed and the operation continued as before until the primary switch again reaches its limit. A voltage of 200, which is the maximum, will then be delivered by the secondary of the transformer.

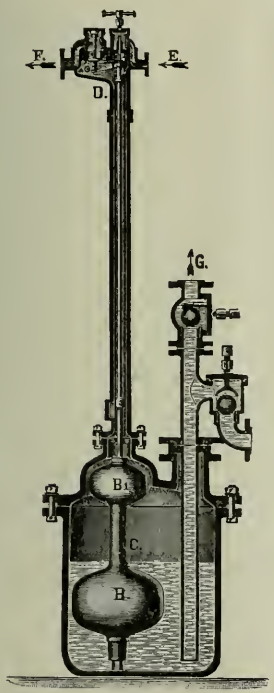


FIG. 1.—FLOAT-TYPE AUTOMATIC ACID EGG.

In order to enable the operator to determine the adjustment of the switch in regard to the secondary voltage, before the transformer is again connected into circuit, a voltmeter connected to the secondary of a small potential transformer is provided, the primary of the latter being connected directly across the line from which the main transformer is operated. This potential transformer has a double winding, each of 100 volts, arranged as shown in the diagram, and by means of a small switch (operated by and simultaneously with the series-parallel switch for the secondary of the main transformer), either the full-voltage or the half-voltage arrangement is connected to the voltmeter, depending on whether the secondary of the main transformer is in series or in multiple.

The current from the potential transformer passes through an adjustable resistance, operated by a sprocket wheel and chain from the shaft which actuates the dial switches. This resistance is such that the voltmeter shows the approximate voltage of the secondary of the main transformer; that is, it not only shows the voltage when the transformer is in operation, but also when the main transformer is entirely disconnected from the line, it will show what the secondary voltage would be with any primary adjustment or secondary connection, if the primary were connected to the line.

In order to avoid breaking any current whatever by means of the dial switches, a contactor switch is provided for this purpose. This contactor is excited from some suitable source of supply and controlled by a two-part commutator mounted on the main shaft. The brushes of the commutator are so adjustable that the contactor is normally energized, while the circuit is opened and thereby the commutator winding as soon as all of the fingers of P of the cylinder switch are on either extreme collector, and before either blade of either dial switch has opened the circuit. With this arrangement absolutely no current is broken by either of the dial switches A or C, the breaking of the magnetizing current of the compensator having been shifted to the contractor switch, the contacts of which are readily removable.

The burning of the contacts of P can be reduced to a minimum by having a large number of taps on the compensator, thereby reducing the voltage between the taps to a very small amount, and also by having a large number of fingers, so that there will be practically no breaking of the current, but rather a gradual shifting due to the gradually varied resistances between the compensator taps, which resistances have a sufficient capacity to allow the compensator switch to be left in any position whatever.

The arrangement described produces the same smooth and gradual regulation that is obtained with the induction regulator. The efficiency and power-factor of the installation are improved, and it is, therefore, particularly adapted for use with large single-phase low-voltage transformers having large current capacity and requiring a considerable range in voltage adjustment.

Automatic Acid Eggs and Gas Compressors.

Acid elevators and gas compressors play nowadays an important part in the equipment of a modern chemical works, and a short description of some new types should be of interest.

Fig. 1 shows Schütze's float-type automatic acid egg, which is built along the ordinary lines, but contains a few improvements in its general design. Its action is as follows:

The liquid flows by gravitation through a check valve into the elevator. During this period the dead weight of the double float C (consisting of two rigidly connected balls B and $B1$) acts on the lever D in such a manner that the compressed air (or steam) inlet E is kept closed, while the outlet valve F is open, allowing the air displaced by the liquid to escape. When the liquid reaches the upper part of ball B, the float rises, whereby air outlet F is closed, and the air inlet valve E opened. The compressed air then presses upon the surface of the liquid which is forced through delivery pipe G. When the surface of the liquid reaches the lower ball B, the float sinks, closing the air inlet valve. The compressed air expands until the egg is nearly emptied, but before the air can enter the delivery pipe, the air outlet valve F opens, and the cycle of operations is repeated.

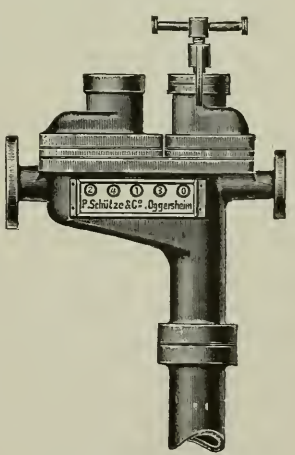


FIG. 2.—COUNTING APPARATUS.

The elevator performs from 30 to 60 operations per hour, according to the varying circumstances. As the quantity of liquid discharged at each operation is always the same, the daily or weekly output can be controlled by means of a counting apparatus, registering the number of pulsations, as shown in Fig. 2.

Owing to the fact that this elevator works with expansion, the air consumption is very economical. For instance, it takes 100 cu. ft. of air hourly under a pressure of 70 lb. per square inch to raise 1000 gal. of liquid per hour against a pressure of 45 lb.

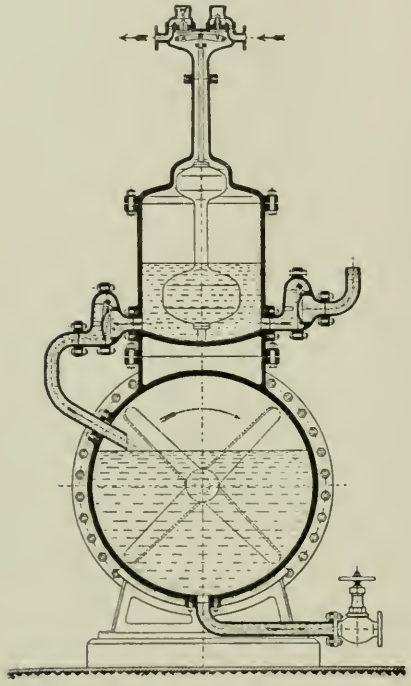


FIG. 3.—CONTINUOUS-FLOW ELEVATOR.

In many works engaged in the chemical industry, it is desirable, if not absolutely essential, to convey liquids to their place of destination in an uninterrupted stream, and for this purpose power-driven piston pumps, rotary pumps, injectors, etc., have hitherto been used.

All these appliances, however, are little, if at all, adapted for corrosive, muddy acid or caustic liquors, because the various working parts are quickly destroyed and the whole apparatus is, therefore, soon rendered useless.

Acid eggs or elevators, on the other hand, whether automatic or operated by hand, all work periodically and are, therefore, not adapted for the above purpose.

To fill this want, Schutze's continuous-flow elevator has been designed to furnish an apparatus embodying the qualities of both the pump and the acid egg.

This elevator, illustrated in Fig. 3, delivers the liquid in a regular uninterrupted stream without shock and is therefore particularly adapted for feeding filter presses, cooling and absorption towers, and other similar purposes. As it does not contain any parts subject to friction the power consumption is very economical.

For muddy liquids, and especially those used for feeding filter presses, these elevators are provided with agitators to prevent settling of the solid matter. The method of operation is as follows:

The liquid to be pumped flows through the inlet valve into the montejus, which forces it periodically into the receiver. Since the receiver is constantly under air pressure, the liquid is driven out in a continuous stream. When the montejus is empty the back pressure valve closes, and the inflow into the montejus commences afresh. A shut-off valve must be fitted in the discharge pipe of the receiver for regulating the outflow so that it does not exceed the amount of liquid supplied by the montejus.

Schütze's single and double-acting and compound gas compressors, as illustrated in Figs. 4, 5 and 6 respectively, are of a similar design and offer the advantages of having no stuffing boxes, piston or other working parts subject to friction.

The pump cylinder *E* alone comes in contact with the gas or vapor to be compressed, and can be supplied of earthenware, cast iron, wrought iron, etc.

Their construction is clearly shown in the accompanying cuts, and needs no further explanation.

An entirely new design is represented by Schütze's automatic acid elevator with electric valve gear.

By this arrangement all floats, bells, etc., in the interior of the acid egg are dispensed with, and the latter contains no

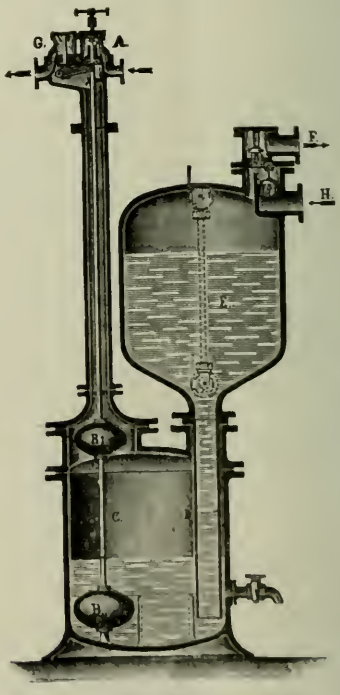


FIG. 4.—SINGLE ACTING GAS COMPRESSOR

movable parts, but only a set of contacts, which pass through a stuffing box and are insulated from the metal of the egg.

The working of the apparatus (Fig. 7) is as follows: The liquid is admitted by the valve *i*, and first reaches the outer contact tube *a*. When the egg is completely filled, the liquid reaches the external contact *c*, and completes an electric circuit. The electromagnet now becomes active and lifts the air inlet *d*. Compressed air is admitted, which closes the exhaust

valve *e*, and forces the liquid out of the egg into the rising main. As soon as the level of the liquid sinks below the edge of the contact tube *a*, the circuit is opened and the electromagnet releases the inlet valve, which closes and cuts off the supply of compressed air. The rest of the liquid and the compressed air contained in the egg escape through the rising main, the exhaust valve *e*, which slightly loaded by a deadweight *g* opens, and the egg again begins to fill.

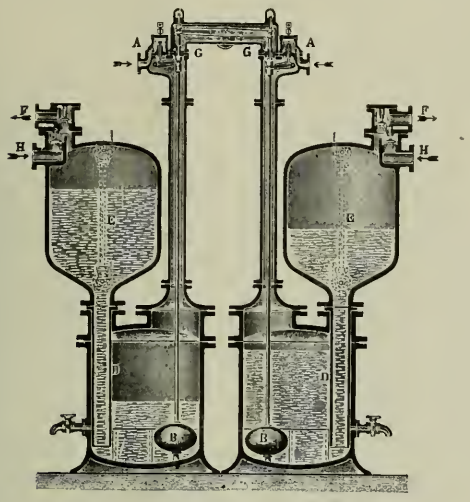


FIG. 5.—DOUBLE-ACTING GAS COMPRESSOR.

A brief explanation of the peculiar arrangement of contacts may be of interest. The tube *a* is connected to one terminal, while the rod *b*—which is, of course, insulated from *a*—and the rod *c* are connected to the other terminal. As the liquid fills the egg, is surrounds the tube *a*, without, however, rising into

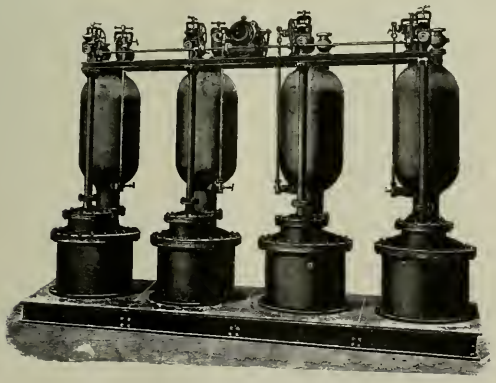


FIG. 6.—COMPOUND GAS COMPRESSOR.

it, as the tube is closed at the top. When it fills the egg it reaches the contact *c*, thus closing the circuit, actuating the electromagnet and admitting compressed air, as already explained. This forces the liquid up into the tube *a* and above the contact *b*, so that the circuit remains closed, although the level of the liquid sinks and leaves the contact *c* exposed. Only when the egg has emptied does the liquid blow out of the tube *a*; the circuit is opened and the compressed-air inlet valve *d* is released by the electromagnet and closes, as described above.

The simplicity of the apparatus and its absolute reliability in action are obvious. There are no floats, which must always be adapted to a certain specific gravity, and cause occasional trouble by sticking in even the best-designed apparatus. The valve gear is connected to the egg only by two wires, and may, therefore, be placed in any convenient position. Existing acid eggs may be transformed into automatic ones simply by the ad-

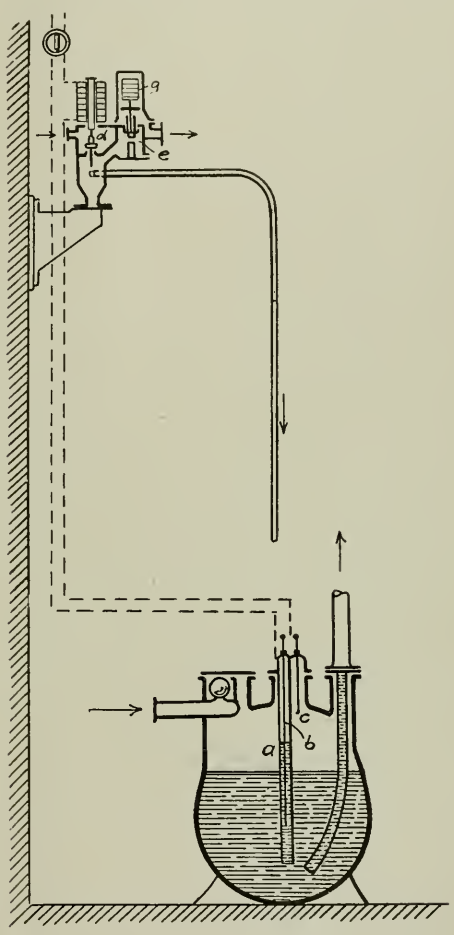


FIG. 7.—AUTOMATIC ACID ELEVATOR, WITH ELECTRIC VALVE GEAR.

dition of the valve gear and the insertion of the contacts. The latter, as well as the egg itself, can easily be made of material suitable for any liquid.

These automatic acid eggs and gas compressors of Schütze are introduced in this country by Mr. A. Innerhofer, 30 Church Street, New York, N. Y.

The New York Section of the American Chemical Society will hold a meeting on June 2. The program of the evening contains papers by Charles Baskerville and W. A. Hamor on the examination of ethyl ether, by M. A. Rosanoff on vapor pressures of binary mixtures, by F. J. Pond on iron corrosion, by A. E. Hill on the constancy of the solubility product, by H. C. Sherman and E. C. Kendall on amylases.

Constitution of Water.

A general discussion, of considerable theoretical interest, was held at the last meeting of the Faraday Society, in London, on the subject of the constitution of water.

Prof. P. Walden, of Riga, Russia, made the first contribution in the form of a paper entitled "Is Water an Electrolyte?" The paper included a number of observations that had been made in order to determine whether water when dissolved in a medium possessing powerful ionizing properties might not itself become an electrolyte.

To secure an adequate answer to this question it was considered necessary to make use of media of which the specific inductive capacity was greater than that of water. The liquids selected were hydrogen cyanide (HCN), formamide ($H{\cdot}CO{\cdot}NH_2$), nitrosodimethylamine ($(CH_3)_2N{\cdot}NO$), formic acid ($H{\cdot}CO{\cdot}OH$) and sulphuric acid, the first two solvents being characterized by a specific inductive capacity greater than 84, the value for water. In passing it may be noted that Professor Walden's discovery of the use of formamide as a solvent represents a "find" of extraordinary importance, which even if it stood alone would form an adequate recompense for the labor involved in his survey of the wide field of organic liquids; this solvent appears to reproduce nearly all the valuable qualities of water, including its convenient freezing point and boiling point and its powerful ionizing properties, but will mix freely with important groups of compounds which do not dissolve to any marked extent in water.

Hydrogen cyanide, which, with its high specific inductive capacity and great fluidity provides ideal conditions for electrolysis, was found when used as a solvent for water to give molecular conductivities of the order of 0.009 as contrasted with 300 for a salt such as potassium iodide; the low molecular conductivity of the water dissolved in hydrogen cyanide finds a parallel, however, in the low conductivity of hydrogen cyanide dissolved in water. Formamide, with a similar specific inductive capacity, but much smaller fluidity, gave for molecular conductivity of water values (about 0.016) only slightly higher than in the case of hydrogen cyanide. In the case of formic and sulphuric acids the specific inductive capacity is lower than that of water, sulphuric acid being further handicapped by its extraordinary viscosity, but both solvents gave increased values for the molecular conductivity of water, namely, about 0.17 in formic acid, but rising in the case of sulphuric acid as high as 74.

From these observations it is clear that the conductivity attributed to the water does not depend on the *physical* qualities of the solvent, but on some *chemical* relationship between solvent and solute. It was suggested by the author that water, acting as an amphoteric electrolyte, could form a salt when mixed either with a strong acid or with a strong base, and that the high conductivity of mixtures of sulphuric acid and water and the slight conductivity of mixtures of formic acid and water, were due neither to free acid nor to free water, but to the presence of an oxonium sulphate or formate in the liquid. The absence of conducting power in mixtures of water with hydrogen cyanide or formamide was attributed to the weakness of their acidic and basic qualities and the impossibility of combining them with water to form a salt-like electrolyte.

Prof. Ph. A. Guye, of Geneva, Switzerland, having given a résumé of the results of diverse researches concerning the phenomena of molecular association, conducted both in his laboratory at Geneva and in that of Prof. P. Dutoit at Lausanne, showed how the conception according to which association was a simple phenomenon of chemical polymerization permitted of our predicting various properties of associated liquids. He then indicated that it was even possible to calculate the order of magnitude of the association factor x in the liquid phase, if the degree of association or of dissociation were known in the vapor phase.

Mr W. R. Bonsfield and Dr T. M. Lowry then read a paper on "Liquid Water a Ternary Mixture; Solution-Volumes in Aqueous Solutions."

Mr. W. Sutherland sent a written communication to the discussion in which he brought forward the view that water of crystallization is solid hydrol in a special state. According to the writer's theory water vapor when nearly a perfect gas is hydrol, H_2O. Ice is trihydrol, $(H_2O)_3$. Water is a mixture of trihydrol and dihydrol, $(H_2O)_2$, in varying proportions, at ordinary temperatures containing about one part of the former to two of the latter. The formula $(H_2O)_3$ for ice explains its crystalline form, the hexagonal prism. This the author pictures, assuming the electron theory of valency, according to which tetravalent O is an atom associated with three negative and one positive electrons, the electron pair being available for the purpose of joining the three oxygen atoms together, while the two negative electrons attach it to two H atoms through their positive electrons. From the kinetic theory of gases it is calculated that such a trihydrol molecule has a density of 1.2. The density of ice, however, is 0.917, hence the hydrol molecule is larger than is supposed by the theory. In water of crystallization one has a form of solid hydrol; the physical properties of this are dealt with in an Appendix. The mean volume of H_2O in water of crystallization found by Clarke from 31 separate volumes is 13.76 cc for 18 grammes, giving a density of 1.31 for solid hydrol, compared with 0.917 for ice, or trihydrol.

The author next applies the hydrol theory to the consideration of solutions. He regards the ionization of all electrolytic solutions as complete at all strengths. The so-called degree of ionization is really the relative mobility of the ions; the original ionic theory ignored the electric forces acting between the ions and the ionizing forces keeping the ions apart. When the ions are moved past one another in solution by means of an e.m.f. two viscosities of electric origin are encountered, one due to the electric forces between the ions and the water molecules and the other due to neighboring ions. Thus the mobility of an ion is proportional to the cube root of the concentration; the hypothesis of chemical union between ion and solvent is as unnecessary as the theory of partial ionization. Nevertheless a term is required for the mutual energy of ion and solvent. This theory explains Walden's law on the dependence of ionic mobility on dielectric capacity and Ostwald's valency rule.

The effect of the solute on the solvent is also discussed. The contraction occurring when electrolytes are dissolved is due to their changing some of the trihydrol into dihydrol. A positive ion changes tri- into di-hydrol, a negative di- into trihydrol. To explain the apparent partial ionization in the case of weak organic acids, the author suggests that these consist of double molecules, some of which are dissociated into single molecules, these being completely ionized.

Prof. W. Nernst communicated a paper on "The Specific Heat of Ice, Water and Steam," which was read by Dr. W. T. M. Wilsmore. The survey covers the whole field from —200° C. to the highest temperatures. At low temperatures the specific heat is calculated by means of the thermo-dynamical theorem established by the writer. The variations in the specific heat of steam under high pressure are accounted for on the assumption of the polymerization $2H_2O = (H_2O)_2 + 2510$. The values for water vapor at very low temperatures are those deduced from kinetic theory. Attention is directed specially to the minima in the specific heat of water vapor under moderate pressures and of liquid water at moderate temperatures. Both minima are attributed to the dissociation of complex molecules.

Messrs. H. B. Baker, V. H. Veley, F. P. Seaton, W. R. Bonsfield, G. Senter, and W. P. Dreaper also participated in the discussion. Mr. Baker gave a striking experimental demonstration of the influence of purification in retarding the action of water on sodium amalgam. The conductivity of the water was a little lower than two reciprocal megohms. It is probable that non-conducting water, if it could be obtained, would be perfectly inactive chemically. Dr. Veley described similar experiments on the inactivity of highly purified nitric acid.

NOTES.

Dry sizing of ores for subsequent magnetic treatment can be successfully carried out in trommel screens as fine as 15 mesh where the conditions are right. If the mineral particles do not fracture in too sharp or very angular fragments, the trommels will not blind and will show good capacity. Some mixed sulphides of iron and zinc will show very good results by this method of sizing.

The Nissen Stamps operating on disseminated copper ores in a porphyry gangue show a capacity of between 9 and 10 tons per day per stamp, screening to 26 mesh. The stamp heads are 10 in. in diameter, weight 1500 lb., drop 105 times per minute with a height of drop of 6½ in.

Extension of the riffles of the Wilfley concentrating table frequently produces an effect which may be valuable under certain local conditions. With extra high riffles extending the full length of the bed of the table a rough concentrate can be produced which will increase the capacity of tables used later in the flow of the ore. This is the effect produced in the so-called Garfield table. Still other effects of more or less value can be produced by extending the riffles to the end of the table, but at a slight angle, say, 10° toward the side of the table. Such alterations are valuable only under special conditions and must be determined by experiment.

Removal of Scrap Iron and Steel from Ore.—The inevitable occurrence of small pieces of drill steel, nails and scrap iron of various kinds in ore delivered to the mill may cause considerable damage to the crushing plant if not removed. A very efficient way to remove such material is to use a magnetic feed sole just before the first crusher. The electromagnet can be made as strong as necessary to attract and hold pieces of steel as heavy as 8 lb. to 10 lb., without allowing them to be brushed off by the stream of ore. When the accumulation becomes sufficient to interfere seriously with the feed, the current can be cut off and the pieces of scrap removed. The quantity of material removed will often be surprisingly large.

The removal of wood from mill ore, prior to the finer stages of crushing and grinding, will contribute materially to the capacity of the plant. A simple device for accomplishing this has been installed in the concentrator of the Colusa Parrat Company at Butte, Mont., by Mr. Fred Pratt. After the first crushing and prior to entering the trommel system, the ore is elevated and discharged into a bin filled with water. The ore sinks through the water and is directed by the sloping sides of the bin to a point where it is removed by a bucket elevator. The wood floats and is discharged through an overflow launder or removed by an attendant. This simple device has been the means of greatly increasing the capacity of the concentrator.

The University of Montana senior class in mining engineering and metallurgy took its annual trip through Western mining and metallurgical centers last month. The party consisted of 30 men and was in charge of Prof. R. H. Bradford, of the department of metallurgy. In Montana the party studied the mining methods at Butte and the concentrating smelting and refining of copper at Anaconda. Lead smelting was studied at Helena and copper smelting at Great Falls. In the Cœur d'Alene district, Idaho, the lead mines and concentrating mills were visited.

Institute of Metals.—The (British) Institute of Metals will hold its annual autumn meeting in Glasgow on Sept. 21 and 22. A local committee, with Prof. Archibald Barr as chairman and Dr. Cecil H. Desch as secretary, has already been formed.

The Forest Products Laboratory at Madison, Wis., will be formally opened on June 4. It is a co-operative undertaking between the United States Department of Agriculture and the University of Wisconsin. This is perhaps the largest and best-equipped wood-testing laboratory in the world. It will be prepared to make tests on the strength and other properties of wood, to investigate the processes of treating timber to prevent destruction by decay and other causes, to study the saving of wood refuse by distillation processes, to examine the fiber of various woods for paper and other purposes, and to determine the influence of the miscroscopic structure of wood on its characteristics and properties.

The Griscom-Spencer Company of New York City, manufacturers of the Osborne blast separating and collecting apparatus and of other apparatus for chemical factories, have lately undertaken the designing of complete manufacturing equipment and the construction of factories for chemical products.

The Hardinge Conical Mill Company, of New York City, have sent us their new, well-written and neatly illustrated catalog, entitled "Advantages of the Hardinge Patent Mills Over Other Grinding Devices." The Miami Copper Company has ordered four Hardinge conical mills to be shipped to Globe, Ariz.; this is the fourth large copper company in the United States to adopt the Hardinge mill. The New Einasleigh Copper Company also have one en route to their plant in Australia. The Automatic Gold Separator Company will also use a Hardinge mill in their test plant at Los Angeles, Cal. A 6-ft. diameter sectionalized ball mill has been shipped by the Hardinge company to the Grand Union Mining Company, Choix, Mexico. Messrs. Frazer & Chalmers, of London, England, have been appointed special sales agents for the Hardinge mill in Africa and Great Britain.

The National Lead Company, of St. Louis, Mo., is making extensive changes and improvements in its plant there, which include an almost entire new power equipment. The contract for the apparatus has been awarded to Allis-Chalmers Company and includes two 20 x 42 heavy-duty Corliss engines. One of these will be direct-connected to a 250-kw, 240-volt, 110-r.p.m. generator and the other will be provided with a rope flywheel for driving shafting. In addition there will be a 100-kw, 240-volt generator for direct-connection to high-speed engine, 14 240-volt motors ranging from 5 hp to 50 hp, and a six-panel switchboard.

Silica Standards of Length.—The silica bar standards of length of the (British) National Physical Laboratory, mentioned on page 257 of our May issue, were made by the Silica Syndicate, Limited, London. By a mistake a different firm was named as manufacturer in our former notes.

Thermometers.—We have received from the Bristol Company, of Waterbury, Conn., their bulletin 125, describing Bristol's Class 1 self-contained recording thermometers for recording temperatures at the point where the instrument itself is. These are now being used to good advantage for recording temperatures in cold-storage plants. Some new models of Bristol thermometers are also illustrated.

Power Development Along Canal.—The old Ohio Canal is now serving a purpose never intended by its builders. Power sites are being developed along its banks and feeders, and the latest of these is one at Roscoe, controlled by the Coshocton Heat & Light Company. This company is largely increasing its power equipment and has recently installed a 750-kw Allis-Chalmers steam turbo-generator in its power house at Coshocton. It has now placed an order with the same company for a 1600-hp twin-horizontal open-flume hydraulic turbine direct-connected to a 1000-kva, 4000-volt, three-phase, 60-cycle alternator. The turbine is designed to operate under a head of 32 ft.

Buying Coal on Specifications and paying for the heat value contained has been growing in favor for the past year or two. The United States Government, being a very large consumer of coal for use in the navy and in public buildings, has quite naturally given the subject some attention. Bidders for a part of the government's coal supply specify a definite standard for their product, and this standard is taken into consideration in awarding the contract. Then any deliveries of fuel falling

below the specified standards are subject to a penalty, and similarly all shipments having a greater fuel value than that specified in the contract are entitled to a premium. The value of the fuel is, of course, determined by chemical and calorimetric tests. The United States Geological Survey has recently issued Bulletin 428, giving a statement of the factors affecting the value of coal, the methods for sampling and testing it, a form of specifications used, and a statement of the operation of the plan in 1908-9.

Steam Turbines.—A recent sales report of the Steam and electrical department of Allis-Chalmers Company shows how favorably the steam turbine has been received as a prime mover by users of power. A little over four years ago the company obtained the right to manufacture in this country the latest improved type of Parsons machine. The end of the first year saw only one unit in operation, but since then there has been a steady and rapid growth in sales, which has reached a total capacity of over 300,000 horse-power. Of this aggregate 48.2 per cent were sold during 1909 and 20.6 per cent have been placed on orders since January 1, 1910. A larger proportion of the units have been for use by power, lighting and railway companies and approximately 40 per cent of the total sales were for this purpose, textile manufacturers being next with 14 per cent of the machine. Practically every type of industrial enterprise is represented in the buyers of these turbines, including gas plants, cement mills, mines, lumber mills, flour mills, paper factories, tanneries, celluloid works, etc. In geographical distribution the machines are to be found in nearly every State in the Union and also in Canada and Mexico.

"High Vacuum Surface Condensers for Steam Turbines" is the title of Bulletin 106, just published by the Wheeler Condenser & Engineering Company, of Carteret, N. J. The high steam economy obtained by steam turbines with vacuums of 28 in. to 29 in. is pointed out, while on the other hand it is shown that in obtaining these high vacuums there should be no unnecessary power consumption by the condenser auxiliaries, or excessive amounts of water used for cooling purposes. This pamphlet describes arrangements of the tube surface, the use of rain plates and other means recently devised for obtaining these results. The construction of the Wheeler dry tube condenser is the outcome of investigations into the several factors affecting transmission of heat through condenser tubes, such as the material and cleanliness of the tubes, the velocity of water within the tube, the velocity of steam against the tube, the percentage of air in the steam surrounding the tube, and whether or not the steam side of the tube is flooded with water. Copies of this Bulletin, which is illustrated by numerous photographs and drawings, will be sent upon request by the Wheeler Condenser & Engineering Company, of Carteret, N. J.

PERSONAL.

Dr. **Richard Amberg**, formerly with the American Steel Foundries, of Indiana Harbor, Ind., has accepted a position with the Crucible Steel Company of America in connection with their plans for the erection of electric steel furnaces.

Mr. E. P. Mathewson, general manager of smelting operations for the Anaconda Copper Mining Co., of Anaconda, Mont., and the International Smelting & Refining Company, was in Salt Lake City early in May on a trip of inspection to the new Toole plant of the latter company.

Mr. I. J. Moltkehansen, who will be remembered by many American electrochemists from his stay in this country several years ago, and who was later connected with Mr Ashcroft in an electrolytic sodium factory in Norway, has opened a Comptoir de Chemie at 37 Boulevard de la Senne (near Gare du Nord), Brussels, Belgium, having bought an old-established business of chemicals and laboratory apparatus. He intends to make a specialty of representing American firms. He will also continue to act as consulting engineer for Norwegian waterfalls, being in contact with owners of from 2 hp to 300,000 hp.

Mr. Oscar Spitzer recently left Butte, Mont., en route to Australia, via Japan, on a business trip in the interest of Beer, Sondheimer & Company.

Mr. Henry N. Thomsen is manager of the Toole plant of the International Smelting & Refining Company at Toole, Utah.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ELECTRIC FURNACES (*Continued*).

No. 672,054, April 16, 1901, G. de Chalmot, Holcombs Rock, Va.

Arc or resistance type, especially for production of titanium and its alloys. The charge is placed in a crucible of carbon, covered by a plate. The crucible is supported in an outer chamber of firebrick by bars of carbon passing through the outer wallls and having beveled ends bearing against the crucible. The firebrick chamber is covered above and is vented by passages in its bottom. This chamber is carried by a truck. Fixed on the truck is a framework carrying a vertically adjustable carbon electrode which passes through the bottom of the chamber and up to or nearly to the bottom of the crucible. An electric current is passed successively through this electrode, the crucible and the support bars, generating heat either at points of imperfect contact between the crucible and electrode or bars, or springing an arc to the bottom of the crucible. Carbon monoxide or water gas may be introduced through one of the vent passages.

No. 672,067, April 16, 1901, F. J. Patten, New York City.

Arc type, especially for production of calcium carbide. The furnace has three flat open-hearths, sectors of a circle, pivoted on a central post. Each sector occupies an arc of 70° and rests on ball bearings. Pairs of electrodes depend from a swinging arm over each hearth, an arc being sprung between their ends and being electromagnetically deflected downward to the hearth. While the material on two hearths is being smelted, the third is discharged and recharged. The electrodes are radially adjustable over the hearths. Three arcs supplied with three-phase current may act on each charged hearth, or two-phase current may be used.

No. 672,601, April 23, 1901, O. Froelich, Steglitz, Germany.

Arc type, especially for calcium carbide. The furnace is a crucible with refractory lining. A tubular carbon electrode depends into the crucible, an arc being sprung between their lower ends. A bent discharge tube of iron extends from a tap-hole in the bottom of the crucible, and is normally closed either by a flexible end piece of asbestos hose or by a hinged cover. The iron tube is counterbalanced, and its upper end is heated by burning the gases from the upper tubular electrode in a casing surrounding it.

No. 676,575, June 18, 1901, Alfred H. Cowles, Cleveland, Ohio.

Water-cooled electrode holder for furnaces with gas reversing flues of prior patents. The holder is a vertical metal tube closed at both ends, the lower end having a socket for the electrode and the upper end carrying an adjusting screw. A small concentric inner tube forces a stream of water down against the socket end and the heated water escapes through an outlet pipe at the top. The furnace has a water-jacket, to which the other electric conductor is attached.

NEW BOOKS.

ARMES, ETHEL. Story of coal and iron in Alabama. 615 pages, illustrated. Price, $5. Birmingham, Ala.: Ethel Armes.

BLANCHARD, ARTH. ALPHONZO. Synthetic Chemistry. Second edition. Price, $1. New York: John Wiley & Sons.

BLANCK, F. CONRAD. Nitration of aniline and certain of its derivatives. Price, 40 cents. Baltimore: Johns Hopkins University.

BURCHARD, ERNEST F.; C. BUTTS AND E. C. ECKEL. Iron ores, fuels and fluxes of the Birmingham district, Alabama. 204 pages, illustrated. Washington, D. C.: U. S. Geological Survey.

CHESNEAU, GABRIEL. Theoretical principles of the methods of analytical chemistry based upon chemical reactions; authorized translation by Azariah T. Lincoln and D. Hobart Carnahan. Price, $1.75 net. New York: Macmillan Company.

CORDEIRO, F. JOAQUIN BARBOSA. The atmosphere; its characteristics and dynamics Price, $2.50. New York: Spon & Chamberlain.

CRAM, MARSHALL PERLEY. Fractionation of crude petroleum by capillary filtration. 50 pages, illustrated. Price, 50 cents. Baltimore: Johns Hopkins University.

ENNIS, W. DUANE. Linseed oil and other seed oils; industrial manual. Price, $4. New York: Van Nostrand Company.

GODFREY, HOLLIS. Laboratory manual to accompany Godfrey's "Elem. Chemistry." 36 cents. New York: Longmans, Green & Company.

GOLDINGHAM, ARTH. HUGH. Design and construction of oil engines, with full directions for erecting, testing, installing, etc. Third edition, revised and enlarged. Price, $2.50. New York: Spon & Chamberlain.

GREGORY, HERBERT ERNEST; KELLER, ALBERT GALLOWAY AND BISHOP, AVARD LONGLEY. Physical and commercial geography; study of certain controlling conditions of commerce. Price, $3. New York: Ginn & Company.

HANCOCK, EUG. T. Notes accompanying the lectures on geology applied to mining; course given at Michigan College of Mines. Price, $1. Lancaster, Pa.: New Era Printing Company.

JONES, HARRY CLARY. Introduction to physical chemistry. Price, $1.60 net. New York: Macmillan Company.

LEEDS, FR. H., AND BUTTERFIELD, W. J. ATKINSON. Acetylene: principles of its generation and use. Practical handbook on the production, purification, and subsequent treatment of acetylene for the development of light, heat and power. Second edition. Price, $2.50 net. Philadelphia: Lippincott.

MORDEN, GILBERT WALWORTH. Determination of thallium in the electrolytic way. 14 pages, illustrated. Philadelphia: University of Pennsylvania.

PURIFICATION OF WATER FOR BOILER FEEDING. Compiled by the Harrison Safety Boiler Works for the confidential information of its agents and employees. 105 pages. Philadelphia: Harrison Boiler Works.

RICHARDS, ELLEN HENRIETTA SWALLOW. Laboratory notes on industrial water analysis; survey course for engineers. Second edition revised. 59 pages. Price, 50 cents net. New York: John Wiley & Sons.

RIPPER, W. Heat Engines. (Being a new edition of "Steam.") Price, 90 cents net. New York: Longmans, Green & Company.

STUNTZ, STEPHEN CONRAD. Reference list on the electric fixation of atmospheric nitrogen and the use of calcium cyanamide and calcium nitrate on soils. 89 pages. Washington, D. C.: Department of Agriculture, Bureau of Soils.

UNITED STATES DEPARTMENT OF THE INTERIOR, U. S. GEOL. SURVEY. Colloid matter of clay and its measurement by Harrison Everett Ashley. 65 pages, illustrated. U. S. Supt. of Docs.

TAINTOR. C. WILSON. Brief analysis of the United States Steel Corporation, with chart of quarterly earnings since its organization. Price, $3. C. W. Taintor.

TAYLOR. ALEX. JA. WALLIS. Sugar machinery. Descriptive treatise devoted to the machinery and apparatus used in the manufacture of cane and beet sugars. Second edition, enlarged and extensively revised. 184 pages, illustrated. Price, $2 net. New York: Van Nostrand Company.

THOMPSON, SILVANUS P. Life of William Thomson, Baron Kelvin of Largs. Price, $7.50 net, boxed. New York: Macmillan Company.

TIBBALS, C. AUSTIN, JR. Study of tellurides. 421 pages, illustrated. Price, 25 cents. Madison, Wis.: University of Wisconsin.

WARREN, W. H. Engineering construction in steel and timber. Second edition. Price, $5 net. New York: Longmans, Green & Company.

WILLARD, G. H., AND CRAWSHAY, FRED DUANE. Pattern-making, core-making and molding. Price, $1. Chicago, Ill.: Popular Mechanics.

BOOK REVIEWS.

Report on the Iron Ore Deposits Along the Ottawa and Gatineau Rivers. By **Fritz Cirkel, M. E.** 147 pages, illustrated. (Series of bulletins on the mineral resources of Canada, issued under the direction of Dr. Eugene Haanel, Director of Mines.) Ottawa, Can.: Mines Branch, Department of Mines.

This is a report on an investigation of the iron ores of the valleys of the Ottawa and Gatineau Rivers. It is of special timely interest in view of the endeavors which are being made to establish an electric iron and steel industry in this region which possesses great water-powers.

After describing in detail various iron ore deposits in the townships of Hull, Templeton, Wakefield, Bristol, Grenville, and others, Mr. Cirkel concludes that many of these would yield ores which could in all probability be treated profitably in the electric furnace. It is stated in the report that this method of reducing the iron ores can compete with the blast furnace for the production of pig iron, if electrical energy can be developed at a low cost. With respect to the possibilities of the electrometallurgy of iron and steel the general conclusions on pages 100 to 107 are of special interest.

An appendix to the report gives a synopsis of the water powers, both developed and undeveloped, in the region under consideration. These data have been compiled from the latest authoritative sources available. The following falls can be developed to produce large quantities of power: Paugan Falls, Cascades, Chelsea Rapids, Great Falls, Coulonge River, Roche Fendue, Calumet Falls, Chats Falls.

The Mines Branch of the Department of Mines and Director Dr. Eugene Haanel must be congratulated on their untiring efforts to lay a solid foundation for a new Canadian industry.

* * *

Verhalten der Wichtigsten seltenen Erden zu Reagentien. Zum Gebrauch im Laboratorium. Von **Dr. Jos. v. Panayeff.** 83 pages. Price, marks 3.60 (retail price in New York, $1.15). Halle a. S.: Wilhelm Knapp.

This is apparently one of the series of German laboratory books for the chemical industries edited by L. Max Wohlgemuth and published by William Knapp, though this is not stated on the title page or cover. At any rate, the appearance of the little book is most timely. The author claims to have made many new observations regarding the behavior of these elements with reagents different from those before published and that in this respect his book differs from most other works on the subject in that he does not offer the results of the observations and investigations of others, but rather the

results of his own work, and only statements which he has, himself, confirmed.

No attempt is made in the book to lay down methods for separation of the metals from each other or from other better known elements, but attention is concentrated upon the individual reactions of each of the elements of the rare earths. These latter he defines to be the oxides of the cerite metals as well as of the ytterite metals so-called, to which must be sometimes added the oxides of zirconium and thorium, since these, in their behavior, exhibit many analogies with those previously named.

From the introduction to the booklet we learn that these rare earths are always found in association with other elements and are found not only in Sweden and Norway, where they were first obtained, but also in Brazil, Canada, Russia, and other countries. In addition to their occurrence in minerals they have been found in various plant ashes, for instance, in ashes of tobacco and barley and also in bones. He calls attention to the fact that these rare earths have become well-known earths, that they have become numerous and among them may be enumerated the cerite metals—cerium, prascodymium, neodymium, lanthanum, samarium and of the yttrium metals (also known as gadolinium metals) which furnish a large number, yttrium, erbium, terbium, ytterbium, scandium, disprosium, holmium, thulium, europium, gadolinium, and others. The author believes this list should be considerably extended because many of those here named are in all probability not simple bodies, but a mixture of various others yet unknown and very similar elements.

The system followed in the book is to set forth in each case: 1, the occurrence of the element in nature; 2, the principal characteristics of the element; 3, its combinations; 4, its reactions in the wet way; 4, its reactions in the dry way. The elements discussed in this way and the number of reactions described for each respectively, are as follows, in the order named: Zirconium, 24 wet reactions; thorium, 21 wet reactions; cerium, 24 wet reactions for cerous salts and 22 for ceric salts. Didymium—praseodymium and neodymium—19 wet reactions, 4 dry reactions; lanthanum, 20 wet reactions, no characteristic dry reactions offered.

Of the yterrite earths elements and their reactions, the following are treated respectively: Yttrium, 19 wet reactions, no dry reaction; erbium, 19 wet reactions, 2 dry reactions.

This little book is probably the only one in any language treating of these elements individually and collectively. It should certainly find a place in the library of every instruction laboratory, while for those specially interested in these rare earths and their study it will be invaluable.

* * *

Vectors and Vector Diagrams Applied to the Alternating-Current Circuit, with Examples of their Use in the Theory of Transformers and of Single and Polyphase Motors, etc. By **William Cramp**, M.I.E.E., Consulting Engineer and Special Lecturer on Electrical Design in the University of Manchester, and **Chas. F. Smith**, M.I.E.E., Assoc. M. Inst. C.E., Lecturer in Electrical Engineering, the University of Manchester, and Director of the Electrical Engineering Laboratories, Municipal High School of Technology, Manchester. Price, $2.50. New York: Longmans, Green & Company.

This book assumes that the reader has already a knowledge of alternating-current laws and of alternating current machinery. In the early chapters the vector idea is developed and also the principles of vector algebra, using the symbolical method of Steinmetz with slight alterations in symbology. The problems of the design and characteristics of alternating-current machinery are solved by the use of vector methods, though the vector diagram occupies a relatively unimportant place. The treatment is simple and clear and easily followed by one having the presupposed familiarity with the subject. The work covers quite completely the field of alternating-current machines and goes into more than the elementary details. It should lend itself particularly well to home study.

* * *

The Corrosion and Preservation of Iron and Steel. By **A. S. Cushman** (Assistant Director and Chemist in Charge of Physical and Chemical Investigations, Office of Public Roads, United States Department of Agriculture) and **H. D. Gardner** (Director, Scientific Section, United States Paint Manufacturers' Association). 8vo, 373 pages, 68 illustrations. Price, $4 net. New York: McGraw-Hill Book Company.

The senior author is also chairman of Committee U on Corrosion of Iron, and both authors members of Committee E on Preservative Coatings, American Society for Testing Materials. The work is very appropriately dedicated to Dr. C. B. Dudley, the lately deceased president of that society.

The book may be divided into two distinct sections, the experimental and theoretical part, and the practical part. The latter occupies Chapters VI to XII, and discusses most satisfactorily the preservative coatings for iron, such as with zinc, tin, copper, lead and other metals or alloys, with paints and pigments of various kinds, including the properties of paint vehicles.

This is, we think, by far the best part of the book, and is full of useful information intelligently discussed and practically set forth. This part of the book will be of interest and value to everyone having iron to protect from rusting under almost any conceivable condition.

The first five chapters are a mixture of observation, experiment and theory. As far as the observations and experiments are concerned, they are all interesting and well worthy of close study. In fact, the book contains many valuable records from which much insight into the problem can be obtained.

The authors classify the theories of corrosion as the "carbonic-acid" theory, the "peroxide" theory, and the "electrolytic" theory, and give their unqualified adhesion to the latter as the correct explanation. While not desiring to oppose the final conclusion to which the authors have arrived, the reviewer cannot refrain from noticing the partisan spirit which colors their interpretation of the observed facts. Many of their arguments are surprisingly weak, and really oppose rather than support their theory. For instance (page 110), "All substances in solution which contain hydrogen ions, such as acids, stimulate the corrosion of iron." But (page 111), "solutions of chromic acid . . . inhibit the rusting of iron."

The trouble here, and the chief defect of the authors' reasoning, seems too hasty generalization. The problems are so various, the corrosive agents so diversified, the conditions so multitudinous, the qualities of iron and steel so numerous that many causes of corrosion may be active even in one instance and to ascribe the whole effect to one cause is a risky conclusion. It is especially when the authors bring to their aid the assumptions of the theory of ionization in solution, and speak of various hypothetical ions as if they really existed and were directly under observation, that their explanations lose the merit of lucidity and pass into the field of the speculative and hypothetical. Their explanations then take on an apologetic tone, necessitated by the conflict between conclusions from their theory and the observed facts. The conflict is not between the "electrolytic" theory of corrosion *per se* and the facts, but between the facts and the language in which the electrolytic theory is expressed and set forth.

This explanation made, we commend the work highly to all interested in the facts and theory of the corrosion of iron, and particularly to those directly concerned in studying practical means of its prevention. The book is timely and will be of great service.

The paper, typography and press-work are exceptionally good. There is a long bibliography added.

Metallurgical and Chemical Engineering

A Consolidation of Electrochemical and Metallurgical Industry and Iron and Steel Magazine

Vol. VIII. New York, July, 1910 No. 7

PUBLISHED MONTHLY BY THE

Electrochemical Publishing Company

239 West 39th Street, New York.

Pittsburgh Office, 201 Oliver Building.
Western Office, 1510 Court Place, Denver, Colo.
European Office, Hastings House, Norfolk St., Strand, London, Eng.

J. W. Richards, Ph.D., President.
Curtis E. Whittlesey, Treasurer. E. F. Roeber, Secretary.
Telephone Call: 4700 Bryant.

E. F. Roeber, Ph.D., Editor.
H. C. Parmelee, Western Editor.

Yearly subscription price for United States, Mexico and United States dependencies, $2.00; for all other countries, $2.50 (European exchange, 10 shillings, 10 marks, 12.50 francs).

Copyright, 1910, by the Electrochemical Publishing Company.

Entered as Second-Class Matter at the Post Office at New York, N. Y., under the Act of Congress, March 3, 1879

New York, July, 1910.

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Dictionary of Metallurgical and Chemical Material.

At about the same time that this issue reaches our readers, they will receive the 1910 edition of our Dictionary of Metallurgical and Chemical Material. The first edition of this dictionary, issued last year, has found such a friendly welcome as to convince us of its usefulness for the many users and buyers of chemical machinery, appliances and material. The second edition has been completely revised and considerably enlarged. Our thanks are due to many friends for corrections and suggestions, all of which have found due consideration. For the future we ask again for the co-operation, by criticism or suggestion, of all who will use the little book in the task, often so troublesome, of looking for information "where to get things."

Bureau of Mines.

The final establishment of a Bureau of Mines in the Department of the Interior has occasioned some inquiry as to the relation of the Bureau to the metallurgical industry. Briefly, there is no provision for any work or investigation that would be of interest to metallurgists, and any such provision must be a development of the future. As finally approved on May 16, 1910, the bill provides for the establishment of the Bureau and for the appointment of a director "who shall be thoroughly equipped for the duties of said office by technical education and experience." The original appropriation suggested to carry out the work of the Bureau was $546,050, and this included an appropriation of $150,000 for the testing of structural materials, clays, cements, etc. The latter was, however, taken out of the bill in the House, and that phase of the work transferred to the Bureau of Standards. Unless it is restored in the Senate and finally agreed upon, the testing of materials will not be a function of the Bureau of Mines.

* * *

Practically speaking, the work of the Bureau of Mines will, for the present, be a continuation of the work of the technologic branch of the Geological Survey. An appropriation of $100,000 has been made for "the analyzing, testing and treatment of coals, lignites, ores, and other mineral fuel substances," which is one phase of the work that is of interest to engineers, and which was discontinued under the Geological Survey on account of lack of funds. Further appropriation of $310,000 is asked for the investigation of mine explosions, the safety of miners, the improvement of the conditions under which mining is carried on, the use of explosives and the prevention of accidents. This will also provide for the establishment of rescue stations where apparatus useful in entering mines in which disasters have occurred will be kept for use, and where experts in the use of these appliances will have headquarters for instructional purposes, and to render aid when called upon. Thus it will be seen that the coal-mining industry is the first and greatest beneficiary of the Bureau, and that the interests of metallurgy in the Bureau will depend on future developments.

Carbon Cells.

As the idea to devise a "carbon cell" continues to occupy the minds of inventors and investigators, in spite of the many failures of former attempts, a brief review of what is being attempted and what has actually been accomplished should not be out of place. What is the object of a carbon cell? We can make this clear by the perhaps unfamiliar yet striking analogy with Mr. Edison's iron-nickel cell. In the Edison accumulator the reaction during discharge is the reduction of a higher oxide of nickel to a lower oxide on one side and the oxidation of iron on the other side; the electrolyte, a solution of potassium hydroxide, simply serves as the carrier of the ions, so that the electrolytic reaction may go on, but the electrolyte itself is not changed at all, in the whole, during discharge. This particular nickel-iron accumulator is a special and the most successful example of a general type of cell to which the name "oxygen-lift cell" has been given. If M and R are the materials of the two electrodes, the general equation of the discharge of an oxygen-lift cell is $MO + R = M + RO$; during charge the reaction is reversed. Now, the ideal carbon cell, as it should be, is a special type of oxygen-lift cell; namely, a cell in which the material R is carbon. Its equation is $2MO + C = 2M + CO_2$. The problem is to find a suitable electrotrolyte and a suitable material M.

* * *

While the Edison cell is an accumulator to be discharged and recharged periodically, the carbon cell is essentially intended to be a primary cell for continuous discharge. That is, when the material MO is reduced to M during the discharge of the cell, the intention is to reoxidize the M continuously at once by means of new oxygen from the air. Such a cell will give out continually electrical energy, according to the free energy of the reaction $2MO + C = 2M + CO_2$, at the expense of carbon and atmospheric oxygen which combine to carbon dioxide. But in contradistinction to the combination of carbon and oxygen by direct combustion, the scheme of the carbon cell requires that the oxygen is not brought into contact with the carbon, but that the fresh supply of oxygen is made to the other electrode, the M electrode. The energy of combination of carbon and oxygen by direct combustion appears in form of heat, in electrochemical combination it appears directly as electrical energy. Hence the necessity of producing electrical energy by the roundabout way using steam engines and electric generators is avoided, with its necessarily very low efficiency, limited by the second principle of thermodynamics. We are not limited in this way in the carbon cell; in principle we could transform the whole free energy of the reaction into electrical energy. We thus would get several times the amount of electrical energy from a ton of carbon which we get now by roundabout methods. That this would mean an industrial revolution there can be no doubt.

* * *

The attractiveness of the proposition is evident. The scheme is undoubtedly sound in principle and theory. But to devise a theoretically sound scheme of an electrolytic reaction and to build in practice a cell which operates according to such a scheme are two very different things. Cells have been built which have been believed to realize more or less the scheme outlined above, and they have been called carbon cells and have been heralded as the dawn of a new civilization. But detailed investigation has always shown later on that they were not true carbon cells and nothing came of them. In two of the most notable examples fused sodium hydroxide and concentrated sulphuric acid have been used as electrolyte respectively, the former in the widely discussed Jacques cell, consisting of carbon and iron covered with oxide, in fused sodium hydroxide. Haber has shown in a paper before the International Electrical Congress in St. Louis, and his results have later been confirmed, that what primarily happens is a direct chemical reaction between carbon and sodium hydroxide, giving sodium carbonate and hydrogen; the cell then acts simply as a hydrogen-oxygen gas cell; the carbon acting as a "hydrogen electrode" and the iron as an "oxygen electrode." The reaction is, therefore, entirely different from the desired reaction of the imaginary carbon cell. The cell is of no practical value, because the expensive carbon electrode and the expensive sodium hydrate are changed into cheap soda, only to yield a little hydrogen which acts electromotively with the oxygen at the other end.

* * *

Junger has later built a "carbon cell" with concentrated sulphuric acid as electrolyte. Recently Taitelbaum has made quite extensive and interesting experiments along similar lines, in which he found vanadium salts to be particularly suitable in connection with the material M. Without entering here into a discussion of these researches, we will simply emphasize that here again the carbon electrode has been found not to act in the way desired. Hot concentrated sulphuric acid dissolves carbon, oxidizing it with formation of sulphur dioxide. The carbon electrode then acts as a sulphur dioxide gas electrode; and the desired carbon cell is in reality a sulphur dioxide-oxygen gas cell. Thus the result is again negative, as before. The difficulty would seem to be to find a suitable electrolyte which, just as in the Edison cell, acts simply as a carrier of the ions. So far it has not been found for the carbon cell. But this fact does not prove the impossibility of the scheme.

A Disadvantage of Reinforced Concrete Buildings.

The undeniable and important advantages of reinforced concrete for industrial buildings are now so widely recognized that we may call attention to the fact that under the peculiar conditions of industrial development in our times one of the strongest points of reinforced concrete may become a disadvantage—that is, its practical indestructibility. We no longer pretend to build "for eternity." We know we live in a transition era and the idea of scrapping good things for the sake of better ones has lost all terror for us and has even become familiar. In view of the constantly changing character of our industries, the lack of flexibility of a reinforced concrete structure, the difficulty of tearing it down may occasionally prove embarrassing; and we have an actual case from chemical engineering practice in mind in making these remarks.

Treatment of Slime.

More and more we tend to believe in fine grinding in the treatment of ores bearing precious metals, and the improved methods of handling slime successfully are enabling us to make better extractions, so that the percentage recovery of

gold and silver from an ore amenable, say, to the cyanide process, should be relatively high. On the other hand, the slime formed in the concentration of copper, lead, zinc and tin ores is the source of a great deal of trouble, and its treatment constitutes a problem that is not near solution in the customary mechanical concentration processes. While some of the devices in use produce high-grade concentrates in the final treatment of slime, they also have the defect of making high tailings. In lead and copper concentrating is this especially true, the vanner being the final concentrating machine, with retreatment of the tailings, in some instances, on buddles. Elsewhere in this issue our readers will find a paper by Mr. Walter McDermott which is a valuable contribution to the literature of slime concentration. It clarifies the atmosphere surrounding this problem and presents certain conclusions that are deduced from a consideration of the progress in concentrating methods during the past 30 years.

* * *

The author intentionally leaves out of consideration those methods which are based on principles other than gravitation. Looking into the future he believes that an advance will be made only by a concentrator which embodies a large and smooth surface, a regular settling motion which shall not jar the finest particles from the surface when settled, and the production of a clean concentrate without a middling. The question may be raised whether it is necessary to look for such a mechanical concentrator, when there are other forces which can be utilized to save fine mineral from slime. The various flotation schemes, which make use of the property of surface tension of the mineral particles, seem to meet with success, and suggest that perhaps the solution of the problem of slime concentration is to be found elsewhere than in mechanical concentration. The attempts to employ centrifugal force have not been very successful, although the idea is attractive; but in view of the fact that an extended trial of a centrifugal slime concentrator is soon to be made on copper ore slime by a large Western company, the last word cannot yet be spoken on this class of mechanism.

Available Iron Ores.

Two publications of widely different character bearing upon the iron ore resources of the world have recently appeared. One is a couple of ponderous volumes containing more than 1000 pages, with an atlas of 43 maps embodying the results of an enquiry edited by Mr. J. G. Anderson, general secretary of the International Geological Congress, which will meet at Stockholm later in the year, and the other is a popular article in the *Atlantic Monthly* by Dr. Henry M. Howe. The report for the Geological Congress aims to cover all that is known of ore deposits the world over, and tabulates the deposits, both as to tonnage of ore, as such, and as to tonnage of iron which may be regained therefrom. There is a great maze of figures, by far the worst we have ever encountered on this subject, and clearing away a mass of almost infinite detail, we reduce the figures to the lowest terms, to represent the tons of iron which may be regained:

Actual ore supplies.....$20 \times 10^9 +$
Potential ore supplies.....$10 \times 10^9 +$

Dr. Howe's article is intended to be a popular one. If Wall Street had not taken a hold upon the iron industry some years since it would not have been necessary for the Doctor to write the article. He tells the public what it needs to be told, that of iron ore there is no end, and that it is only a question of development of processes whereby apparently negligible amounts of iron may be regained. In Wall Street there was promoted the idea that there was no ore but Lake Superior ore. It was easier to frame up that idea than to explain that there was plenty more iron, but it would be much more expensive to isolate it. The question of how much available ore there is in the world is as indefinite a one as is the question what is the volume of a cone with a very obtuse point, neither the angle at the point nor the distance from the point at which the plane is to be drawn being given. Dr. Howe does not need to bother with the angle, for he puts the plane at what amounts to an infinite distance when he suggests that the earth itself is iron. With that conception we are not sure there is left anything for us mortals to stand on, since it is not certain that in gaining the iron there will be any slag produced!

* * *

These presentations are useful in their way, the one as an aid in the study of geology, the other to correct a popular misapprehension. The metallurgical world is not concerned with either, but with the relatively minute—from a mathematical standpoint—question of drawing the plane, very near the apex of the cone in the present state of the art, between the commercially available ores and those not commercially available. That successive generations will be able to draw the plane farther and farther down in the cone, moving it more rapidly than the wants of commerce expand, is perfectly obvious. The world has made, approximately, one billion tons of iron. All the knowledge we have now has been acquired during the manufacture of that tonnage. The ores at present easily available will give us, with no advance in methods of treatment, many times as much iron, but—how much more shall we know when we have made another billion tons? Along mechanical, rather than metallurgical, lines, a great deal of progress has been necessitated by the needs already encountered. The ores from which, centuries ago, wonderful grades of iron and steel were made by the Catalan forge, are not available to-day in commercial quantities, but there is still an enormous tonnage of ore which can be treated by the processes of a quarter or a half century ago. Along mechanical lines the Mesabi range has been the field of wonderful progress. Two decades ago the presence of ore there was not suspected. Suddenly large deposits were found, almost underneath the sod. Had it been necessary to strip 100 ft., or even 50 ft., of overburden to gain that ore it would have awaited a more convenient time, but to-day there are several stripping operations involving the removal of more than 100 ft. of overburden. Enormous investments are required, for, as a rule, a 40-acre or 80-acre tract will be stripped before any ore is mined. Thus capital is easily found for such advances as may be required. For the much more intricate problems of the future, involving the regaining of iron from very low-grade ores, the capital will no doubt be found, and the engineers and metallurgists will welcome the problems they will be asked to solve.

Niagara Falls Meeting of the American Institute of Chemical Engineers.

The second semi-annual meeting held at the Clifton Hotel, Niagara Falls, Canada, from June 22 to 25, was very successful. The weather was ideal and the attendance satisfactory.

In the opening session on Wednesday, June 22, the Institute was welcomed by Mr. A. Monroe Grier, K. C., of the Canadian Niagara Power Company, in a witty address in which he emphasized the ideal conditions of the locality for both work and play. The membership committee reported an increase of 17 for the past six months. The committee on medal reported specific plans for a permanent foundation for the awarding of a medal annually for the best paper presented during the year to the Institute. This medal will first be awarded at the next semi-annual meeting.

Action toward incorporation of the Institute was taken, to be consummated at the annual meeting. This will be held in New York during the early part of December. The amendment to the constitution to make ex-presidents ex-officio members of the Council for two years was favorably considered.

Mr. Edward R. Taylor, of Penn Yan, in a paper on The Changes in Industrial Chemistry Caused by Electricity called attention to the very extensive use which may be made of electricity in our municipal life, for instance, for water and air purification and sewage disposal, as well as an aid to agriculture, installations even on a small scale being feasible. Many ways were pointed out in which strictly chemical industries could be economically carried out by means of electricity.

A paper on the Corrosion of Iron and Steel and Its Prevention, by Mr. G. W. Thompson, of New York, read by Mr. Andrew Moyer, Jr., of New York, gave a summary of what is known at present on this important subject. As a general rule it was claimed that corrosion is not dependent on the purity of the steel so that the use of an expensive pure steel is not warranted. All the known causes of corrosion were given and methods of prevention in each case were outlined. Specific rules on cleaning of the steel and the paint to be applied were given.

Dr. F. G. Wiechman read a paper on "Protal: a new product for use in the arts." It is a new plastic which has many of the valuable properties of rubber and can be substituted for it in all cases where either soft or hard rubber is required. It can be moulded into any form and has the very decided advantage that protal does not deteriorate on account of oxidation. It is made from vegetable albumen, which for this purpose is best obtained from the seed of a palm which can be produced in any quantity. These seeds are used for the manufacture of buttons in which there is very great waste, which can be used in the manufacture of protal.

Protal can be compounded with rubber and a large variety of other resinous substances. Among these the most important is bakelite making protal-bakelite which is absolutely waterproof and unaffected by both cold and hot water. It resists the strong oxidizing influence of exposure for three months to strongly oxidized air.

Wednesday afternoon an excursion was made to the plant of the International Paper Company in Niagara Falls, N. Y. At this plant the wood mill was visited for both mechanical wood pulp and sulphite pulp, as well as the paper mill in which ordinary new paper is made. The capacity of the mill is 110 cords of wood per day.

The plants of the Canadian Niagara Power Company and of the Ontario Power Company were then visited.

The address of the president, Dr. Charles F. McKenna, on The Study of Materials as a Subject in a Course of Chemical Engineering was delivered on Wednesday evening. Dr. McKenna emphasized the great importance of a more careful study of the properties of materials. A masterly discussion of the principles of the inspection and testing of materials under practical conditions was given. A more detailed account of this address is reserved for the July issue of this journal.

The report of the committee on Chemical Engineering Education was then read by Dr. F. W. Frericus. The committee reported that the ordinary four years' course in chemical engineering as given in American universities and technical schools does not give sufficient training for technical work, but that at least five years' study would be necessary. The committee was also of the opinion that the courses offered in European technical schools were superior to those offered in American schools. The teaching staff of American schools is often too limited in number and deficient in practical experience.

CHARLES F. M'KENNA, PRESIDENT AMERICAN INSTITUTE OF CHEMICAL ENGINEERS.

The entire second day, Thursday, was devoted to visits to several important industries of Buffalo. A special car was chartered to take the members of the Institute and their guests from the hotel to the various works and return to the hotel. The attendance on these excursions was about 45.

At the plant of the Lackawanna Steel Company the large ore handling apparatus was first inspected. The blast furnaces and Bessemer converters, and the very economical system of utilization of the blast furnace gases proved very interesting. The hot gases were cooled by passing through the stoves used for heating the air blast, after which the dust was removed. The gases were then burned under boilers or utilized to drive gas engines of 2000 hp. The open-hearth furnaces were also visited.

At the plant of the Larkin Company a complimentary luncheon was served to the visitors, after which a portion of the plant was inspected. The great variety of operations carried out and the excellent organization of the plant was remarked by all.

The last plant visited was that of the Linde Air Products Company. Here the visitors found air liquefied on a large scale in a very well-designed plant. The nitrogen is allowed to evaporate leaving oxygen of 96 to 99 per cent purity. This is charged into cylinders and sold as compressed oxygen. About half of the oxygen of the air liquefied is obtained. The cold gases coming from the liquefied air are used to cool the compressed air before liquefaction.

On Thursday evening a dinner was held at Hotel Clifton. The president, Dr. Chas. F. McKenna, acted as toastmaster in his usual tactful and charming manner. Mr. J. A. DeCew, of Montreal, Canada, welcomed the Institute to Canada.

Dr. Edward G. Acheson said that as he was a resident of Canada and a citizen of the United States he felt that he also desired to welcome the Institute. He expressed the opinion that in the name "Chemical Engineer" the Institute had selected a name implying a profession which studied matter and its transformation both on a small and large scale and so was more fundamental in scope than any other profession.

M. de Chatelain, professor of the Polytechnic Institute of Russia and president of the Electrolytic Section of the Imperial Technical Society of Russia, said that he found America very much in advance of Russia in the application of electricity. He had come to America in order to study our methods of harnessing and utilizing water-powers for the generation of electricity.

Past-President Samuel P. Sadtler gave an outline of the growth of the Institute during the past six months.

Mr. A. Monroe Grier, K.C., of the Ontario Power Company, then made an eloquent address on the future of electricity

On Friday morning the remarkable growth of the chemical industries of Canada was outlined in a paper by J. A. DECEW, of Montreal, Canada. The greatest growth was shown to have taken place in the sugar industry and the distillation of wood and the utilization of the products.

A very comprehensive paper was read by Dr. OSCAR SNIDER, of the Western Electric Company, Chicago, Ill., on the manufacture and industrial application of Ozone.

The most recent development in this modern industry was given. The use of low voltages was declared to give a low yield as well as a low concentration. At least 10,000 volts must be used, while voltages as high as 40,000 are being used.

The application of ozone to the purification of water and especially of contaminated air was discussed at length. The use of ozone for the sterilization of a great variety of substances, as well as food products and as an oxidizing agent in chemical industries, was explained. Among these are the preservation of eggs and milk, bleaching, the cyanide method of extracting gold and the production of ferrocyanides.

"Some Problems in Chemical Industry," giving some theories of chemical action, was then read by J. T. BAKER. Dr. RALPH E. MEYER, of the Westinghouse Lamp Company, then presented a paper on commercial manipulation of refractory elements for incandescent-lamp purposes. The efficiency of the various materials for lamp purposes was discussed. Tungsten has been found to be the most practical material for this purpose. The method of testing tungsten ore for the extraction of the metal was given, as well as the method of forming the filaments for the lamps. In a similar manner the manipulation and properties of metallic molybdenum were described. This metal is well adapted for use as filament supports. The new method of using wire structure for lamps was explained. Lamps of this type having a candle-power varying from 12 cp to 500 cp were shown. Samples of all lamp materials in various stages of manufacture were shown.

On Friday afternoon the manufacture of graphite, aquadag and oildag at the plant of the Acheson Graphite Company was inspected. The remarkable manner in which the element carbon has been modified to suit a great variety of purposes was very suggestive of what can possibly be done with other elements.

At the evening session the papers by A. BEMENT, of Chicago, on the changes in coal due to storage; by SCHUYLER FRAZIER, on nitric and mixed acids, and by Dr. WM. M. GROSVENOR, on plant design, were read. Mr. Bement's paper gave the results of tests on coal stoves for three years.

Mr. Schuyler's and Dr. Grosvenor's papers were very suggestive and led to an interesting discussion. A very comprehensive paper on the subject "underground waters for manufacturing purposes," by WM. M. BOOTH, was read by title.

The total attendance at the meeting was about 75. The attendance at all the sessions and excursions was excellent and a marked increase in interest was shown as the meeting progressed.

San Francisco Meeting of the American Chemical Society.

The complete program, just issued, shows that the arrangements for the San Francisco meeting of the American Chemical Society have been made with utmost care. There can be no doubt that also the trip with the special train, which leaves Chicago on July 4 and arrives in San Francisco on July 12, will be greatly enjoyed by all who will participate. Headquarters in San Francisco will be at the St. Francis Hotel.

At the first session, to be held on July 13, the president, Dr. Wilder D. Bancroft, will present an illustrated lecture on positive photography. Papers will be presented by Dr. E. C. Franklin on liquid ammonia as a solvent and the ammonia system of acids, bases and salts; by Dr. W. F. Hillebrand on chemistry in the Bureau of Standards, and by Dr. H. E. Barnard on the use of sodium benzoate as a preservative of food.

In the afternoon an excursion will be made on the Ocean Shore Railway to Half Moon Bay and Lunitas. In the evening a smoker will be held at the Fairmount Hotel, where there will be a formal dinner for ladies at the St. Francis Hotel.

On Thursday, July 14, in the morning, sessions of the Industrial, Physical and Inorganic, and Organic Divisions will be held at the St. Francis Hotel. At this meeting the Industrial Division will hold its symposium on smelter smoke. This will be opened by the following three papers:

Mr. Charles Baskerville. "The Smoke Problem and the Community."

Mr. W. E. Ebaugh. "The Neutralization and Filtration of Smelter Smoke."

Mr. F. G. Cottrell. "The Electrical Precipitation of Suspended Matter."

Another paper has been promised by Mr. Edmond O'Neill. This will be followed by a general discussion.

Other sessions of the different divisions will be held in the mornings of July 15 and July 16.

Excursions have been arranged for the afternoon of Thursday, July 14, to Mount Tamalpias and the Muir woods; for Friday afternoon to the Selby Smelting & Lead Company, and a steamer trip around the Bay of San Francisco and out through the Golden Gate; for Sunday, July 17, to the Italian-Swiss colonies at Asti, and for Monday, July 18, an excursion on the Sacramento River.

For the entertainment of those who choose to return by way of Seattle, the Puget Sound Section has arranged a very attractive program.

As has already been noticed, the special train which is to take the Eastern members to San Francisco, will run from Chicago (July 4), via Colorado Springs, Grand Canyon of Arizona, Redlands, Riverside, Los Angeles and Santa Barbara; stopping at Adamana for a trip to the petrified forest and at Lanz for a visit to the borax mines. This train will arrive at San Francisco in the morning of July 12.

Dr. Wilder D. Bancroft is the president and Dr. Charles L. Parsons, Durham, N. H., is the secretary of the American Chemical Society. Mr. Ralph A. Gould is the chairman of the executive committee of the local arrangements committees for the San Francisco meeting. He will meet the party on the special train at the Grand Canyon as the representative of the California Section and will accompany them as they enter the State and until they reach San Francisco.

Iron and Steel Institute.

The autumn meeting of the Iron and Steel Institute will be held at Buxton from Sept. 26 to 30, 1910 (Monday to Friday).

On Monday, Sept. 26, there will be a reception, conversazione and concert, by invitation of the reception committee, the chairman of which is the president of the Institute, the Duke of Devonshire.

The first professional session will be held in the morning of Tuesday, Sept. 27, for the reading and discussion of papers. For the afternoon visits have been arranged to the works of the Midland Railway Company at Derby and to the Royal Crown Derby Porcelain Works.

On Wednesday, Sept. 28, a second professional session will be held in the morning, while in the afternoon visits will be paid to the Staveley Coal & Iron Company and the Broad Oaks Iron Works in Chesterfield. In the evening there will be a dinner at the Palace Hotel, in Buxton.

On Thursday, Sept. 29, an excursion will be made to the Crewe Works of the London & North Western Railway Company, and an alternative excursion to Manchester to visit the university, the Town Hall and Ryland's Library.

On Friday, Sept. 30, the Duke of Devonshire will invite the members and their ladies to a garden party at Chatsworth.

Mr. G. C. Lloyd, 28 Victoria Street, London, S. W., is the secretary of the Iron and Steel Institute.

The Western Metallurgical Field.

The Knudsen Furnace.

The report that the Knudsen furnace for copper matting is to be given a trial in Arizona has directed attention to a process of smelting that is not widely known in this country. Originally developed at the Sulitelma mines and reduction works in Norway, it has been in operation there for about five years, in units handling charges from 8 to 20 tons. The furnace is practically a basic lined converter into which a charge of fuel is dumped, followed by the charge to be smelted. A blast of air is turned on and the operation continued until a high-grade matte is formed.

The Sulitelma ore is a copper-iron sulphide, carrying about 15 per cent silica. It is mixed with silicious ore so that the charge smelted contains about 6 per cent copper, 36 per cent sulphur, 33 per cent iron, 4 per cent alumina and 21 per cent silica. In operating the furnace a charge of about 200 lb. of bituminous coal is charged and ignited, and a gentle blast turned on. The ore charge is then introduced and the blast continued at low pressure until the charge is molten, after which a full pressure of about 20 lb. is applied. Apparently any grade of matte can be produced according to the time allowed, and blister copper could probably be made. When the matte has been smelted to the desired grade the furnace is discharged into a settler from which the matte is tapped. It is stated that slags of low copper content, say, 0.5 per cent, are readily secured. The success of the furnace at Sulitelma will encourage a fair trail of the furnace in this country, especially in those districts where pyritic smelting would be an advantage.

Sale of the Clark Copper Interests.

The sale of the copper interests of Senator W. A. Clark to the Anaconda Copper Mining Company will give the latter practical control of the high-grade copper properties in the Butte district. This will result in the closing of the Butte Reduction Works as a copper producer. The 500 or 600 tons of ore formerly treated daily at this plant will be diverted to the Washoe plant at Anaconda. On account of the large capacity of the latter, this quantity of ore can be handled without difficulty in addition to the regular daily shipments from the Anaconda Company's mines.

By a slow process of elimination, due to one cause or another, Butte has gradually declined as a copper smelting center, although maintaining an eminent position as a copper mining district, and supplying ore to the two large reduction works at Anaconda and Great Falls. With the closing of the Butte Reduction Works there will be but one smelting plant left at Butte and the city will be practically free from the disagreeable effects of smelter smoke.

Basic-Lined Converters.

In the minds of some metallurgists the basic-lined Bessemer converter for copper matte is destined to supersede the one now generally used. These devices have been contemplated for some time and have been tried in several places. The Omaha plant of the A. S. & R. Company installed one about two years ago, but it is reported to have given poor satisfaction. Better reports, however, come from the Garfield Copper Company, Utah, where the basic-lined converter is in operation and giving sufficient satisfaction to warrant the erection of three more. Magnesite bricks of special shape are used in the lining and the life of the converter is sufficiently long to make the cost of converting compare favorably with the same operation in acid-lined converters. As in many other metallurgical operations, local conditions must play a large part in the advantage or disadvantage of this process.

Other suggested improvements in the bessemerizing process deal with the shape of the converter and the manner of applying the air blast. The suggestion of a "vortex" spherical converter is referred to at length in the Synopsis of Chemical and Metallurgical Literature in this issue. Mr. Haas, the inventor of this idea, also favors a basic-lined converter, although he does not advocate the use of magnesite brick.

Zinc.

The demand for zinc ores, which was rather brisk a short time ago, is falling off. A number of causes are contributing to this condition. Foremost is the plentiful supply, and in this factor the Butte district is exerting a strong influence. Apparently the future of Butte as a zinc producer is limited only by the desire of the operators to fill the demands made upon them. It is significant that in the transfer of the Clark interests to the Anaconda Company, Senator Clark retained the use of the zinc concentrator of the Butte Reduction Works, where the concentration of zinc ore is being carried on successfully. It is understood that the exploration and development work which has been done by Mr. Clark in the past few years will be continued until the zinc business becomes an important element in the future of Butte.

The smelting capacity of the zinc reduction works of the United States is not on the increase, and in fact there are a number of idle plants. In 1909 there were 30 plants in existence, with 211 furnaces and 90,550 retorts. Several of these plants are now idle and the number of retorts is reduced by at least 10,000.

Smelter Fume Disposal.

The different methods being adopted by various companies in disposing of smelter fumes without discharging into the atmosphere substances deleterious to vegetation are worthy of note. One of the first companies to observe the mandate of the law was the Ducktown Sulphur, Copper & Iron Company, of Isabella, Tenn. The chamber process of making sulphuric acid was installed there, and about 150 tons of acid are now made daily. Details of this plant and process are referred to in the Synopsis in this issue. The system of condensing the fumes by effecting a combination with zinc oxide or calcium oxide has been installed at the U. S. Smelting Company, at Midvale, Utah. This was reviewed in our April, 1910, issue (page 203). At the plant of the Selby Smelting & Lead Company, Selby, Cal., the Cottrell process of electric deposition is used, and the same process is being installed at the Balaklala smelter, Coram, Cal. Finally, at the Mammoth Copper Company's plant, at Kennet, Cal., there is being erected a structural steel baghouse for the condensation of furnace fume. This structure will contain about 3000 woolen bags, each 30 ft. long and 18 in. in diameter.

Recent history seems to indicate that the careless discharge of smelter fume into the atmosphere is rapidly becoming unpopular, and that the future will see but little or none of this practice. Different systems of disposal or utilization are available for plants operating under varying conditions, and undoubtedly one of these will be found adaptable at different plants.

Cripple Creek District.

The reported decision of the Colorado Springs & Cripple Creek District Railroad to advance rates of ore transportation from the Cripple Creek mines to the cyanide mills at Colorado Springs, has caused the Golden Cycle Company to build an aerial tramway from its mines to the mill. The equipment will cost about $200,000 and the plant will be in operation before the contract with the railroad company expires in April, 1911. This will place the Golden Cycle Company in an independent position, as it already owns mines and one of the best cyanide plants in the State.

Colorado School of Mines.

The thirty-sixth annual commencement exercises of the Colorado School of Mines was held in Golden, May 27, 1910. Pursuant to the custom, which has been followed for many years, of having an address by someone prominent in the mining or metallurgical world, the speaker on this occasion was.

Mr. A. A. Blow, mining engineer and a former trustee of the school.

The subject of Mr. Blow's address was "The Responsibility of a Mining Engineer." In the course of his remarks, Mr. Blow stated that one of the desirable qualities of a mining engineer is a rational and persistent optimism, which will give him self-confidence and enable him to discharge his duty to his employer with a full appreciation of his responsibility in the matter.

Mr. Blow regarded the experience of personal responsibility as the most valuable experience an engineer could acquire, and urged the students to accept such responsibility whenever it offered itself.

The Stoiber prizes, which are awarded annually to the students presenting the most meritorious theses in metallurgy, were received by R. M. Keeney and G. M. Lee, whose work was on the direct manufacture in the electric furnace of steels and ferro-alloys; and Jean McCallum and E. J. Bruderlin, who submitted a design for a dry chlorination plant.

Utah Society of Engineers.

At the May meeting of this society the following officers were installed for the ensuing year: President, O. A. Hannold, electrical engineer, Utah Light & Railway Company; first vice-president, C. H. Repath, chief engineer, International Smelting & Refining Company; second vice-president, F. E. Arnold, designing engineer, The J. J. Burke Company; secretary, W. C. Ebaugh, professor of chemistry, University of Utah; treasurer, B. F. Tibby, consulting engineer.

The Non-Ferrous Metal Market.

For some weeks past the non-ferrous metal market has been practically without feature.

Prices have not ranged widely, and such fluctuations as have been produced have been without material effect one way or the other.

Consumers generally seem to be following a hand-to-mouth policy, and to be satisfied with it, and it has been noticeable that most of the purchases have been for immediate consumption.

The impression prevails that manufacturers are not plentifully supplied with stocks, but they seem content to pursue their present policy.

Copper.—Lake has been steady and quiet at 12⅞ to 13c., only a moderate amount of business being transacted. With plenty of the metal for sale at these figures, and lower in some instances, there is no anxiety among the consumers. Reports have been current to the effect that the future production from the low-grade copper properties is to be effected at a low cost, and this, together with the fact that the world's visible supply of the metal is undoubtedly large, has contributed to a quiet market.

Lead.—The demand for this metal has been fairly good, and the market has been firm at 4.30 to 4.35c., New York, and 4.17 to 4.20c., St. Louis. Missouri lead has shown a specially strong demand.

Zinc.—The market has shown a tendency to decline under the slow demand for the metal that has been offered. Manufacturers are buying only for immediate needs. Prices have been hovering around 5c., St. Louis, with the customary advance for New York.

Tin.—The demand has been confined to the immediate needs of consumers and the market has not been marked by any very large sales.

The price has been fairly steady at about 33c.

Other Metals.—Aluminium is the only metal for which the demand continues strong. The market has been steady at 23¾ to 24c., and business is reported good. Prices for antimony are nominal in a dull market, at 7½ to 8½c. for various brands.

The Iron and Steel Market.

There was no material curtailment in pig-iron production in June, and the rate toward the close of the month was probably in excess of 28,000,000 tons annually, representing a decrease of but slightly more than 10 per cent from the maximum rate, which was 31,600,000 tons in February. The production in the twelvemonth ending with June may be estimated at 29,750,000 ± 150,000 tons, production in the second half of the twelvemonth probably exceeding that in the first half by about 200,000 tons. Three calendar years, 1906, 1907 and 1909, have shown outputs between 25,000,000 and 26,000,000 tons, but no calendar year has yet shown 26,000,000 tons, and no twelvemonth ending with June has hitherto shown as large a production as the 26,202,985 tons made in the twelvemonth ended June 30, 1907. Despite the fact that a slight further curtailment in output is likely in July and August, the prospect for a large increase in tonnage thereafter is such as to make chances substantially even whether the second half production will exceed or fall short of 15,000,000, tons, and as that was approximately the output in the first half, the prospects are for approximately 30,000,000 tons in the calendar year.

The persistence of a really very heavy production, which is certainly going into consumption, is remarkable, considering the entire reversal in sentiment which has occurred during the half year. Liberal forward buying and the accumulation of stocks by jobbers and manufacturing consumers have given way since late last year to curtailment of stocks and hand-to-mouth buying. Since the curtailment in production has been only about 10 per cent, it seems certain that the actual rate of ultimate consumption has been greater in the past two months than it was six months earlier.

The continued sagging tendency in prices and the general depressed tone of the market have been due not to a lack of tonnage considered absolutely or by comparison with the tonnages of previous years, but to its insufficiency as compared with capacity, which is represented, as to pig iron, by about 33,000,000 tons a year. Either the profits are still fairly large, in relation to actual capital investment, or there is a smaller spread between the best and worst positioned producers than would be assumed from a general survey of the industry.

During June the placing of new orders for finished steel products was very light, while specifications on old contracts were at a fairly satisfactory rate. Few branches of the finished steel trade reached July 1 with an accumulation of business sufficient to maintain even fairly full operations. As many contracts expired June 30, and are likely to be renewed quite leisurely, the volume of business in July promises to be very light.

A number of steel works with their attendant finishing departments are scheduled to stop July 1 for a fortnight or longer for inventory, repairs and accumulation of specifications. The iron mills of the Central West will have a general closing, their business being particularly light. The two labor organizations controlling most of the iron mills in the Central West, the Amalgamated Association and the United Sons of Vulcan, have demanded important wage advances for the year beginning July 1 and a positive agreement or disagreement on the scales will probably be deferred until August.

Pig Iron.

Minimum prices of pig iron at valley furnaces, 90 cents higher delivered Pittsburgh, are as follows: Bessemer, $15.50; basic, $14.50; No. 2 foundry, $14.35; forge, $13.60, representing declines during the month of 50 cents on Bessemer and basic, 65 cents on foundry and 90 cents on force. These lower prices are approximately $1 per ton below the cost of production to merchant furnaces which must buy their coke and ore, as most of them do, and this can be shown fairly conclusively by estimates of actual costs and by comparisons with previous low points, particularly in 1904 and 1909, when the striking of

higher prices than these, having regard to the difference in costs of wages, coke and ore, was followed promptly by advances. The situation is peculiar. There is a large profit in the Lake Superior ore, although some ore producers, working relatively expensive deposits, have much higher costs than those which made the region famous.

A few merchant furnace interests in the Central West have ore of their own, and can follow the pig iron market down still farther. The furnaces which must buy ore did not promptly withdraw from the market, having fairly large stocks in some cases, while apparently they were in doubt whether the market would ever react in their favor. While there is no prospect of any reduction in ore for this season, it would be absurd not to expect a moderate reduction for next season and it may not be long until this prospect will begin to influence a market which might otherwise advance.

After sales of Southern iron at $11.25, Birmingham, confidently reported in some quarters as amounting to 60,000 tons or more, the Southern market seems to be a trifle firmer, with prices held at $11.50, Birmingham, for prompt and third quarter, and $12 for fourth quarter. Buyers, however, are not quite ready to take hold at this range.

Steel.

Unfinished steel is slightly easier in tone and prices are quoted about 50 cents lower, as follows, f.o.b. maker's mill, Pittsburgh or Youngstown district: Bessemer billets, $25 to $25.50; sheet bars, $26 to $26.50; open-hearth billets, $27.50 to $28; sheet bars, $28.50 to $29; rods, $30.50 to $31.

Finished Products.

Plates and shapes have yielded fully a dollar a ton in price, and are quotable at not over $1.45, Pittsburgh, with occasional sales at $1.40. Indeed, in rare cases plates have been done at $1.37½. Sheets are slightly lower, and regular prices, as quoted below, are shaded about $2 a ton, except in the case of blue annealed, which are lower only by the disappearance of premiums over the regular price, for quick delivery. Steel bars have held up remarkably well.

Regular prices f.o.b. Pittsburgh are as follows:

Standard steel rails, $28 for Bessemer; $30 for open-hearth, f.o.b. mill, except Colorado.

Plates, $1.40 to $1.45 for tank quality, ¼-in. and heavier.

Shapes, $1.40 to $1.45 for I-beams and channels, 15-in. and under, zees and angles 2 x 3 and larger.

Steel bars, $1.45, base. Iron bars, $1.55, Pittsburgh; $1.45 to $1.50, Philadelphia; $1.45, Chicago.

Wire nails, $1.85, base; plain wire, $1.65, base; galvanized barbed wire, $2.15; these prices regularly shaded 5 cents per 100 lb.

Black sheets, 28-gage, $2.40; galvanized, $2.50; painted corrugated roofing, $1.70; galvanized, $3; blue annealed, 10-gage, $1.75. Tin plates, $3.60 for 100-lb. cokes.

Wakeman Luncheon.

A farewell luncheon was tendered at the Engineers' Club in New York to Mr. James M. Wakeman, retiring vice-president of the McGraw Publishing Company, on June 21, his birthday. On account of ill health Mr. Wakeman will shortly sail for Europe for a prolonged rest.

Some 130 friends of Mr. Wakeman were present. Mr. T. C. Martin was as graceful a toastmaster as ever. Speeches were made by Messrs. James H. McGraw, John A. Hill, Arthur Warren and Hugh M. Wilson. The McGraw Publishing Company presented to Mr. Wakeman a beautiful set of the works of his favorite author, Mark Twain, while the members of the advertising staffs presented him with a magnificent loving cup.

This journal is indebted to Mr. Wakeman for many a good advice, sincere criticism, and friendly encouragement, and in acknowledging this, joins his many friends in wishing him godspeed.

CORRESPONDENCE.

Titanium in the Steel Industry.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—I note in your June issue an abstract of a paper by C. E. Stromeyer, of Manchester, England, which purports to give some facts concerning titanium as an antidote for the brittleness of mild steel and its action in connection with phosphorus and other impurities. In the discussion of Mr. Stromeyer's paper, which followed in the Iron and Steel Institute, of London, his statements were torn to shreds by such well-known steel experts as E. H. Saniter, of Rotherham; Steel, Peech & Tozer, Ltd., of Sheffield; Hjalmar Braune, the celebated Swedish expert; Harbord, Twynam and others. Mr. Saniter, who is to receive the gold medal of the Institute this year for his efforts and researches in steel, stated among other things that "it would be noted in the investigation by all four of the above made by different men over a period of 30 years that the minimum of nitrogen found was 0.006 per cent. On referring to the nitrogen results in Mr. Stromeyer's paper it appears that the latter is again to be congratulated on the purity of his material, as 16 out of 20 steels (excluding the high phosphorus samples) showed less nitrogen than the minimum found by the other observers noted."

Now, lest we inadvertently do Mr. Stromeyer an injustice, let us reinstate his paper for a short discussion of its merits as relating to titanium and the elimination of nitrogen to some extent or to no extent as the facts may develop. Mr. Stromeyer makes the following remarkable statement: "It seems as if nitrogen when once it has entered the pig iron in the blast furnace cannot be removed by subsequent heating. The suitable chemical," he continues, "is believed to be titanium, but the addition of ferrotitanium alloy to steel has not demonstrated that this beneficial action takes place," and he goes on to show (sic) that he thinks titanium possibly does do the work of removing nitrogen, for in visiting the Leeds' Museum he found a specimen of titanium nitride.* Such indefatigable research scarcely requires an answer, so I quote another astonishing piece of information from this paper: "Of late, since the production of practically pure titanium has been affected, its addition to steel and cast iron has been attended with varying degrees of success, more particularly in removing blow holes."

On the strength (sic) of this glowing tribute to a new alloy he claims to have reduced the infinitesimal quantity of 0.0014 per cent of nitrogen down to the still smaller proportion of 0.0013 per cent by the addition of some "practically pure titanium." Now, this statement shows the futility of undertaking to answer a paper like Mr. Stromeyer's. Who can state that he had been able to determine the element of nitrogen in iron or steel down to 1-10,000 of 1 per cent? How can metallic titanium with its high melting point per se of, say, 5000° Fahr. affect molten iron or steel to any extent whatever?

To illustrate exactly what I mean to distinguish between the museum results, as quoted by Mr. Stromeyer, and the results actually obtained in the use of ferrotitanium in iron and steel, permit me to quote briefly from men whose names are not used except in connection with facts. Dr. H. Goldschmidt says in a paper read at the International Electric Congress, 1904: "Through the introduction of a small quantity of ferrotitanium which binds the nitrogen the composition of the bath undergoes a chemical improvement. Impurities, such as slag and others, are driven up and the bath is further purified. It is remarkable that the sulphur content is appreciably reduced."

E. von Maltitz, for years holding the position of metallurgical engineer with the Illinois Steel Company, says in *Stahl und Eisen*, No. 41, 1909: "In order to secure complete results it is necessary to employ not metallic titanium, but an alloy of

*Referring to the discovery inside the dismantled blast furnaces of the Farnley Iron Company of some crystals of titanium nitride, which are now in the Leeds Museum (p. 360 of June issue of this journal).

titanium and iron. By far the best results are obtained with an alloy containing 10 to 15 per cent titanium and no more. . . . The addition of this alloy is simple and meets with no difficulties."

P. Delville, in a paper read at the International Congress of Metallurgy, Liege, 1905, says: "The effects of titanium used as ferro in a cast-iron bath is to diminish considerably or to eliminate entirely the blow holes. This takes place by chemical action, reduction of the oxides dissolved, elimination of nitrogen," etc.

Dr. A. J. Rossi, the pioneer in titanium research and most indefatigable worker in its development, says: "I find that any steel or any iron which may have been improved and refined and strengthened with nickel or with chrome or with any other alloy may be still further purified and still further strengthened by the addition of a small quantity of ferrotitanium. Its limit as a strengthener in iron and steel has not yet been fully ascertained, but that it is the most powerful agent for strengthening and improving the baser metals is now beyond all cavil."

Dr. G. B. Waterhouse, the well-known metallurgist of the Lackawanna Steel Company, in *Iron Age*, June 2, 1910, says: "It has been his (Waterhouse) good fortune to be in the closest touch with the manufacture of over 140,000 tons of steel, in the making of which 1,310,000 lb. of ferrotitanium alloy has been used. The most suitable contents of titanium in this alloy have been found to be from 10 to 14 per cent. . . . Using an alloy with less than this percentage would not give any further advantage. As to the amount of nitrogen removed from the metal by the use of the alloy conclusive and definite experiments have not been made. One day the iron gave 0.015 per cent nitrogen and the finished steel 0.007 per cent; another day 0.021 per cent and 0.014 per cent respectively, and on still another day 0.015 per cent and 0.014 per cent. What evidence there is, therefore, points to improvement in this respect . . . and it has been found that the steel treated with the alloy shows much less segregation than the ordinary steel. Improvement has also been noticed in the working of the steel in the rolling mills and although the time has been short (from one to three years) which the rails have been in service on the different roads they have been found to give much better service than ordinary Bessemer steel rails."

From the foregoing it would seem evident that Mr. Stromeyer was somewhat at sea in his premises and in his conclusions. From the standpoint of the writer's experience it is safe to say that the story of the results to be secured by the use of titanium alloy will never be entirely complete until the day when iron and steel are no longer used as a means of carrying loads or stresses and of improving the strength to be secured by the use of substitutes for these metals.

I have only to say in conclusion that the largest steel makers in this country without an exception are using this alloy, and hundreds of the smaller manufacturers are using it as well.

Pittsburgh, Pa. CHARLES V. SLOCUM.

Furnace Electrodes.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—The discussion of a paper by the writer on this subject, at the recent Pittsburgh convention of the American Electrochemical Society, showed that there still exists a doubt in the minds of some that the final and much simpler method devised by the writer for determining the proper proportions of electrodes for furnaces is still subject to the former criticisms concerning an assumption of constant conductivities. As these final results are free from all doubt based on such criticisms, and can therefore be safely used, the writer gives below the substance of his closing and unpublished remarks to that discussion, which include a brief demonstration of the final method based on those new quantities, so that anyone may satisfy himself as to its correctness and its independence of any assumptions concerning the conductivities; if so, then it is of general application. This non-mathematical demonstration is not supplementary to the others, but replaces them, hence the older ones need no longer be considered.

The general plan of this proof is to first assume certain new, definitely defined quantities, then to determine their values for any one particular electrode in accordance with those definitions, and then finally to show that these quantities are truly specific and therefore apply to other electrodes of other dimensions. This may not be an orthodox method of proving something, but I believe its correctness cannot be questioned, and it is certainly far simpler than any algebraic proof would be.

In this explanation the two conductivities are dropped out of consideration entirely as factors in the calculation. Hence, neither their values nor the temperature variations of these values need be known or considered; they may be great or small, and they may vary much or little—it does not matter; there are no conditions stipulated or involved. The only assumption is that these conductivities will always have the same fixed and definite values for the same material at the same temperatures—that is, that they are definite specific physical properties; no further assumption is necessary.

Instead of these conductivities, let us adopt two entirely new and different physical quantities, which we have selected and defined to be the watts per ampere, denoted by E, and the section per ampere and per inch length, denoted by S'; we, of course, have a right to make these new quantities anything we please. Their numerical values are then determined in any specific case in accordance with these adopted definitions, and must always agree with those definitions.

These two quantities really measure what we want to know about electrodes; unfortunately they have as yet no names; it is as though the ohm had not yet been named and was therefore called the "volts per ampere"; but the lack of a name would not mean that it is not a true physical property, namely, resistance.

Let it be assumed now (to be proved later) that for each range of temperature these two quantities are true specific properties—that is, that like specific gravity, specific heat, specific resistance, etc., their values are independent of the dimensions of any particular piece. And just as the numerical values of these other specific quantities change somewhat for different conditions of temperature, pressure, etc., so the numerical values of these new quantities change somewhat for different temperature drops in the electrode; hence, such values are given either in the form of tables, or (as in that Pittsburgh paper) in the form of equations.

Let it also be assumed that all electrodes are always operated so that there shall be no flow of heat through the hot end and that this, therefore, is the only condition of operation under consideration at any time; let this be referred to as operation "under normal conditions."

Being true specific quantities (so far, by assumption only) they may be determined experimentally once for all for any given material, with an electrode of any one selected size, and for each of any desired number of temperature ranges, and then tabulated as constants; just as the specific resistance may be determined from a piece of a metal of any selected size under each of a number of conditions of temperature or pressure or tension, etc. This is what I did, as described in the other Pittsburgh paper; the values of E and S' were therein calculated according to the above definitions.

Now, by definition, $E =$ watts $\div$ amperes, just as resistance might be defined as volts $\div$ amperes. Hence, as this is a definite specific quantity for each range of temperature, it follows without any further proof or mathematical deductions that the watts lost in any given electrode operating under normal conditions are equal to the value of E (for that given temperature drop) multiplied by the current. Hence, this method of calculating electrodes is proved as to E.

It will be seen that in this demonstration of the writer's method, the conductivities and their variations are completely eliminated from the calculation and from all consideration, although they and their variations are really what determine the result; they may vary with temperature or not, and little or much; their values and variations are all taken care of very effectively in the tabulated values of E.

In precisely the same way, by definition, $S' =$ section $\div$ (ampere $\times$ length). Hence, it follows (if this is a specific quantity) that the cross-section of any electrode operating under normal conditions, is equal to the value of S' (for that given temperature drop) multiplied by the current and the length. All that was said about determining the watts lost applies similarly to this section, and the result as to S' is therefore also free from any assumption as to the values of, or the constancy of, the conductivities, although they really determine that section.

I hope this is hereby made clear, and that the persistent claims of some academicians that my final method is based on constant conductivities and that it is, therefore, unreliable, will hereby be completely set at rest.

It remains to be demonstrated that these new quantities are true specific quantities under the normal electrode conditions. Although this will appear self-evident to anyone giving it due thought, the following proof based on the simple, neat and ingenious one, devised by Dr. E. F. Northrup, and referred to in my papers, ought to satisfy even the most skeptical academicians who have persistently doubted its correctness. It might be added here, for their benefit, that we are here concerned only with electrodes of uniform cross-section from end to end, and that the Thomson effect, heterogeneity of material, lateral leakage, chemical changes in the material, and all other like minor matters which so greatly complicate the complete mathematician's equations, are here considered to be non-existent.

If it can be shown that the values of E and S' for any one particular range of temperature are the same for any cross-section and for any length then they are true specific quantities.

Imagine an electrode while operating under normal conditions to be cut in two, lengthwise. It is evident that this will change nothing in its operation, as neither heat nor current was traversing across that plane. Hence, one of these halves may be removed without causing any change in the other in which the current and the cross-section will then be just half of the original; the heat gradient of the remaining half will, of course, remain the same. The watts lost in the remaining half will, therefore, now be half of what they were, and will, therefore, still be equal to the same E multiplied by the current through that half, namely, by half the original current. The same will evidently be true for any other fractional part of the original cross-section, hence for all sections. That is, E is the same for all sections, and is, therefore, a true specific quantity in so far as the cross-section is concerned. No questions concerning the conductivities or their variations are here involved; they may have any values, great or small.

Now imagine this original electrode to be cut transversely into a very large number of extremely thin laminæ, like the leaves of a long row of books. Suppose now every alternate leaf was removed, and the rest brought together, making a new electrode just half as long. It is supposed, of course, that the contact resistance between the leaves is neglected or is always the same per leaf. Now, to produce the same amount of heat and the same temperature conditions in each one of the remaining laminæ as were before produced in two neighboring ones will evidently require twice the current that passed through it before; hence, the current must be doubled.

Nothing will hereby be changed in the heat gradient except that it will be condensed or shortened to half its length; the total drop from end to end is the same by definition; the value at the middle, or at 1/10 or 1/100 of its length from either end, will be the same in one as it is in the other, because the total drop is the same. In other words, the heat gradient will be twice as steep at every point; the drop in each leaf will be the same as it was in two leaves before. The new heat gradient will be like the old one seen through a cylindrical lens, or drawn to a shorter horizontal scale. This is true no matter how irregular the shape of the curve of this heat gradient was, provided only that the laminæ were thin enough and numerous enough.

Now, the resistance of this new electrode will be just half as great, as one of each pair of minute laminæ has been removed. The current being doubled, and the resistance halved, the watts will be doubled; hence, they will be again equal to the same E multiplied by the new current. The same argument applies equally well to any other length, hence to all lengths. Therefore, E is the same for all lengths and is therefore a true specific quantity in so far as the length is concerned.

Having proved the same for the section, it follows that this quantity E is independent of both section and length and is, therefore, a true specific quantity.

By the same method it can be shown that the other quantity S' is also a true specific quantity. If, for instance, the original electrode is cut in two lengthwise, the value of S' is the same for each half because the current is half as great in each; hence, S' is the same for all sections. In the second case, if half the transverse laminæ are removed as before, we have found that the current will be twice as great, and, therefore, the value of S' will again be the same, because the length has been halved, the value of S' by definition being $S' =$ section $\div$ (ampere $\times$ length). Hence, S' will be the same for all lengths. Being the same for any cross-section and any length it is a true specific quantity. Here again we are not in any way concerned with the values of or the variations of the conductivities, they being taken care of completely in the experimentally determined values of S'.

In introducing the quantities k and r in that paper, the writer did so merely to show how their electrode mean value could be determined from these new quantities if it is desired to know them; but it was clearly stated that k and r were no longer involved as factors in the present simplified method, in which they need not be known or considered at all.

In view of the criticisms made by some academicians of the writer's original paper in the preceding volume of these Transactions, it may be of interest to add that the present simplified treatment of this subject, the correctness of which it is believed will no longer be questioned, gives exactly the same results as the original and more complicated method described therein is used—that is, when the constants are described therein is used, that is, when the constants are determined as therein described. Hence, if the present treatment is correct, the complete method described in that former paper is correct and does not involve the supposed errors on which so much stress was laid by some of the critics, none of whom could, however, show that there were any real errors, but simply assumed them to exist because their mathematical treatment became too complicated to give a solution. The case is of interest as showing how an approximate mathematical treatment (and the unfinished highly mathematical solution offered by one of the critics in the last volume of the Transactions is at best only approximate) can be extremely complicated, when the mathematically exact one (as I believe the present one to be) is ridiculously simple.

Philadelphia, Pa Carl Hering.

The International Electrotechnical Commission has just issued its first annual report. So far the work of the commission seems to have been chiefly in effecting organization. At present the following countries are represented in the commission: Belgium, Brazil, Canada, Denmark, France, Germany, Great Britain, Hungary, Italy, Mexico, Spain, Sweden, United States. Prof. Elihu Thomson is the president of the commission.

A Quantitative Investigation of the Source of Hydrogen in Blast Furnace Gas.

By R. J. Wysor and W. D. Brown.

The dearth of available literature concerned with the subject of this paper induced the authors to undertake the present investigation. Doubtless the most classical memoir on the influence of hydrogen in the blast furnace has been furnished by Sir I. Lowthian Bell in his "Principles of the Manufacture of Iron and Steel." So far as we are aware, no attempt has previously been made to account quantitatively for the presence of hydrogen in blast-furnace gas.

The general impression, sometimes voiced in text-books, that most of the hydrogen in the gas is due to dissociation of the aqueous vapor in the air blast was proved to be erroneous, by calculation from the available analyses which we have examined. In the expectation that all the excess hydrogen, over that produced from the water vapor in the blast, originated from the coke charged into the furnace, this investigation was designed to indicate the amount of hydrogen resulting from each source. Although the outcome of our experiments has not substantiated the original idea, we believe the results here presented have an intrinsic value in furthering present knowledge of the subject, and in encouraging a new scientific thought as to the ultimate source of hydrogen in blast-furnace gas.

At the outset, it was realized that it would be difficult to obtain ideal conditions of experimentation, such as could be observed or measured with strict accuracy. All due precautions, however, were taken to maintain these conditions under close control. All of the operations of the test, subsequently described, were personally performed by the authors or under their direct supervision.

Number 5 furnace at the plant of the Duquesne Steel Works & Blast Furnaces was selected as the basis of the test. Its chief dimensions, height and bosh diameter, are 94 ft. and 23 ft. respectively. This furnace had been in blast for about one year, and had been blown out for repairs to the lining a short time previous to the test. The air blast was delivered in part by a cross-compound, vertical, steeple-type, steam-blowing engine; the remainder by a twin tandem, horizontal, gas-blowing engine.

A preliminary experiment of seven hours' duration was followed by a 24-hour test, the results of which appear in this paper, as it was deemed necessary to have a continuous record for a considerable length of time in order to apprehend general working conditions.

Details of the Test.

The coke used came from one mine in the Connellsville district. Large, representative samples were secured at two-hour intervals from the chutes of the furnace bins, as the coke was being emptied into the buckets. The initial and final samples were taken about two hours before the beginning and end, respectively, of the continuous test. The remaining burden of the furnace, chiefly ore and limestone, had been previously sampled and analyzed. The pig iron produced was represented by samples regularly secured for analysis.

Owing to a constant leakage of condensed steam into the intake valves of the compression tubes of the steam blowing engine, it was impracticable to effect moisture determinations in the air blast by the use of a psychrometer in the immediate vicinity. These determinations were made in the air blast passing through a 6-in. pipe connected to the blast main near the hot-blast stoves; a strong current of air was maintained through this line. A small petcock, situated on the 6-in. pipe, afforded an easy means for the control of pressure. The air current was forced through a suitable absorption apparatus containing concentrated sulphuric acid, a meter completing the train. Continuous samples of approximately 12 cu. ft. each were taken over two-hour periods for the entire 24 hours. The relatively low moisture results obtained in the air blast are due to the fact that the average temperature record of the test, conducted in April, was less than 60° Fahr.

Continuous gas samples were obtained in large aspirator bottles, at a uniform rate, over periods of time corresponding to those of the moisture samples. The samples were secured from the gas main leading from the dust catcher and were analyzed immediately.

Furnace Data.

The furnace was in good condition and worked with fair uniformity over the whole period. No tuyeres were removed on account of water leakage, nor could the furnace men detect any such leakage. Several small slips occurred, none of them exceeding 8 ft., and none occasioning loss of stock through the explosion doors. The blast pressure averaged about 14 lb., the calculated, approximate volume of air delivered to the furnace being 39,950 cu. ft. per minute. The hot-blast temperature fluctuated between the limits of 900° Fahr. and 1350° Fahr., the average being about 1150° Fahr. The average temperature of the top gas was approximately 550° Fahr. The ratio of coke to pig iron produced was 1.11, and that of the limestone 0.51.

Analytical Determinations.

All due precautions were taken in the various analyses, the results of which appear in subsequent tables. An average coke sample representing the entire time of the test was prepared for the ultimate analysis. Published statements as to the loss of hydrogen in coke samples due to fine grinding are not decisive. For the hydrogen determinations in this work, the samples were rather coarsely pulverized and the estimations were made upon the undried coke, due correction being effected for the moisture content.

Some attempt was made to determine the manner in which the sulphur existed in the coke, but aside from the fact that no appreciable amount was present as sulphate, the analytical evidence was not conclusive. However, in the absence of sulphur existing as sulphate, it may be shown by calculation that whether the sulphur is present in the organic form, as iron pyrites, or as ferrous sulphide, a negative correction of three-sevenths the iron content of the coke should be applied to the apparent ash to obtain the true ash. It happens in this instance that three-sevenths of the iron content equals three-eighths of the sulphur, the value generally accepted for effecting the correction, the iron (0.76 per cent) being present in just sufficient quantity to combine with all the sulphur as iron pyrites.

Although it is impracticable to utilize the individual analyses in the calculations, they are presented in considerable detail to indicate the range of variation.

Analyses.

The tabulated results of the moisture determinations in the air blast, all due corrections having been made, are calculated on the basis of dry air at standard conditions. The corresponding moisture determinations in the atmosphere, made by means of a psychrometer, are also presented. The excess moisture in the air blast is due to the steam leak into the compression tubes.

	MOISTURE.		GAS.					COKE.
	Grains per Cu. Ft.							
	Atmosphere.	Air blast.	Carbon Dioxide.	Carbon Monoxide.	Hydrogen.	Methane.	Nitrogen.	Ash.
1.	2.91	3.36	14.3	24.8	4.0	0.0	56.9%	11.08%
2.	2.39	3.31	13.4	25.4	4.0	0.2	57.0	11.08
3.	2.39	3.49	13.1	26.4	4.5	0.1	55.9	11.48
4.	2.52	3.53	12.4	26.8	4.5	0.0	56.3	10.86
5.	....	3.59	13.1	24.0	5.3	0.1	57.5	11.27
6.	2.66	3.67	12.8	25.9	5.5	0.1	55.7	12.22
7.	....	3.55	14.0	24.3	4.2	0.1	57.4	13.30
8.	2.40	3.07	13.6	24.6	4.3	0.2	57.3	12.32
9.	2.55	2.97	13.4	25.3	4.0	0.1	57.2	12.48
10.	....	3.00	12.9	25.0	3.9	0.2	58.0	11.24
11.	2.70	3.17	13.7	24.0	3.8	0.1	58.4	12.51
12.	....	3.23	13.2	24.4	3.6	0.1	58.7	13.92
Average Gas Analysis...			13.3	25.1	4.3	0.1	57.2	

	Carbon.	Hydrogen.	Oxygen.	Nitrogen.	Ash.	Sulphur.
Average Coke Analysis.	86.12	0.55	0.44	0.35	12.00	0.86

Average Moisture in Air Blast..... 3.33 grains per cu. ft.
Average Carbon in Limestone Charged.....11.1 per cent.
Average Carbon in Pig Iron Produced..... 4.0 per cent.

previously mentioned. The numbers designate the sequence of samples taken over two-hour periods, or at two-hour intervals. The percentages of ash in the coke samples are expressed to indicate somewhat the variation in composition. Hydrogen estimations in several samples showed the percentage of this constituent to be fairly uniform.

Calculations.

It is hoped that the subsequent methods of calculation by volume may be explained with sufficient clearness as to be readily intelligible to those not at present familiar with this system.

Hydrogen Derived from the Coke.

The percentage of hydrogen in the gas originating in the coke may be determined from three factors: the hydrogen and carbon entering the gas, and the percentage of carbon compounds in the gas. The ratio by weight of hydrogen to carbon gasefied is converted to the ratio by volume of the hydrogen to the carbon compounds in the gas by multiplying by six; this factor is deduced by the following reasoning.

One molecule of hydrogen contains two atoms. One molecule of carbon monoxide, carbon dioxide or methane contains one atom of carbon, having a weight equal to 12 atoms of hydrogen. According to Avogadro's law, the molecules of all gases occupy equal volumes. Therefore, two atoms of hydrogen, or one molecule, occupy a volume equal to that occupied by a molecule of the gaseous carbon compounds, containing one atom of carbon, which weighs 12 times as much as the atom of hydrogen and six times as much as the hydrogen molecule. Conversely, for unit weights of hydrogen and carbon entering the gas, the ratio of volumes of hydrogen to carbon compounds must be as 6 to 1. The calculations are tabulated as follows:

Hydrogen in coke	0.55	per cent
One-eighth oxygen in coke (0.44%)	0.055	per cent
Residual hydrogen in coke	0.495	per cent
Coke used per ton of pig iron	1.11	tons
Residual hydrogen per ton of pig iron	.00549	ton
Carbon in coke	86.12	per cent
Carbon used per ton of pig iron	1.11	tons
Carbon in coke per ton of pig iron	.9559	ton
Carbon in limestone	11.10	per cent
Limestone used per ton of pig iron	0.51	ton
Carbon in limestone per ton of pig iron	0.0566	ton
Total carbon entering the furnace per ton of pig iron	1.0125	ton
Carbon in pig iron	4.0	per cent
Carbon in a ton of pig iron	.04	ton
Carbon gasefied	.9725	ton

$$\text{Ratio by weight of hydrogen to carbon gasefied} = \frac{0.00549}{0.9725} = 0.00565$$

Multiplying above ratio by 6, we obtain:

Ratio by volume of hydrogen to compounds of carbon in gas	0.0339
Sum of percentages of carbon dioxide, carbon monoxide and methane in gas	38.5
Hydrogen in gas	1.30%

Hydrogen Derived from the Moisture in the Air Blast.

Moisture in the air blast on coming in contact with incandescent coke reacts with the formation of one molecule of hydrogen from one molecule of moisture. Moisture in the air, being unsaturated, acts as a perfect gas and, in accordance with Avogadro's law, occupies a volume equal to that of the hydrogen derived from it. Therefore, the actual volume of hydrogen in the gas is equal to the volume of moisture in the air blast. However, for statement in percentage, correction must be made for the ratio of gas to air volume. This correction is based on the percentage of nitrogen in dry air and in the gas. By

Moisture accompanying 1 cu. ft. dry air @ 62° F. and 30" mercury	3.33	grains
Weight of 1 cu. ft. aqueous vapor @ 62° F. and 30" mercury	332.01	grains
Volume of 3.33 grains @ 62° F. and 30" mercury	0.0100	cu. ft.
Hydrogen entering gas from 1 cu. ft. dry air	0.0100	cu. ft.
Nitrogen in gas	77.2%	
Nitrogen in air	79.1%	
Ratio of gas to air blown	1.38	
Hydrogen derived from moisture per cu. ft. gas	0.0072	cu. ft.
Percentage of hydrogen derived from moisture	0.72%	
Hydrogen found in gas by analysis	4.3 %	
Hydrogen in gas calculated from coke and moisture in air	2.03%	

calculation the nitrogen in the gas derived from the coke is 0.07 per cent of the gas, and is omitted from the calculations as being negligible.

Conclusions.

The result of this test demonstrates that some other source or sources than the two discussed in this paper were responsible to a large extent for the hydrogen in the furnace gas. The most obvious inference would be that the extraneous source was from a leakage in the cooling-water system around the furnace, or in the water-cooled valves at the hot-blast stoves. We do not believe an appreciable error could have existed at the furnace, as the furnace men claim the ability to detect a very slight leakage. It was impossible to determine definitely whether or not a leak existed in one of the water-cooled valves though they were examined and represented to be in good order; no trouble was detected from this source until a considerable time after the test.

By calculation it may be shown that with the conditions obtaining in this test there must be a leakage of approximately 3.1 gal. of water per minute to yield 1 per cent of hydrogen in the gas, or nearly 7 gal. per minute would be required to produce the hydrogen unaccounted for in the gas. We believe a leakage of no such magnitude could have existed throughout this test.

Likewise it may be proved by calculation, or by simple examination of the foregoing figures, that one grain of moisture per cubic foot of air blown is responsible for only 0.22 per cent of hydrogen in the resulting gas.

We desire to call attention to the fact that numerous analyses made in this laboratory, of gas samples caught directly after slips of considerable magnitude, show an increased percentage of hydrogen, sometimes reaching a very high figure. One sample obtained directly after a 21-ft. slip showed the abnormal result of 30 per cent hydrogen. It is difficult to account for such high percentages of hydrogen except by the theory that the water, present as moisture or in the combined form, carried in the burden is dissociated when the stock drops down into contact with incandescent coke. Of course, the hydrogen in the fresh coke additions to the furnace might also be expected to be liberated with greater rapidity.

Further, might not this dissociation occur at a higher level in the furnace than is generally supposed? The Mesaba ores used in the furnace contained between 5 and 6 per cent combined water. It is known that a fairly high temperature is required to expel the water of hydration of certain silicates. We believe it is not impossible, nor even improbable that part of the combined water suffers dissociation with the consequent liberation of hydrogen. This point has not been tested experimentally.

In consideration of the rapid progress being made in the installation of large gas power engines, using blast furnace gas as fuel, and the deleterious influence of high percentages of hydrogen in the explosive mixtures used, the question of hydrogen in the gas, its quantity, and its source, is one of peculiar interest.

Duquesne Steel Works, Duquesne, Pa.

Convention of American Institute of Electrical Engineers.—The annual convention of the Institute will be held from June 28 (Tuesday) to July 1 (Friday), at Hotel Waumbeck, Jefferson, N. H. The program contains 20 papers, dealing with lighting, high-tension transmission, telegraphy and telephony, industrial power and traction matters. Among the entertainment features are a reception and dance on Tuesday evening at the Waumbeck and a trip to Mount Washington, which will take practically the whole of Thursday. For Thursday evening a dinner and discussion by the sections committee and section delegates has been arranged; for Friday night, a dinner, followed by a discussion by the educational committee. The professional program contains also the presidential address of Mr. Lewis B. Stillwell and a report of the standards committee, which offers certain standardization rules for approval.

Chloridizing Roasting of Cuprous Pyrites Cinders.

BY STEN LILJA.

In the middle of last century iron pyrites began to be commercially used in the manufacture of sulphuric acid. From the beginning, it was thought that the residues from the roasting operation, on account of their content of such valuable elements as copper and iron, ought to be further treated, and a number of prominent metallurgists have bent their efforts toward developing methods by which the copper could be extracted and the residues, or cinder, converted into a commercial product.

To extract the copper by means of chloridizing roasting, followed by leaching, was early proposed by Orshall, Longmaid, Maumené, Bechi and Haupt, Schaffner, not to mention others. Longmaid obtained a patent dated Oct. 20, 1842, and another in January, 1844, both relating to the treatment of cinder by roasting it with NaCl, but his real intention seems to have been to produce principally salt cake and chlorine.

Bechi and Haupt were operating, in 1856, a plant located in Capanne Vecchi, where they were recovering copper by a similar process.

Henderson carried the process to a greater perfection. In 1865 he built a factory at Hebburn (Bede Metall. Comp.) with the object of extracting copper from pyrites-burners by a method which he had protected by patent. It was Henderson who introduced the absorption towers through which the acid gases from the chloridizing roasting operation have to pass.

The Henderson plant at Hebburn was from the start a great success to such a degree that in five years, according to Wedding, not less than 20 plants were operating in a similar way, treating together 250,000 tons of cinder yearly. Since then the Henderson method, or the chloridizing roasting process, is exclusively used for extraction of copper from pyrite cinders, by which process the residues incidentally are converted in a form more suitable for the ultimate recovery of the iron.

The chemical principles of the chloridizing roasting, as given by Henderson 45 years ago, still stand as correct, but the mechanical arrangements naturally have developed with the progress of modern time, lowering the cost of treatment and thus making the process more economical.

The greatest number of plants using this method and now operating are situated in England, in the Tyne district and in South Lancashire. The others are distributed over Germany (Duisburg, Königshütte), Belgium (Hemixen), Sweden (Helsingborg), United States (at the Pennsylvania Salt Manufacturing Company's factories in Philadelphia and Natrona), Austria (Wittkowitz, near Mährisch-Ostrau).

Henderson Process.

The principal features of the Henderson process can be described shortly as follows:

Cuprous cinder is mixed with a certain proportion of common salt (the amount depending upon the sulphur and the copper content) and the mixture crushed to 8 to 10 mesh. After crushing, it is subjected to chloridizing roasting in a reverberatory or a muffle furnace at as low a temperature as possible (about 1100° Fahr. being most advantageous) The following reaction will take place:

$$CuS + 2NaCl + O_4 = Na_2SO_4 + CuCl_2.$$

$CuCl_2$ is soluble in water. The aim is, therefore, to convert the total copper in the residues into this form, but on account of the chemical and physical conditions of the residues and the imperfect method of operation, other reactions will take place, causing a resulting product with only part of the copper soluble in water, but with all the rest soluble in weak acids. To facilitate the extraction of copper at the leaching, the acid gases —HCl and SO_3—which develop during the roasting, are absorbed in towers, and with the diluted acid solutions thus obtained the roasted residues are treated after having been transferred to leaching tanks.

In the cupriferous lyes the silver is precipitated with KI and afterward the copper with scrap iron. The silver precipitate contains about 6 per cent of metallic silver, which can be recovered by smelting. The copper precipitate—usually called cement copper—contains 50 per cent to 95 per cent copper. It is directly smelted and refined.

The final residue, after the leaching, goes under the trade name of purple ore or blue billy. On account of its high iron content it is a valuable raw material for blast furnaces, but must on account of its fineness first be briquetted.

Nearly all the cuprous pyrites, i.e., iron pyrites with over 1 per cent Cu, which is being used in the manufacture of acid, is produced in Portugal, Spain, Norway and Italy. Spain and Portugal produced in 1907 about 3,500,000 tons of cuprous pyrites. The chief producers in Spain are the Rio Tinto Works and Tharsis Sulphur & Copper Company.

The Rio Tinto Works' production of cuprous pyrites was, according to *Mineral Industry*, in long tons.

	Locally treated.	For shipment.	Total.	Per cent Cu.
1907....	1,265,090	641,858	1,906,948	2.417
1908....	1,115,610	604,275	1,719,885	2.265
1909....	1,184,188	604,795	1,788,987	2.349

The copper in the locally treated pyrites is extracted by smelting or by the famous Rio Tinto leaching process (described in the *Mineral Industry*, Vol. XII, p. 112; XV, p. 288; XVII, p. 278).

The exported pyrites are sent all over Europe and to the United States.

Norway exported in 1908 not less than 250,000 tons of pyrites. The two biggest producers were Sulitjelma and Foldalen, the former with a production for the year of nearly 100,000 tons, while the latter shipped about 50,000 tons.

The greater part of the Sulitjelma ore is desulphurized at sulphuric acid plants and sulphite mills in Norway and Sweden, and the Cu in the residues extracted partly at Helsingborg, Sweden, and partly at Hemixen, Belgium. The copper content in the Norwegian pyrites varies from 2 per cent to 5 per cent.

In the United States, according to *Mineral Industry*, 206,471 tons of pyrites for acid making were produced in 1908. This pyrites contains generally little copper.

The pyrites imported, 1909, according to one of the biggest import firms, was 669,992 tons, with following Cu content:

232,028@1.80 per cent Cu.
281,196@0.73 per cent Cu.
156,773 under 0.25 per cent Cu.

According to the burning the best results with highest copper recovery are obtained by chloridizing and leaching when the pyrites are uniformly roasted, not being sintered and not containing kernels of green ore. There ought still to be 1 to 1½ times as much sulphur in them as the amount of copper they contain.

By the burning, the pyrites will lose in the average 30 per cent of their weight, so that the copper contents in the residues will be correspondingly increased (10/7 of the original).

For Rio Tinto ore it can be seen that with 2.35 per cent Cu in the crude pyrites, the cinder will contain 3.34 per cent in average. This Cu content varies considerably in shipments from time to time. For instance, during last year the Rio Tinto pyrites which was imported in the United States did not contain more than 1.6 per cent of Cu, making the copper content in the residues 2.30 per cent.

Arrangement of Plant.

The most advantageous location of a chloridizing plant for the treatment of pyrites residues is, of course, in close proximity to the acid manufacturing plant, so that there will be a minimum of transportation. In some cases it might be necessary to locate the plant at some central point to where raw material from several acid works can be brought together. A large plant treating a great tonnage can be operated more economically than a smaller one. It is then also possible to arrange for further treatment of the purple ore in the same place.

On the arrival of the residues at the works, they have to be carefully stored, preferably under a roof, as too great a percentage of moisture will cause not only trouble in the crushing and screening, but also added cost for the evaporation of the water in the roasters and a lowered production. Another reason for great care in the storing of the residues is the variation of the sulphur content in material from different desulphurizing plants. The proper way is to test incoming cinder for sulphur before unloading and place it in the storage sheds according to sulphur content so that in each compartment there is always cinder of a certain grade. It is then easy in the operation of the chloridizing roasters to mix the residues so that the charge always contains a uniform amount of sulphur. This facilitates the work and insures good results.

In describing more in detail the chloridizing roasting, we can to advantage subdivide the operation into the following five processes:

I. Mixing with salt and the crushing of the mixture.
II. Roasting proper.
III. Leaching.
IV. Precipitation of silver.
V. Precipitation of copper.

I. Mixing with Salt and Crushing.

The mixing with salt, with the subsequent crushing, must be carefully done for a successful roasting and leaching. It has been the custom to spread the residues in a layer on a platform or the floor and add 12 per cent to 15 per cent of salt, and if needs be, some green ore. The whole mass was afterward thrown with shovels directly into the grinding machine. In some places there is employed a mechanical mixer consisting of a rotary drum, where the ground cinder is mixed with salt. When the grinding and mixing take place simultaneously the simplest way is to feed them to the grinder by means of some feeding apparatus, by which the proportions can be easily governed. This will incidentally make the feed to the grinder more uniform, which is beneficial for an even grinding. The crushing and grinding machines which are usually found in plants of this kind are Cornish rolls, ball mills, Chilian mills, Carlin mills and Kent mills. The last mentioned are best suited for fines. When used for lump cinder, a preliminary crusher is required.

The fineness to which the material is ground depends to some degree on the nature of the residues, but usually they are comminuted to 8 to 12 mesh.

The necessity for a uniform grinding can be understood from the fact that too large grains would at the chloridizing be only partially chloridized. The material can be allowed in the kiln only a certain time, as otherwise the fines would be "dead roasted." A too fine grinding, on the other hand, would at the subsequent leaching cause too much slime that makes trouble at leaching.

After the crushing, the material is preferably stored in bins, from where it is drawn at intervals to be charged into the kilns.

II. Roasting.

Kiln Construction.—The first type of kilns used for chloridizing roasting was the reverberatory kiln with the flame passing directly over the ore. On account of the uneven temperature caused in the kiln by the direct flame, the results were usually unsatisfactory, because, in the part of the charge which was nearest the fireplace, the copper, too much heated, was transformed into oxide—"dead roasted."

To overcome this disadvantage an arch was built across the hearth nearest the fireplace over which the flame would strike, in this way a kind of muffle would be formed, open toward one side. This construction was used at the Bede Metall Comp. in England.

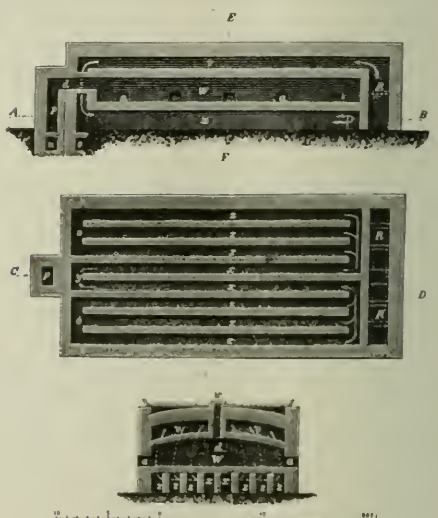
Another type of reverberatory furnace is used at Widness, in Lancashire. It is fired with producer gas with the combustion gases passing in channels under the hearth in the length direction of the kiln and returning over the ore to the stack flue.

All the reverberatory furnaces have the common drawback that the outgoing roasting gases are mixed with the combustion gases, increasing the difficulties of the absorption of the acids causing a direct loss as the acids for the leaching have to be acquired from elsewhere.

At nearly all the chloridizing roasting plants the ore is, therefore, treated in muffle kilns. Of the several types of such kilns that have been designed and are in use, the most common type is the one that is employed, for instance, at Tharsis Sulphur & Copper Company's works. It is described in Lunge's "Sulphuric Acid and Alkali." The hearth is 9 ft. to 14 ft. in width and 30 ft. to 40 ft. long, and takes a charge of 2.5 tons to 4.5 tons. The combustion gases pass through channels around the muffle and do not come in direct contact with the ore, which, therefore, is more uniformly heated. The draft of the combustion and of the roasting gases are regulated independently of each other. A similar type of kiln has been used by the Pennsylvania Salt Manufacturing Company.

Figs. 1, 2 and 3, showing a muffle furnace for chloridizing roasting, are reproduced from Schnabel's "Metallurgy."

Another type of furnace was used for several years at Pitkaranta, in Finland, and some features of this construction were



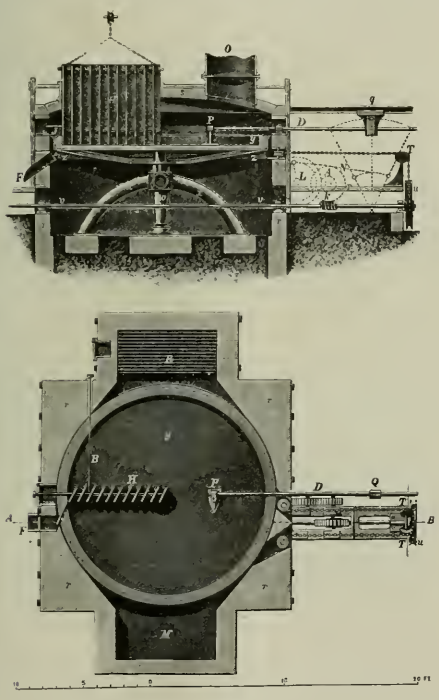
FIGS. 1, 2 AND 3.—MUFFLE FURNACE FOR CHLORIDIZING ROASTING.

adopted, and are still used to some extent, at the Helsingborg plant. The Pitkaranta furnace consists of four muffles 6 ft. in width, 1 ft. high and 12 ft. long placed over each other and connected by vertical holes, through which the ore is raked from muffle to muffle in the same way as in a maletra furnace. The muffles are open at both ends to facilitate the operations. For each furnace there are four fireplaces, located two and two under the lowest muffle. The combustion gases pass around the muffles and escape through two horizontal conduits running over the furnaces, which are arranged together. The gases from the ore are collected by other conduits lying parallel with those just mentioned.

Mechanical Roasters.—The chloridizing roasting in hand operated furnaces is rather expensive and more or less dependent on the skill of the workmen. It has, therefore, been the aim to substitute the hand labor by some mechanical device. As early as 1872 Gibb patented a mechanical roasting furnace, in detail described in *„Dingler's Polytechn. Journal,* No. 204, page 288, and in Lunge's "Sulphuric Acid and Alkali," Part I.

The kiln consisted of a round hearth, 15 ft. in diameter, supported in the center by a vertical shaft rotating in a step bearing. The hearth was enclosed by brick walls and a brick arch. The fireplace was at the base of the furnace and the combustion gases were allowed to pass over the ore to the flue of the stack opposite the fireplace. By an endless chain running in a groove around the hearth and driven by a tooth wheel on a vertical transmission axle outside the furnace, the hearth was rotated. To distribute the ore, which was fed centrally, there was used a system of obliquely arranged rabble plates, suspended from above and together reaching from the center to the periphery of the hearth. The plates were lifted out as soon as the material was evenly distributed. During the rotation of the hearth the ore was continuously stirred by a plow-shaped raker, which, by some mechanical contrivance, was pushed back and forth over the hearth. Figs. 4 and 5, showing Gibbs' mechanical furnace for chloridizing roasting, are reproduced from Schnabel's "Metallurgy."

When the roasting was finished, the rabble plates were again lowered and the ore was automatically discharged at the continued rotation of the hearth. The usual charge was 5 tons, for



FIGS. 4 AND 5.—GIBBS MECHANICAL FURNACE FOR CHLORIDIZING ROASTING.

the roasting of which nine hours were required. Gibb states that the ore was better roasted than at the hand-operated kilns. More copper was soluble in water and less salt was consumed. It was used for some years at the Bede Metall. Works, but is now superseded by later constructions. A drawback in this construction was the same as at reverberatory furnaces, i.e., combustion gases and roasting gases were mixed.

Later, mechanical roasting kilns of various types have been tried at several places. The Pearce Turret-kiln, for instance, is used at Natrona, Pa. At the Falun Copper Works, in Sweden, the White Howell kiln was used for some time. The results were not satisfactory, and the use of the kiln was discontinued.

The same principle as in this kiln was found in the type employed at the Eastern Nodulizing Works, at Hackensack River, Newark, where a cement kiln was adapted for chloridizing roasting. It is said that the results were good. This type of kiln would seem, however, not to be quite suitable for this purpose, partly for the reason that the flame comes into direct contact with the ore when the roasting is complete, whereby $CuCl_2$ is decomposed, partly on account of the mixing of the roasting and combustion gases as in the reverberatory furnaces.

The Wedge Furnace (this journal, Vol. VI, p. 457, VII. p. 225) is a distinct improvement on all the previous types. When used for chloridizing roasting, one hearth is employed of 32 ft. diameter. From a vertical rotating axle in the center which supports horizontal with rabble plates furnished arms, the ore is stirred and moved forward from the center to the discharge at the periphery. The combustion gases pass through muffles over or under the hearth.

For the recovery of the acid gases developed in the chloridizing roasting process, they are led through absorption towers built in the same way as those employed in the manufacture of muriatic acid. These are filled with coke, clay pipes or something similar.

The gas enters at the bottom and meets water dripping down from the top, thus the most part of the HCl, SO_3 and CuCl that the roasting gases contains is obtained in a weak acid solution 2-5° BB, which is led in wooden pipes to the leaching department.

Operation.—According to Wedding, the method of operation at the use of reverberatory furnaces is as follows: The ore after the charging is spread out and is slowly heated up until it becomes red hot nearest the fireplace. The ore then is turned over and the firing stopped. It is then left lying for a few hours without firing, but with excess of air, during which time the temperature decreases to dark-red color. Now a vigorous stirring of the ore with broad iron bars and hand rabbles follows, which will cause a raising of the temperature to a light cherry red.

In the beginning a greater part of the gases develop and blue flames indicate that sulphur is oxidized. By and by the flames decrease in strength until they finally disappear at the same time as the ore gets colder. Most of the sulphur is now oxidized. A sample is taken and a simple test is done to ascertain how soluble the copper is. If the test is satisfactory, the ore is discharged on the floor to cool off before leaching.

The usual test to try whether the roasting is completed is made in the following manner by the foreman or the roaster: Five grains of the material is boiled with weak (2 per cent) HCl for five minutes. The copper is then washed out with hot water.

The rest, which has not gone into solution, is boiled with aqua regia until the iron is dissolved. Ammoniac is added and the precipitate is filtered and diluted to a certain volume. The intensity of the color of the copper solution indicates whether the roasting is finished or not.

At the Helsingborg Copper Works the roasting in the Pitkaranta furnace is thus carried on. The crushed cinder is stored in bins, from which it is at intervals drawn into buckets traveling on a tramway to the conical bunkers over the roasting kilns. The bunkers are closed at the bottom by an iron ball, at the lifting of which a charge of 2000 lb. drops into the uppermost muffle. It lies here three hours, and is then raked down to the next muffle. The temperature has slowly increased, and at the end of the three hours it is so high that the reaction has begun, as can be noticed from the blue flames of burning sulphur.

On the second muffle the ore is distributed and left for a second period of three hours, at the end of which it is transferred to the third muffle. The sulphur flames are not any longer so pronounced and at the transfer of the ore to the fourth shelf, which is the lowest, most of the sulphur is eliminated.

After resting here three hours, the ore is finally discharged. While on the third and fourth shelf it is to advantage to occasionally stir the ore. The discharge opening in the lowest shelf is always left open for the entrance of the air required for the roasting, and all the time a uniform fire is kept in the fireplace.

To obtain an even roasting of the ore during the 12 hours it remains in the kiln it is essential that the S content always is the same. In some cases it is necessary to add some green pyrites. In other cases, when the S-content of the charge is too high, the feed is lowered, so that the same result is obtained as if the ore had been lying in the kiln a longer time.

It is of importance that the roasted ore is allowed to cool off without too much disturbance. At the discharge, the various salts are still in melted condition. At the cooling, if left alone, they will solidify in a porous mass easy to leach. At the Helsinghorg plant the ore is discharged on inclined cooling plates located in the basement under the kilns. From there the

Fig. 6.—Arrangement of the Tanks at Eastern Nodulizing Works Plant.

ore will slide into small cars running on an industrial railway. The cars are raised to the level of the leaching room by an elevator

III. Leaching of the Roasted Cinder.

A convenient arrangement of the leaching department is to have the leaching tanks on a higher elevation than the precipitation tanks, so that the copper-carrying lyes can run by gravity from the former tanks to the latter ones. If the elevation of the ground is unsuitable, the leaching tanks can, for instance, be placed on an upper floor. After the treatment, the blue billy can be loaded directly into small cars which run out on a stock trestle to be dumped.

Fig. 6 shows how the tanks are placed at the Eastern Nodulizing Works plant, while Fig. 7 gives a general view of the leaching department at this plant.

The leaching tanks usually are built of wood, square or round, and are large enough for a 10-ton to 25-ton charge. They are lined with lead or bricks in cement or asphalt, which latter must be protected by an inside lining of wood.

A very good construction of leaching tanks is used at Helsingborg (Fig. 8), and is well worth describing in detail. The tanks are elliptical, with diameters of 10 ft. and 6 ft. and an inside depth of 4 ft. For building material Swedish resinous pine is used. The bottom and the staves are composed of 5-in. wood. The latter are held together by three iron bands 2 in. x $3\frac{1}{4}$ in. kept at a short distance from the sides of the tanks by pieces of wood so that lye leaking over the edges will not disintegrate them.

As near the bottom as possible there is a hole through the wall for the removal of the lyes. To the sides of the hole a lead pipe is attached. This is continued by means of a rubber hose with squeeze for the drainage of the tank.

Wooden launders have to be used for the transportation of the lyes to and from the tanks. For whatever pumping is required either steam injectors or air-lift pumps are employed

In preparing the tanks for the ore standard bricks are placed on edge right on the bottom. On these, special perforated bricks, 12 in. square, are laid as a false bottom for a filter composed of straw or screened lump cinder, iron ore, or something similar, on top of which the ore is charged and leveled.

The leaching itself is done approximately in the same way at various places. At Helsingborg, for instance, after the leveling of the charge, acid lyes of 5° to 8° B strength, obtained in previous leaching, are let into the tank. The ore is still warm from the roasting. The first lye that is drawn is very strong (40° B). It contains about 5 per cent copper and nearly all the silver. It is, therefore, transferred immediately through a special launder to the silver precipitation department.

When the strength of the lyes as they are drawn has gone down to 22° B, they proceed to the copper precipitation. This continues until the strength of the outgoing lyes has gone down to 10° B. The incoming acid lyes are shut off and in their place acid from the towers is let into the tank. This acid is allowed to remain in the tank two hours, at the end of which time the lye is exchanged for fresh acid, which in its turn remains two hours. The lye so obtained is used as the first lye for freshly filled tanks.

After this leaching, the ore is washed with water, first with such as has been used at the washing of the previous tanks and finally with fresh water. To test whether all the copper is extracted, a well-polished iron plate is stuck down into the offgoing lyes. If there is no coloring of copper, the leaching is finished.

The wash water ought to have a temperature of about 50° C. The time of leaching is usually 40 hours for a 10-ton charge.

Of the copper in the roasted ore about

80 per cent is soluble in water,

16 per cent is soluble in weak HCl,

4 per cent requires HNO_3 to be put in solution, and will remain in the purple ore.

IV. Precipitation of Silver.

At the chloridizing roasting AgS will be converted into Ag Cl, which at the leaching is transferred into solution by the

Fig. 7.—General View of the Leaching Department at Eastern Nodulizing Works Plant.

NaCl and $CuCl_2$ lyes. If the cement copper is to be refined electrolytically a special precipitation of the silver is not necessary. If this is not the case, it is usual to extract the silver before the copper precipitation.

Claudet's process is most common. At this the lyes, which contain 0.0025 per cent to 0.01 per cent silver, are precipitated with ZnI_2 or NaI.

The following reaction takes place:

$$2\,AgCl + NaI = NaCl + AgI$$

AgI forms a heavy insoluble precipitate, which is allowed to sink to the bottom in tanks prepared for this purpose. The precipitate, containing besides AgI also $PbSO_4$ and $CaSO_4$, is washed until it is clean from copper. It is thereafter treated with Zn and H_2SO_4, which causes the iodine to go into solution as ZnI_2, to be washed off and re used.

Instead of this process with Zn and H_2SO_4, sometimes Na_2S is employed, the reaction then being $2AgI + Na_2S = Ag_2S + 2NaI$.

The NaI is washed out and re-used.

The first lyes from the leaching of the roasted cinder, which contain most of the silver, are transported to special tanks in which slime, mechanically carried over by the lyes, will settle together with some $CaSO_4$ and $PbSO_4$, which are precipitated at the cooling of the solutions. The remaining lye is analyzed for silver, a sufficient quantity of KI in a solution of 0.1 per cent to 0.2 per cent strength is added, and the mixture is well stirred, preferably with some arrangement operated by compressed air or mechanical means. For that reason it is better to carry out the precipitation in a special tank from which the lyes are transferred to other tanks constructed similarly to the leaching tanks, where the precipitate will settle.

After two or three days the remaining lye is drawn off. It may then still contain 2 mg. to 3 mg. Ag per liter. For emptying the tanks, they are furnished with two holes, one very near the bottom and the other about 8 in. higher. The desilverized lye is drawn through the upper hole, so that the precipitate in the bottom is left undisturbed.

New lye from the precipitation tanks is let in and the procedure is repeated until there is so much silver slime in the bottom that it has to be removed. This has usually to be done every month or two, depending on the original silver percentage.

The silver slime is drawn through the lower hole. Some gypsum, which has crystallized on the sides of the tank, is removed separately and treated. It contains 0.5 per cent silver. The slime is washed with hot water and some H_2SO_4 for dissolving copper salts, and is afterward treated with Na_2S or Zn and H_2SO_4 in rotating wooden cylinders. As soon as the iodine is completely washed out, the slime is taken to filter presses or drying kilns. It is then ready for melting. It contains 3 per cent to 10 per cent silver and usually some gold.

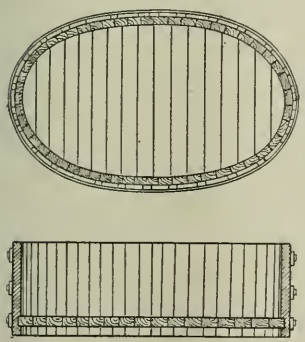


FIG. 8.—LEACHING TANK USED AT HELSINGBORG'S COPPER WORKS.

V. Precipitation of Copper.

For the precipitation of copper of the lyes from the leaching plant and the silver precipitation tanks, iron scrap is employed.

The tanks here also are built in the same way as the leaching tanks. Fig. 9 shows the precipitation tanks at the Eastern Nodulizing Works plant.

The tanks are filled loosely with thin scrap iron before the lyes are let in. These are heated with steam to nearly boiling temperature, which hastens the precipitation. The reaction is quickest when the iron is clean and the lyes are strong. They are left there until a test with a polished iron plate does not show indication of copper.

They are then exchanged for new lyes and this is repeated until the precipitation takes place so slowly that it is better to take away the precipitated copper and clean the scrap iron. It is on account of the copper covering all the iron that the reaction is delayed. The cement copper is removed with copper shovels and the iron pieces are collected on a copper sieve, while the fine copper is washed off with water.

The cement copper is further washed with hot water in special tanks for the removal of chlorides, and is then dried in drying chambers or by pressing out the water in hydraulic presses.

The decopperized lyes are transported through a series of safety tanks in which traces of copper, which they still may contain, are caught, together with fine copper slime, which is mechanically carried over. The finished lyes usually contain small percentages of Zn, Ni, Co, together with Fe and Na_2SO_4. It has long been desired to find some method by which these

FIG. 9.—PRECIPITATION TANKS AT EASTERN NODULIZING WORKS.

substances could be economically saved, but so far the problem has not been satisfactorily solved.

Treatment of Final Residue.

In the leaching the cinder is freed from the greatest part of its copper and sulphur content and incidentally from a number of other elements, such as Ni, Co, Mn, and the residues—purple ore or blue billy—consist chiefly of Fe_2O_3 with some SiO_2 and Al_2O_3, the qualities of which depend on the purity of the original pyrites.

In the blue billy from Rio Tinto pyrites, the iron content usually is 64 per cent. It can, therefore, be considered as a valuable iron ore, although its fine granulation makes its direct use in the blast-furnace charge rather disadvantageous. A large part of it is liable to be carried away as flue dust with the gases.

Nowadays it is, therefore, usually converted into lump form by some sintering process before it is used in the blast furnace. The nodulizing method and the Gröndal briquetting process are among the best known and the most successful processes. In the former, the cinder is passed through a long rotary cylinder similar to cement kilns, in which it is heated to a high temperature by means of powdered coal blown in at the discharge end. The product varies in size from that of rice to that of walnuts, and forms hard balls more or less thoroughly sintered.

By the Gröndal method the cinder is formed into briquettes of uniform size by an automatic plunger press. The briquettes are loaded on flat cars, which are slowly pushed through a channel kiln 150 ft. to 200 ft. long. Here they are gradually heated until they arrive at a zone of the kiln where the temperature attains the high degree of 2400° Fahr. As fuel, producer gas is usually employed.

The finished product is hard and strong, but porous, so that it is well suited to blast-furnace work and open-hearth practice

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Edward C. Hegeler.—By the death of Mr. Edward C. Hegeler, at Lasalle, Ill., on June 4, the zinc industry of this country has lost one of its most distinguished pioneers and founders. Together with F. W. Matthiessen, he founded the Matthiessen & Hegeler Zinc Co., at Lasalle, Ill., this place being selected as the most satisfactory contact point between the Wisconsin zinc mines and the Illinois coal field. The works started in 1860 and have been very successful ever since. Many important improvements in the zinc smeltry are due to Mr. Hegeler's originality, as the very long gas-fired furnace, labor-saving devices, and especially his muffle roasting furnace, introduced in 1881, which started the manufacture of sulphuric acid as a byproduct.

Furnace Electrode Losses

In our issue for May, page 238, we reviewed and abstracted Mr Carl Hering's paper on furnace electrodes read at the recent Charlotte convention of the American Institute of Electrical Engineers. In the opinion of Mr. Hering we did not, in that abstract, give sufficient prominence to the fact that Mr. Hering now claims that his experimental results supply electric furnace designers with two quantities, E and S', which are independent of any consideration whatever of electrical and thermal conductivities of the electrode material and, for any given material, are functions simply of the temperature drop through the electrode, and are true whenever the electrode tip is at the furnace temperature, so that no heat passes through the furnace tip in either direction.

Mr. Hering obtains these quantities directly from his experimental results. He measures the amperes and volts passing when temperature equilibrium has been reached, the maximum and minimum temperatures, and the dimensions of the electrode. E is then the measured voltage from hot to cold end for this definite temperature drop through the electrode; while S' is the cross-section divided by the amperes flowing and the length for the same conditions of temperature equilibrium at the electrode tip in the furnace. As these values were based only upon experimental measurements, they are altogether independent of any assumptions concerning thermal or electrical conductivity of the electrodes.

Specifications for Foundry Coke *

By Dr. R. Moldenke

The subject of specifications for foundry coke has been long agitated, but very little has so far been accomplished, and this probably because we have been going through just such a change in practice in our coke as we have in our pig irons ever since more liberal and yet more exact views have prevailed. The appearance of a coke no longer carries the weight it formerly did, and all that is now asked is that it melts iron properly and has a composition to do this without serious detriment to the metal.

This change in practice has undoubtedly been the result of the educational efforts of the "by-products" coke people, and as it has meant the saving of much money formerly spent in freight, the foundry industry has been the gainer in every direction on the coke item.

There is still much to be learned by the foundryman regarding the fuel he uses for melting purposes. Too often the blame for poor results is placed on the coke, when the real trouble lies in his melting practice. So that there may be no doubt about the fuel end, it is highly advisable that a good and yet liberal set of specifications for foundry coke be drawn up so that coke bought under them may assure the foundryman that on this item of his daily work he may be reasonably safe. If then he runs into trouble, he will have eliminated the coke from his study of the difficulty.

There are many points which have a bearing on the value of a coke and yet cannot well be added to such specifications. Thus, the cellular structure is an important factor on the working of the coke in the cupola. It must be remembered that in contradistinction to blast-furnace practice, where there is a reducing atmosphere, in the cupola it is oxidizing. In the blast furnace the cellular structure of the coke should be as great as possible, so that every molecule of oxygen is used up in burning the carbon, and with an excess of incandescent carbon make carbonic oxide gas. This means that the lighter the coke—consistent with its carrying power—and the more porous, the better.

On the other hand, for foundry purposes, it is desirable that coke be burned with a slight excess of oxygen, as in boiler practice; so that as complete a combustion of the fuel as possible may result. That is, the gas should be carbonic acid with but little carbonic oxide. Now, this can best be accomplished by presenting smaller surfaces to the passing oxygen. For that reason the smooth surfaces of anthracite, with no cellular structure whatever, give a most excellent fuel for the cupola, though coke can be burned faster.

Considering average coke to be 50 per cent cellular structure, the blast furnace can take care of higher percentages, while the cupola should have them lower. Very light cokes are therefore to be avoided, and rather the denser, heavier ones—provided the weight does not come from an excess of ash—are to be recommended.

On the question of sulphur, we have still something to learn. As between the volatile sulphur and that fixed in the ash of coke, we do not yet know positively why sometimes with the latter we get more into our castings and at other times proportionally less. We know that temperature and slag conditions have much to do with this, but until we are more certain, a differentiation of the sulphur content of a coke is not yet safe, the idea being that the dangerous portion of the sulphur only need be kept down.

Further, there is no doubt but that we will eventually have some means of reducing the sulphur in our molten iron before pouring, and then sulphur in coke will lose some of its terrors.

The following suggestions are presented for specifications for foundry coke, and the following acknowledgments are made. The limits in composition and the base analysis are suggested by Mr. A. W. Belden, coke expert of the U. S. Geological Survey. They are the result of his extended observation with cokes from all the fields of this country. The portion relating to the shatter test comes from the regular practice of the Detroit "Solvay Coke" plant, Mr. Warren S. Baluvelt having kindly sent the writer the specifications on this point.

The subject of premiums for extra good coke as against penalties for running below the mark has long been advocated by the writer as the only fair way of handling the subject. So long as the limit for rejection is placed at a point which will do no injustice to the producer, he will study to get the best results so that he not only gains financially, but also acquires a reputation for it. Penalizing without giving corresponding premiums for special excellence has always been held as against fundamental law, and where the base analysis has been settled upon, and safe physical tests added to the chemical, the results should be good for consumer and producer alike.

The accompanying suggested specifications are presented for discussion herewith, and it is hoped that some progress be made in this important subject.

Suggested Specifications for Foundry Coke.

Coke bought under these specifications should be massive, in large pieces, and as free as possible from black ends and cinder.

Sampling.—Each carload, or its equivalent, shall be considered as a unit, and sampled by taking from the exposed surface at least one piece for each ton, and so as to fairly represent the shipment. These samples, properly broken down and ground to the fineness of coarse sawdust, well mixed and dried before analysis, shall be used as a basis for the payment of the shipment. In case of disagreement between buyer and seller, an independent chemist, mutually agreed upon, shall be employed to sample and analyze the coke, the cost to be borne by the party at fault.

Base Analysis.—The following analysis, representing an average grade of foundry coke, capable of being made in any of the districts supplying foundries, shall be considered the *base* premiums and penalties to be calculated thereon as determined by the analysis on an agreed base price.

Volatile Matter	1.00
Fixed Carbon	85.50
Ash	12.00
Sulphur	1.10

*A paper presented at the Detroit convention of the American Foundrymen's Association.

Moisture.—Payment shall be made on shipments on the basis of "dry coke." The weight received shall therefore be corrected by deducting the water contained. (Note.—Coke producers should add sufficient coke to their tonnage shipments to make up for the water included, as shown by their own determinations.)

Volatile Matter.—For every 0.50, or fraction thereof, above the 1.00 allowed, deduct .. cents from the price. Over 2.50 rejects the shipment, at the option of the purchaser.

Fixed Carbon.—For every 1.00 or fraction thereof, above 85.50 add, and for every 1.00 or fraction thereof below 85.50, deduct .. cents. Below 78.50 rejects the shipment at the option of the purchaser.

Ash.—For every 0.50 or fraction thereof below 12.00, add, and for every 0.50 or fraction thereof above 12.00, deduct .. cents from the price. Above 15.00 rejects the shipment at the option of the purchaser.

Sulphur.—For every 0.10 or fraction thereof below 1.10 add, and for every 0.10 or fraction thereof above, deduct .. cents from the price. Above 1.30 rejects the shipment at the option of the purchaser.

Shatter Test.—On arrival of the shipment, the coke shall be subjected to a shatter test, as described below. The percentage of fine coke thus determined, above 5 per cent of the coke, shall be deducted from the amount of coke to be paid for (after allowing for the water), and paid at breeze prices, previously agreed upon. Above 25 per cent fine coke rejects the shipment at the option of the purchaser. Fine coke shall be coke that passes through a wire screen with square holes 2 in. in the clear.

The apparatus for making the shatter test should be a box capable of holding at least 100 lb. coke, supported with the bottom 6 ft. above a cast-iron plate. The doors on the bottom of the box shall be so hinged and latched that they will swing freely away when opened, and will not impede the fall of the coke. Boards shall be put around the cast-iron plate so that no coke may be lost.

A sample of approximately 50 lb. is taken at random from the car, using a 1¼-in. tine fork, and placed in the box without attempt to arrange it therein. The entire material shall be dropped four times upon the cast-iron plate, the small material and the dust being returned with the large coke each time.

After the fourth drop the material is screened as above given, the screen to be in horizontal position, shaken once only, and no attempt made to put the small pieces through specially.

The coke remaining shall be weighed and the percentage of the fine coke determined.

If the sum of the weight indicate a loss of over 1 per cent, the test shall be rejected, and a new one made.

Rejection by reason of failure to pass the shatter test shall not take place until at least two check tests have been made.

Watchung, N. J.

The Chemical Import Trade of the United States of America.

BY WILLIAM POLLARD DIGBY.

Taking the 10 years ending June 30, 1908, the yearly value of imports of chemical materials has averaged $63,274,000, the maximum being $82,998,000 in 1907 and the minimum being $42,682,000 in 1899. The annual fluctuations for each fiscal year from 1899 to 1908 in the sum total of this trade are illustrated in Fig. 1.

The imports generally greatly exceed the exports. Even in 1908, the year of maximum exports, the imports are about thrice the exports. This, of course, indicates the fact that despite the great industrial strides in this sphere, America must remain for some years dependent upon Europe as a source of supply. The high degree of specialization in the German aniline dye industry on the one hand and the British alkali industry on the other hand are, of course, contributory causes. Another factor is the cheapness of ocean freights as compared with railroad transit, and it is thus apparent that while a factory in the Middle or Western States would be able to snap its fingers at European competition so far as its immediate vicinity is concerned, that factory might be faced with serious competition on the Eastern seaboard.

Examining in detail the sources of the chief materials imported during each of the five years ending in 1908, and again dividing the sources of supply among the six trade zones of Great Britain, British North America, remainder of British possessions, Germany, remaining European countries and South America, we find the state of affairs set out in the following tables:

Sulphate of Ammonia.—This material was not separately enumerated before the fiscal year ending June 30, 1907; about 94 per cent of the imports come from Great Britain, the reason lying in the extensive use of coal gas in that country. The figures are given below:

	1907	1908
United Kingdom	$1,532,127	$1,628,544
Germany	1,574	126
Rest of Europe	44,479	23,458
British North America	25,530	7,983
British Colonies	56,327	128,401
South America		
Rest of world		
Total	$1,660,037	$1,788,512
Average price per pound	$0.027	$0.026

Coal Tar Colors and Dyes.—Germany's world renowned predominance in the aniline dye industry is well exemplified by the volume of her trade with America. The value rose steadily until 1907, there being a decrease of about $500,000 in 1908.

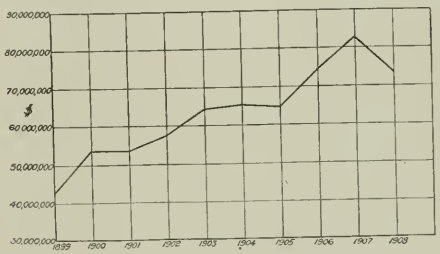


FIG. 1.—TOTAL CHEMICAL IMPORT TRADE OF U. S. A. 1899-1908.

Perhaps now that the German chemical manufacturing companies have been compelled by recent legislation in regard to patents to open works in Great Britain, the values shipped thence to the United States may rise above the insignificant value here recorded:

	1904.	1905.	1906.	1907.	1908.
United Kingdom..	$214,406	$196,393	$177,634	$138,202	$158,997
Germany	3,789,755	4,387,191	4,546,399	4,573,913	3,983,225
Rest of Europe...	888,904	1,101,980	1,014,762	907,989	733,237
Brit. Nth. America.	22,558	19,303	16,440	14,564	7,700
British Colonies...		9	3	12	10
South America....					
Rest of World....	2,880	215	358	321	506
Total	$4,918,503	$5,705,091	$5,755,596	$5,635,001	$4,883,675

Chloride of Lime.—The electrolytic alkali industry is now the one undertaking of an electrochemical nature which is a commercial success in Great Britain; of course, a large proportion of the present exports are produced by the older chemical processes. Cheap raw material, and cheap coal, coupled with a home demand larger than that of any other country are factors tending for the success of this enterprise. Of course, given cheap salt and low railroad freights, there are important development of the electrolytic industry in the United States in conjunction with hydroelectric generating plants. The imports of this material are set out on p. 402. Note may be made in passing of the decline in 1908 which naturally accompanied the decline in the imports of aniline dyes.

	1904.	1905.	1906.	1907.	1908.
United Kingdom.	$585,445	$579,385	$682,554	$710,020	$607,749
Germany	168,483	144,822	154,353	139,531	133,892
Rest of Europe	14,098	51,621	41,497	30,813	12,850
Brit. Nth. America.	4,506	453	945		
British Colonies					
South America					
Rest of World			1		
Total	$772,532	$776,281	$879,260	$880,364	$754,501
Average price per pound	$0.0097	$0.0081	$0.081	$0.0083	$0.0084

Carbonate of Potash.—In regard to all the potassium salts, Germany affords the chief source of supply. For this material and for **caustic potash** separate enumerations as to value were not made before 1907.

Carbonate of Potash.

	1907.	1908.
United Kingdom	$9,098	$18,846
Germany	537,506	522,583
Rest of Europe	212,099	301,641
British North America	1,765	1,947
British Colonies		
South America		
Rest of world	286	119
Total	$760,745	$844,136
Average price per pound	.030	.034

Caustic Potash.

	1907.	1908.
United Kingdom	$15,778	$19,603
Germany	159,312	177,565
Rest of Europe	91,769	55,713
British North America	186	
South America		
Rest of world		
Total	$267,045	$242,881
Average price per pound	.036	.040

Muriate, Nitrate and Sulphate of Potash.—Here again the trade is mainly German. The detailed sources of supply are set out below:

	1904.	1905.	1906.	1907.	1908.
United Kingdom..	$66,933	$65,068	$91,688	$98,765	$22,503
Germany	3,639,378	4,745,794	4,805,788	4,737,266	4,839,264
Rest of Europe....	272,220	265,242	252,327	9,791	24,528
Br. Nth. America.	8,333	2,733	3,179	2,376	372
British Colonies..	396,507	348,350	399,409	386,965	515,070
South America....					209
Rest of world....	8,463	15,323	26,858	26,469	6,587
Total	$4,491,834	$5,442,510	$5,579,249	$5,261,632	$5,408,533

Nitrate of Soda.—Pending the fulfillment of Sir Joseph Swan's forecast as to the coming prominence of this electrochemical industry, the United States in common with the rest of the world must look to the guano deposits of South

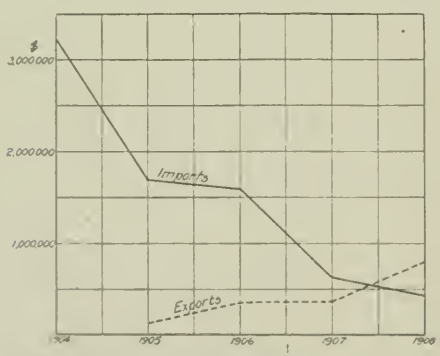


FIG. 2.—COMPARISON OF EXPORT AND IMPORT TRADE IN SULPHUR, U. S. A., 1904-1908.

America for her supplies. Figures are given below of the yearly value of this trade. The steady rise in price since 1904 is significant.

	1904.	1905.	1906.	1907.	1908.
United Kingdom				$46	
Germany		2	66	2	1,354
Rest of Europe					
Br. Nth. America		5,887	2,096	848	230
Brit. Colonies.					
South America	9,259,655	9,677,512	11,115,753	14,040,306	12,545,027
Rest of world.	1				
Total	9,259,656	9,683,399	13,117,887	14,041,202	12,546,611
Av. price per ton	11.6	34.1	35.1	41.1	38.0

Caustic Soda, Soda Ash and Other Soda Compounds.—These alkaline substances whose production accompanies, and whose use frequently precedes, chloride of lime, are furnished mainly by Great Britain. There are, however, marked signs of a diminution of this demand. Of course, part of the retrogression in the value of the imports in 1908 is attributable to the same causes as those which in this year checked the imports of aniline dyes and chloride of lime; and part to the development

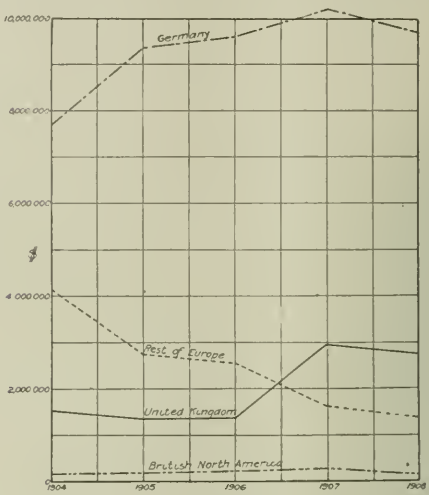


FIG. 3.—DISTRIBUTION OF CHEMICAL IMPORT TRADE, U. S. A., 1904-1908.

of the electrolytic caustic soda industry in the United States. The following table gives the relative values of the supplies furnished by different countries:

	1904.	1905.	1906.	1907.	1908.
United Kingdom.	$433,345	$380,968	$298,152	$320,838	$247,773
Germany	65,445	61,120	85,158	68,780	33,070
Rest of Europe....	62,951	25,047	30,263	20,315	8,240
Br. Nth. America.	219	2,372	754	30,511	159
Brit. Colonies.....					3
South America.....					
Rest of world....	50	62	204	84	71
Total	562,010	469,569	414,531	440,528	289,316

Sulphur.—When reviewing the export trade in sulphur in our last issue we drew attention to the fivefold increase in exports in 1908 as compared with 1905. The figures of the import trade are equally startling. An import of $3,231,000 in value in 1904 decreased to $430,840 in 1908. Whereas, in 1904 supplies from Great Britain were worth $157,251, in 1908 they had fallen to $4,379, while the supplies from European countries other than Germany (of course mainly from Italy) fell in value from $2,855,000 to $211,000.

	1904.	1905.	1906.	1907.	1908.
United Kingdom..	$157,251	$51,342	$55,503	$41,951	$4,379
Germany		1,531	308		40
Rest of Europe ..	2,854,827	1,305,605	1,216,359	305,512	211,144
Br. Nth. America	60		472		485
British Colonies	43				
South America		56	464		
Rest of world.	219,442	327,663	324,457	290,759	214,792
Total	3,231,623	1,686,107	1,597,563	638,222	430,840
Av. price per ton	19.1	18.4	18.1	18.1	16.4

Emery and Corundum.—Although considerable quantities of these materials are still imported, the very striking increase in the export trade for the last year under consideration has made itself felt in a decrease of the value of the amount imported, as the following figures show.

	1904.	1905.	1906.	1907.	1908.
United Kingdom..	$60,847	$72,576	$50,389	$26,082	$18,757
Germany	2,001	1,980	2,026	3,239	416
Rest of Europe	53,662	27,988	13,951	10,421	4,587
Br. Nth. America	41,183	90,908	147,210	151,259	79,831
British Colonies	7,718	2,342	678	1,188	3,485
South America	5,118	1,824			
Rest of world	86,334	98,426	179,272	107,215	214,036
Total	253,953	296,134	413,426	430,304	341,142

Lime.—Accompanying the increase in the exports of this commodity to Canada, the imports have fallen more than 50 per cent in the last five years, although this country still accounts for about half the total trade. The distribution is shown in the following table:

	1904.	1905.	1906.	1907.	1908.
United Kingdom .	$1,329	$2,896	$7,146	$21,102	$18,902
Germany	10,003	12,479	14,074	14,665	13,422
Rest of Europe....	88	59		924	87
Br. Nth. America..	75,475	71,478	85,566	57,308	32,918
South America....					
Rest of world....	3,493	831	1,875	1,236	1,558
Total	90,388	87,743	78,671	95,245	66,890
Av. price per pound	.0017	.0019	.0019	.0024	.0032

A final table is added, showing the total distribution to the various trade zones, and is chiefly interesting in showing how the United States depends very largely on Germany and South America, on the former chiefly for aniline dyes and on the latter for manures. Great Britain being mostly responsible for general commodities.

	1904.	1905.	1906.	1907.	1908.
United Kingdom..	$1,519,656	$1,349,628	$1,382,966	$2,963,999	$2,766,053
Germany	7,675,155	9,354,919	9,608,162	10,235,788	9,704,987
Rest of Europe...	4,146,550	2,777,842	2,569,069	1,634,013	1,375,494
Br. Nth. America..	152,334	193,219	226,662	284,347	131,626
British Colonies...	408,776	341,154	401,035	444,492	646,970
South America....	9,261,773	9,679,392	13,116,190	14,040,306	12,545,239
Rest of world.....	320,663	432,520	533,025	516,084	437,847

Grüner's Ideal Working of a Blast Furnace.*

BY PROF. JOSEPH W. RICHARDS.

For many years the ideal perfection of blast-furnace running has been that promulgated by Grüner. This dictum states that the blast furnace works most economically when the largest proportion of the carbon of the fuel is oxidized by the blast before the tuyeres, and the smallest part oxidized above the tuyeres by the oxygen of the solid charges (reductions).

This ideal is true for most blast furnaces working under usual conditions; we hope to show that it is not true for some blast furnaces under usual conditions, and not true for many blast furnaces under certain unusual conditions.

We will now discuss

(I) Why it is true for most blast furnaces under usual conditions.

(II) Why it is not true for some blast furnaces under usual conditions.

(III) Why it is not true for many blast furnaces under unusual conditions.

I.

The critical part of a blast furnace is the smelting zone. At that place, a high critical temperature must be maintained in order to smelt the charges, and the necessary heat quantity in calories must be furnished. This heat is furnished by the combustion of carbon to CO gas before the tuyeres and by the sensible heat in the blast. These two factors govern the amount of heat available, and its initial temperature. Only CO gas can be formed in the smelting zone; enough carbon must be thus burned to produce the desired smelting effect upon the iron and slag.

In most furnaces the amount of carbon which it is thus necessary to burn, to produce the smelting effect, will furnish far more CO gas than is necessary to reduce the iron oxides in the charge.

It follows from this statement that since more CO gas is necessarily produced in the smelting zone than is necessary to reduce the iron oxides any use of solid carbon as a reducing agent above the smelting zone is a clear waste of carbon. Having more than sufficient reducing agent in the CO gas formed in the smelting zone, this gas alone should be the reducing agent for the charges.

It follows that in all furnaces with the usual quantity of slag to be melted, and the usual temperature of blast, economy of fuel is reached in measure as Grüner's ideal is approached, and that oxidation of carbon by the oxygen of the solid charges should be a minimum in order to attain maximum economy of fuel.

II.

In furnaces with pure fuel, or pure ores, or using little flux, or combinations of these conditions, the heat required for smelting purposes rapidly diminishes. It may very easily happen that by a combination of these circumstances so little heat is needed in the smelting zone (per unit of pig iron made) that the CO gas necessarily produced in developing this necessary heat just suffices to perform the reduction of iron oxides in the furnace. In this case, Grüner's ideal working is still consistent with maximum economy of fuel. We will call this the "equilibrium point."

But, if the heat necessary in the smelting zone requires so little carbon to be there burned to CO gas that the gas produced is not sufficient in amount to reduce the iron oxides of the charge, then Grüner's ideal no longer holds, and carbon can be oxidized by the solid charges with increased economy of fuel.

Such conditions frequently happen in charcoal furnaces working with rich ores, which, therefore, produce very little slag. In such cases, Grüner's ideal working would be wasteful, and greater economy is attained by departing from it as far as is possible.

[A consideration of the usual reactions in the blast furnace shows clearly the economy of using up carbon in direct reduction if it is possible to dispense with burning it all at the tuyeres. The usual reactions are

$$9C + 9O = 9CO \quad \text{(at the tuyeres)}$$
$$Fe^2O^3 + 9CO = 2Fe + 3CO^2 + 6CO$$

The limiting condition for the best attainable reduction by CO gas is

$$6C + 6O = 6CO \quad \text{(at the tuyeres)}$$
$$Fe^2O^3 + 6CO = 2Fe + 3CO^2 + 3CO$$

If some reduction is performed directly by solid carbon the reaction is

$$Fe^2O^3 + 2C = 2Fe + CO^2 + CO$$

It is thus evident that carbon used directly in reduction performs three times as much reduction as carbon first burned at the tuyeres to CO gas. In the usual case, first given, it is even more efficient. The total carbon requirement of the furnace, therefore, decreases rapidly as soon as it becomes possible to use carbon for direct reduction, a condition which becomes possible as soon as the "equilibrium point" has been passed.]

III.

With any given furnace, the heat available in the smelting zone can be increased by increasing the temperature of the blast. If the charge is such that Grüner's ideal working applies to the furnace, then increased temperature of the blast does not disturb that condition until the "equilibrium point" is reached.

After the increased heating of the blast has brought the furnace to the "equilibrium point," further heating of the blast will allow of departure from Grüner's ideal, and with increasing economy of fuel the further the departure.

With any given furnace, the heat available in the smelting zone is increased by removing moisture from the blast. If the charge is such that the furnace is normally at the "equilibrium point," any increase of available heat allows departure from Grüner's ideal with resulting economy of fuel.

Removing moisture from the blast of a furnace at the "equilibrium point" will allow of increased use of solid carbon in reductions with increased economy of fuel.

It has been proposed to send hot blast into the furnace in which some oxides of nitrogen were placed. The splitting up of the nitrogen oxides would *increase* the available heat.

If a furnace were at the "equilibrium point," introducing nitrogen oxides with the blast would allow of departure from Grüner's ideal, with resulting economy of fuel.

Finally, auxiliary heat may be introduced into the smelting

*A paper read before the International Congress of Mining and Metallurgy in Düsseldorf, June, 1910.

zone of the blast furnace electrically. If the furnace is working under the *regime* of Grüner's ideal,, such electrically introduced heat will *tend* to bring it toward the "equilibrium point." Until that point is reached, however, Gruner's ideal holds true for the furnace.

When the equilibrium point is thus reached, further introduction of electrical heat enables a rapid departure from Grüner's ideal; the more electrical heat used the less carbon need be burned to CO gas, and the more carbon can be consumed by direct reduction, with resulting economy of fuel.

At the limit, we can easily suppose *all* the necessary heat for smelting purposes supplied electrically, in which case *all* the carbon charged would be used in direct reduction, and this with great total economy of fuel.

Application of electrical heat to a furnace at the "equilibrium point" will enable the furnace to rapidly depart from Grüner's ideal working with great resulting economy of fuel.

Résumé.

It follows from the above discussion that Grüner's ideal working is true only under certain conditions, which are, however, usually present. Under changed conditions the furnace reaches an "equilibrium point" beyond which Grüner's ideal working is inconsistent with economy of fuel, and greater economy of fuel is attained the further Grüner's ideal can be departed from. Heating the blast, drying the blast, faster working, introducing nitrogen oxides or using electrical heat in the smelting zone are means for reaching the "equilibrium point" and eventually of working in the region past that point where *Grüner's ideal no longer applies, but is a false principle* leading away from maximum economy. In this region, greater economy of fuel is attained the further Grüner's ideal is departed from.

Lehigh University.

A New Process (Spray Process) for the Production of Metallic Coatings.

By M. U. Schoop.

There are known quite a number of different processes for the production of metallic coatings, like the electrolytic or electroplating methods, the "hot galvanizing" method and others. All these require that there be a metallic foundation surface on which the desired coating is produced. Other inherent disadvantages, especially of the electroplating methods, are that the process cannot be carried out at any desired place and that the size of the articles to be plated is restricted, in view of the given location and the given dimensions of the plating tanks.

The following notes will show that these disadvantages are directly overcome by my new spray process for the production of metallic coatings. I have described it in a paper read on April 13, 1910, before the Engineers' and Architects' Society in Zurich and the process was also discussed by Professor d'Arsonval on April 19 before the Academy of Sciences in Paris.

The essential features of my new process are that metal in liquid and highly heated condition and in finely subdivided form is sprayed onto the surface to be coated.

The fine subdivision of the metal may be accomplished in very different ways. The method of subdividing or atomizing the metal is itself not an essential point of my invention, or, rather, discovery. For what I have discovered is a new technical principle, namely, that it is possible to produce from "atomized" metal dense and workable metallic films which may be made at will firmly adhering or detachable.

For the subdivision of the metal, gases or vapors can be used as agents both for exerting a pressure on the metal and for atomizing it. But the subdivision of the metal can also be obtained by simple mechanical devices. For instance, a method which does not use gases or vapors is to pass two jets of liquid metal in continuous stream out of two nozzles in such a way that the two jets meet each other under a certain suitable angle. When they make contact they mutually tear each other up. Further centrifugal nozzles may be used in which the jet of liquid metal emitted under high pressure is atomized by centrifugal force.

According to the experience so far available the easiest and the most rational method of subdivision makes use of compressed gases or vapors. A suitable method of this kind is shown in the accompanying illustration. The metal, a coating of which is to be produced, is molten in the crucible *e* and is then forced out by means of a compressed gas, or air, or vapor, so as to leave the nozzle *g* through a capillary opening.

Immediately after leaving the nozzle, it is acted upon by a stream of gas or vapor of suitable cross-section and is thereby subdivided so that a metallic mist is produced through which the articles to be coated are passed quickly, the passage lasting for one to three seconds.

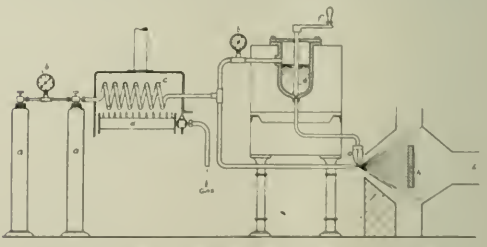
One and the same gas may be used to exert the pressure by which the metal is forced out of the nozzle and also to atomize the metal. This is the case in the illustration where the gas employed is contained in the cylinders *a* and is preheated in *c* by means of the gas burners *d*; *b* and *b* are manometers.

In some cases, however, it is preferable to employ different gases and also different pressures. For instance, to press the metal out of the nozzle, a chemically inert or reducing gas may be employed like nitrogen, etc., while for atomizing an oxidizing gas may be used. By this arrangement it is, for instance, possible to produce lead peroxide coatings in form of firmly adhering and strongly resittant films.

The different metals behave, of course, differently when being atomized. The melting point is of importance, also the greater or smaller degree of fluidity; further, the affinity of the metal to the gas or vapor used for exerting the pressure and for atomizing. The special devices for atomizing, the pressures employed, the preparation of the articles to be coated, and quite a number of other conditions are also of importance.

With certain metals care must be taken in selecting the proper material for the crucible in which the metal is to be molten. For instance, for melting aluminium and its alloys, iron or steel crucibles cannot be used, nor crucibles of fire brick, porcelain or silica, since aluminium has a tendency to enter into combination with materials containing silicon, and to form an alloy.

Numerous tests have proven beyond doubt that when an artificial pressure is used, metals alloy much more quickly and more easily than is the case under similar conditions at ordinary



SCHOOP PROCESS FOR PRODUCTION OF METALLIC COATINGS.

atmospheric pressure. Thus I have found that ordinary gray iron castings and steel castings easily alloy with aluminium as well as with lead or tin if a sufficiently high pressure and a correspondingly high temperature are employed.

The new spray process also permits to produce alloy coatings, but the results so far obtained appear to show that a homogeneous subdivision in the spray and a homogeneous alloy coating result only if the two metals are mixed in eutectic proportions. If this is not the case, one of the two metals being in excess, the lack of uniformity of the coating produced can be easily established by means of microscopic examination. In other

words, my process gives a simple and sensitive method of testing the constitution and nature of alloys.

It is probable that coatings of alloys can be produced by my process if the same surface is simultaneously sprayed by two jets of different metals. It is also possible to produce metallic coatings in which other substances, especially hard ones, are mechanically included, thus producing "near alloys," such as have been recently discussed by K. Friedrich (this journal, April, 1910, Vol. VIII, No. 4, page 190).

A sensitive method of determining the quality and purity of the metallic coating consists in an investigation of coatings produced on mirror glass. For instance, coatings of pure tin produced with the aid of pure inert gases yield mirrors which can hardly be distinguished from the ordinary mercury or silver mirrors. On the other hand, when using compressed air or other oxidizing gases or in general a chemically active gas or vapor, the mirrors produced are more or less dull.

Microscopic investigation of metallic coatings produced by the spray process also permit to determine the size of the little metallic particles sprayed onto the glass—that is, to determine the degree of subdivision of metal in the spray method.

We may form about the following idea of what happens in the spray process. As soon as the stream of metal forced out under pressure leaves the nozzle, it is met by the vapor or gas stream under pressure. It is thereby torn up and changed into a form of mist. Each individual metallic particle when sprayed onto the surface to be coated loses its surface tension and its initial globular form and is pressed out into the form of a little circular film. All the little circular films produced in this way on the surface are automatically welded together into a continuous metallic coating.

When a pressure of 25 atmospheres is used, calculation shows that the metallic particles in the spray have a speed which is more than 20 km per second—that is, a speed which is 25 times that of a bullet from a gun used in the German army. This enormous speed explains why metallic particles while no longer fluid, when sprayed onto the surface, form a closely adhering and homogeneous film of surprisingly high density.

Under suitable conditions and with high gas pressures the density of the coating produced is not very different from the density of the metal under normal conditions. As an example, I mention the result of specific weight determinations which were carried out in the chemical and metallurgical laboratory of Bueler in Zurich. The density of tin, rolled or hammered, was found 7.47; that of cast tin, 7.29, and that of tin coatings produced by the spray method, 7.42. Other similar results were obtained for other metals.

The following fact is of interest: The zone of metallic mist is at a comparatively low temperature, about 40° C. to 60° C., although the molten metal may have a temperature of several hundred degrees and although the gas used for pressure and atomizing (nitrogen or superheated steam) is also at a high temperature. This fact, which was surprising at first, must be due to the sudden reduction of tension of the gas whereby the extraordinary cooling effect of the expanding gas also chills the finely subdivided metal with its enormous surface. It is, therefore, possible without any difficulty to apply the spray method for coating substances with a low melting or ignition temperature, such as paper, wood, celluloid, etc.

The hardness of the metallic coatings produced by my method when measured by the well-known Brinell test is essentially greater than the hardness of metallic films produced by other methods (casting or rolling). This is surprising. In tests carried out by the Bureau of Testing Materials of the Institute of Technology of Zurich, the mean hardness of cast films was found to be 9.6; that of films produced by my spray method, 14.2. This great hardness should be an advantage in the manufacture of cuts and electrotypes in which hardness and resistivity are mainly important.

Finally, it may be mentioned that microphotographic investigation of the structure of coatings produced by my spray method showed, in agreement with the results of specific weight determinations, that porous places or slag enclosures are absent. Where the object is simply to produce an ornamental effect or to make a surface electrically conducting a certain porosity of the coating would not be of great account. But in most cases the object is to protect the surfaces against chemical or physical influence, or against the effect of weather, and in such cases it is, of course, necessary that the coating not only be adhering, but also as dense and homogeneous as possible. This is the case for the metallic coatings of my spray method and they can be subjected to any mechanical treatment like pressing, polishing, etc.

The metallic surface to be coated is first to be freed from adhering dirt or oxides by some suitable process. Further, in many cases, especially where metals are to be coated, it has been found advantageous to preheat this surface before the other metal is sprayed on it.

Iron rods which had received a tin coating by the spray method were subjected to severe tests. In tensile strength tests, up to the breaking point, it was found that the tin coating extended to the point where the break occurred. Tortion tests up to the breaking point showed that the tin coating would not crack off. Great changes of temperature, like heating up to 50° C. and cooling at 25° C., did not exert any appreciable effect on the coating.

I may add that in order to produce a non-corrosive coating of tin, zinc, or lead on iron, it is not absolutely necessary to preheat the piece of iron if only the surface is pure.

There can be no doubt that by means of compact transportable spray apparatus (somewhat of the kind of vacuum cleaners) the possibility is given of providing any piece of iron of any size anywhere with a metallic coating for protection against corrosion—for instance, steel structures like bridges, cranes, etc. This would be a cheap and easy substitute for the usual paint coating.

Applications.

The fields of application of this method are so exceedingly manifold and relate to so many different industries that it is not possible to enumerate them here in detail. In the main there are two large groups of applications, namely, first, the production of adhering thin metallic coatings for ornamental purposes or for the protection of given articles, and, secondly, the production of detachable metallic deposits of a certain form. The deposit of metal on an insulating body in order to get electric conductivity, for instance, in the manufacture of electric condensers, belongs to neither of these two main groups. The same is the case with the production of parabolic and other mirrors by the spray method.

We will mention a few applications belonging to the first group, namely, the production of adhering thin metallic coatings to ornament or protect a certain article. One application is the metallic coating of wooden telephone and telegraph poles at that point where they are most liable to go quickly to pieces—that is, where the pole is inserted in the ground.

Another interesting new application is the covering of wood propellers with metal for aeronautic purposes, whereby the wood is protected effectively against the influence of the weather and besides the resistance due to air friction is very materially decreased.

In the case of balloons a very thin film of aluminium about 1/30th mm thick, applied by the spray method, is sufficient to make the material of the balloon air-tight and gas-tight.

In the case where bottles are to be closed so as to render refilling impossible and at the same time protect the contents from the air, a thin layer of tin applied over the top of the bottle by the spray method is cheaper and easier than the usual methods of hermetically closing the bottles. Paper metallized by the spray method may be used instead of tinfoil or aluminium foil for the packing of chocolate, etc.

The double problem of plating articles with aluminium and

of plating aluminium articles with other metals in an effective and durable manner is solved in a very simple and effective way by the spray method, and there seems to be no doubt that this method will open new fields to the use of aluminium.

As other applications of the spray method, I may mention the metallic coating of products of the textile industry, the manufacture of works of art, the lining of tanks and other vessels for chemical engineering purposes, the metallizing of wall paper, the metallizing of stage scenery, etc.

The second large group comprises the manufacture of detachable metallic deposits in certain desired forms. This class comprises the manufacture of cuts, gives a substitute for electrotypes and also includes the manufacture of hollow bodies, of seamless tubes, etc. It may be that the present electrotyping industry wil be seriously threatened by the competition of the new spray method, because the latter permits to make metallic copies of the original cut in fractions of a minute. The metallic copies thus obtained are in no way different from ordinary electrotypes, except that they are much cheaper. The manufacture of hollow articles of metal is also rendered simpler and cheaper and it may be expected that all industries which at present produce metallic coatings or metallic reproductions will feel in a near future the competition of the spray method.—*Zurich, Switzerland.*

The Elements of Slime Concentration.*

By Walter McDermott.

Introduction.

In considering the progress of concentration methods for the last 30 years, it appears possible to draw certain conclusions which may be considered as elements governing the probable developments in the future. To those who have been in touch with this branch of ore treatment it is very evident that there has been nothing revolutionary in the practice for the period mentioned.

There have been hundreds of inventions; there have been some dozens of commercial introductions of new machines, which were expected to be revolutionary, but which have had only a short period of favor, followed by disappearance from use; and there have been a very few new types which have survived the struggle for existence. The most striking fact has been the very wide and increasing use of two types of machines, the shaking belt and the jerking table.

Inventors, of course, will invent, and they will hope; but very often they will not investigate what has already been done, and those persons who have been in the position to hear of and examine many new inventions must be continually struck with the fact of recurrence of exploded ideas, and the existence of certain limitations which must be considered in a successful slime saver.

It is easy enough to save the coarser particles of any mineral with a specific gravity admitting it into the concentration class, so that invention is more prolific in the means of saving the finest particles. Experience all over the world, on all kinds of ore, has certainly demonstrated a number of facts which enable one to decide pretty accurately the chances of success for any new invention by a mere examination of the principle on which it works, and of its details of construction, without any actual trial on a given ore. Many think that every new invention must be tried before it is judged.

Even on actual new inventions this is only partially true, because life is too short and capital too limited for trial always to precede judgment, but in the vast majority of inventions the novelty is only a novelty to the inventor, the machine itself is an immaterial modification of something already tried and condemned, or it is opposed in principle to established conditions of success.

* A paper read before the Institution of Mining and Metallurgy in London on April 22.

The following notes are an attempt to put down certain elements which to the writer seem established by practice; and to draw certain conclusions which may be of service as indicating the most promising direction for improvements in the arrangement of old machines, or in the designing of new ones.

Definitions.

Although the chief subject matter of these notes is the recovery of slime mineral, some of the processes to be discussed cover a wider range of sizes than what are usually understood by the term "slime," and a number of machines are in present use which deal at one operation with this wider range. To restrict the field of discussion it is proposed to confine this paper to the treatment of material which has passed a screen equivalent to 40-mesh I. M. M. standard, that is, apertures of 0.0125 in.

As the functions and efficiencies of a number of well-known machines are to be referred to, it is desirable to point out that to-day there are practically no master patents in existence on the best known types of machines: that these types are all (with varying details) manufactured by a number of firms; and, therefore, it is possible to refer broadly to *types* of machines without being misunderstood as referring to *makes* of machines. The following classification is herein adopted as covering all the machines necessary to refer to for the purposes of this paper:

(1) Fixed inclined tables (including slowly traveling tables).
(2) Jerking tables (including bumping tables).
(3) Shaking traveling belts.
(4) Jerking or bumping traveling belts.
(5) Shaking side-inclined belts or tables.

Class I.—"*Fixed Tables*" are illustrated by various forms of frames, with wood, canvas, blanket or other surface; by the old Cornish buddle, and by various forms of revolving buddles and side-inclined non-shaking belts. The slow revolution of a buddle or belt, and the intermittent automatic discharge of a frame, do not affect the actual operation of concentration on the surface, so all are included in the class of fixed tables for the sake of simplicity and brevity.

In this class, all the machines depend for results simply on the greater resistance of the specifically heavy minerals to the wash of water down an inclined surface, as compared with the resistance of the lighter waste. The more perfect of these machines do not make a simple division between "concentrate" and "tailing," but they have to make an intermediate or "middle" product in order to yield a clean concentrate.

Class II.—"Jerking Tables."—This class depends for results on a combination of greater movement across a table and greater resistance to wash of water down the table of the specifically heavy particles, as compared with the motion and resistance of the lighter waste particles. For high efficiency this class, like Class I, must produce a middle product, because, in most cases, a hard line of division between concentrate and tailing is not possible.

Class III.—"*Shaking Belts*."—The operation of this class depends on a greater variety of conditions than need be considered in the preceding two classes; and a misunderstanding of these conditions has led frequently to hasty generalizations as to the requirements for most useful efficiency.

The shaking or vanning motion not only settles the heavy mineral on to the traveling surface of the belt, but it prevents the larger particles from clinging to the surface. The upward travel of the belt is in opposition to the flow of water; it tends to keep a bed of the coarser particles on the belt, and this bed checks the too rapid flow of slime material down the incline. The shaking motion, by keeping the bed from packing on the belt, allows of much less inclination on this than on a fixed table, therefore of less rapid current of water. This class makes no middle product, which fact is a limitation on some ores, but a great advantage on others, where only one class of concentrate is required.

CLASS IV.—*"Jerking Belts."*—The actual concentrating process on this is the same as in Class II. but it is carried out on a side-inclined traveling belt, going in the direction of the "throw" of the concentrate, and so the discharge is assisted and does not depend entirely on the "throw." This class makes a middle product also. In some modifications, owing to the nature and frequency of the jerking or bumping motion, a complication is introduced in the relative conditions as the settling motion approaches one of the conditions in Class III machines and has less of the jerking action so important in Class II machines. Such modification merges then into Class V.

CLASS V.—*"Shaking Side-Inclined Belts or Tables."*—This type of machine is not much employed at present, though it has been reinvented a number of times in different forms. A slowly revolving buddle with a shaking or bumping motion works in effect the same way as a side-inclined belt. The shaking motion yields some of the advantages of class III by assisting the settling of the slime mineral, diminishing the inclination of bed and lessening the water required on fixed surface machines; but a middle product is produced, and the longer settling surface and thickening effect of the upward travel of the belt in class III is not obtained.

Conditions Affecting Efficiency.

To understand the conditions necessary for a close saving of fine mineral, it is advisable to begin by examining the action of the simplest (and at the same time the most perfect) appliance in use; and this is the vanning shovel, or some equivalent tool, in the hands of an experienced man. The quantity of material operated on is small; time is of no object, and the action of the tool can be varied to meet the requirements of the moment, so that all the conditions are in favor of the most perfect work.

For proper work the shovel surface must be smooth; and it must be of a very slightly curved section in all directions. There are two distinct systems used with the shovel; the one for throwing up mineral which is not exceedingly fine, and the other system for separating the finest sizes of mineral from the waste. The first employs a motion which is in effect the action of the jerking machines of classes II and IV. The second movement (used for the slime mineral) is the washing back of the waste by a succession of waves of water from the fine mineral—previously settled on to the shovel surface by a shaking motion. This second separation is in effect, but in varying degrees, the action of classes I, III and V machines.

Passing by the question of the saving of the coarser particles of mineral, as being unimportant for the purpose of this paper, the second motion of the shovel may be further explained. To take it in its most refined work, it can be assumed that a sample of tailing, from a concentrator working only on slime, is under examination in order to see how much fine material is escaping. The shovel having its charge of sample (as flowing from the concentrator) is nearly full of water, with fine waste and fine mineral in suspension. No separation of waste and mineral can be at once made, but the first operation is to settle all the solid matter possible on to the surface of the shovel. This settling is assisted by a circular, steady shake of the shovel, which shake at the same time works down the heavier material, however fine, into actual contact with the surface and below the lighter waste. A rapid shake assists in concentrating the settled mineral at one spot.

When the elements of settling time, and settling motion, have sufficiently operated, a series of slower and wider sweeps of the shovel introduces the separating effect of gentle waves of the water, which gradually move the lighter waste, along the shovel, and leave exposed the "head" of the heavy mineral. In this final separating movement there is a combination of the shaking with the washing action, as facilitating the movement of the lighter waste over the surface of the shovel.

The perfect work of the vanning shovel is shown by the fact that it will always give from the most perfect commercial concentrator a definite head of the fine mineral which is going to waste. This perfection of work of the shovel being established, its conditions of working may be taken safely as a guide to the conditions necessary for close slime concentration in practice; and conversely it can be accepted that machines which ignore the conditions essential to good work by hand vanning give only very imperfect results in the saving of slime mineral.

The conditions which are essential to good work, irrespective of the skill of the operator, can be stated as follows:

(a) Sufficient time for settling.
(b) A smooth surface for final separation.
(c) A gentle movement of the washing water
(d) A motion to assist in differential settling and in separation.

These elements deserve detailed examination as to their application to the construction and operation of concentrators of various kinds.

(a) TIME.—It is quite evident, if a very small sample on a shovel must be given a relatively long time to settle before any separation can be effected, that no machine will do close work on slime mineral unless the fine particles are given adequate time to settle from the water in which they are suspended.

As all commercial machines are continuous in operation this function of settling time becomes in effect the ratio of quantity fed to surface and rate of flow. It follows conclusively that if a new machine is produced for which a large capacity is claimed, while the settling surface is relatively small, no arguments based on motion or principles of construction will meet the immediate criticism that an essential element of success has been ignored by the inventor.

It is true that there is a point of difference between a continuous flow of pulp over the surface of a commercial machine, and the settlement of solid particles from a nearly stationary charge on a shovel. This difference lies in the fact that a thin layer of pulp slowly flowing down an inclined machine surface will, by mere frictional contact of some of the particles with the surface deposit such particles. This fact, however, does not affect the general truth of the proposition that feed must be adjusted to settling surface and speed of flow, so as to secure the element of settling time.

(b) SMOOTH SURFACE.—What is well known to be an essential for satisfactory hand vanning—that is a smooth surface—has been proved by general experience in the use of concentrators for slime mineral to be also best in commercial work.

It is true that canvas tables and blankets are used under certain circumstances; but such surfaces cannot be considered in the list of perfected concentrators; they are either used for a special purpose (as in the case of blankets for the saving of fine gold) or as supplementary catchers of a little valuable mineral from the tailings of more perfect machines. It is a constantly recurring fallacy that a rough surface must be good for saving slime mineral. The fact that it is also good—and in fact relatively better—for saving fine waste quite annuls the supposed benefit of its retaining power on the mineral. If a clean concentrate is required the current of washing water must be so great, to prevent the waste clinging, as to involve loss of fine mineral; and if clean concentrates are not required, the machine is not a concentrator: it is only a contrivance, like a settling pit.

The history of past practice and an inspection of the most successful working of the present day show that the advantage of a smooth surface is generally recognized. Tables, buddles and belts have been made of wood, glass, slate, marble, cement, rubber, etc., all with a view to getting the necessary degree of smoothness.

The extraordinary clinging power of very finely divided mineral, when once settled on to a smooth surface, is seen in the action of any of the machines of classes I, III and V when an excess of wash water is turned on: for the lighter waste and the larger particles of mineral are driven off long before the finest mineral can be moved from the surface it has adhered

to. To free a smooth vanning shovel from fine mineral after a test, it has to be immersed again and again in water, or put under a strong stream. The possibilities of rough surfaces, varying all the way from coarse riffles in a launder to a smooth close canvas, have been exhausted long ago; and therefore a new concentrator, for which some form of rough surface is claimed as the chief advantage, can be safely left for time to dispose of.

The action of stratification within definite cavities by a shaking motion is quite different from the mere arresting effect of a rough surface above referred to.

(c) Speed of Water.—The bad effect of an excessive rate of flow of water over a concentrator surface is twofold. A rapid current is, of course, in the first place unfavorable to the settling of fine mineral; and, in the second place, it is apt to refloat and carry off some of the particles which by rolling contact with the surface of the concentrator have succeeded in settling.

It is elementary that a too great inclination of a concentrator and an excess of water are both unfavorable to close work. It is unnecessary, therefore, to illustrate the effects of these conditions, combined under the head of "speed of water"; but the importance of the point is sometimes overlooked in the comparison of certain types of machines for certain work, and in the designing of new machines for obtaining large capacity.

(d) Settling and Separating Motion.—It is difficult to explain to those unused to hand vanning how important is the shaking motion imparted, and in how many varying ways it can be used, guided by the eye according to how the mineral and waste are settling and distributing over the surface of the shovel. The whole success of the operation, and the clear exhibition of the "head" of fine mineral separate from the body of the lighter waste, depend on the way the settling is accomplished, and on the way gentle washing waves of the water are made to travel. A proper motion keeps the lighter waste in semi-suspension in the water, free to move over the clinging, sluggish mineral particles; whereas if both waste and mineral are allowed once to settle fully on the shovel (to pack) the waste cannot be moved except by such a strong wave of motion as is certain to pick up and move much of the fine mineral also.

In considering the effect of the shaking motion in actual concentrators in practice, it will suffice to take a machine of the shaking belt class, and note the effect of stopping the shake. The machine becomes at once of the class I type; it is a self discharging frame. Its inclination must be increased, or water supply made much larger; its loss of fine mineral will be much greater, and its concentrates will be less clean than when running in its normal way on the same feed of material.

Another very important effect of the shaking motion is that unclassified material can be treated in a way not otherwise possible, both on the shovel and on shaking belt machines; and this will be referred to more fully below in treating of classification.

Direction of Probable Improvements

If the above evidence and conclusions are accepted, it is perfectly justifiable to judge any proposed new concentrator by its agreement with, or departure from, certain principles. The design will doubtless fall within one or the other of the five classes described above; unless it falls into one or other group of certain well-traveled and exhausted lines, such as rate of settling in water, resistance to upward flow of water, centrifugal separation, separation by air, separation in a denser medium than water, and flotation processes.

Putting aside the last named group (flotation processes) as not really in the nature of concentration by specific gravity, and so not within the limits of this paper, it is worth while pointing out one of the great obstacles to the perfection of a machine for saving fine mineral by any of the other principles just mentioned.

No system of practical classification has yet been invented which does not in the end leave a great distance in the sizes of particles contained in the final class, the class in which the great loss occurs in concentration. As a consequence, separation by rate of settling, by upward flow of water, by a denser medium than water, all become imperfect and are usually quite impracticable.

It is well known that air separation—already tried in dozens of different ways—is most unsatisfactory on dust, and it is usually inapplicable for commercial reasons.

Many have been the attempts to employ centrifugal force, which on first thought seems eminently adapted to the saving of fine particles by its power of intensifying the function of specific gravity, but in practice its application becomes impracticable. With such fine and unequally sized particles as are under discussion, a separation by mere form of trajectory is not possible; and, as a consequence, the action of a centrifugal concentrator must be almost necessarily one of packing the material against the sides of the revolving machine.

This action of packing, or building up on the walls, is in its very nature absolutely opposed to any continued separation of particles of differing specific gravities when once fairly started, and hundreds of failures witness to the apparently insuperable difficulties of making a continuous discharge to this class of machine. Attempts which have been made to combine a washing separation by water (requiring a freedom of motion between particles) with the packing action of centrifugal force seem to point to an inherent antagonism between the two forces which would require a delicacy of adjustment impracticable in practice.

From the above it seems probable that new concentrators which require serious consideration will be modifications of established types, and to meet the expectation of doing work comparable with that of existing machines they can pretty safely be judged by the extent to which they meet conditions which have been referred to.

Of course, all new machines are said to do wonderful work; but history shows how exceedingly difficult it is to get at comparable facts, and how very unreliable as a guide are the testimonials and figures of millmen in the matter of concentration. The great variety of practice all over the world, even on similar ores, in both the use and distribution of different types of machines, shows at once how difficult it is in this branch of milling to get rid of the personal element, and the limiting effect of special experience; and again, the influence of mere fashion is often apparent in the arrangement of mills.

It may be mentioned here that on ores from which a single concentrate is required (not, for instance, concentrates separate of galena and zinc blende from waste) it is not advisable to make a middle product. A middle product may be needed when the coarsest particles carry attached mineral and need regrinding; but on material finer than 40 mesh this is not often the case. Many concentrators do make middle products, and it is certain that losses are incurred, and are often obscured by the fact that the middle product must be redressed; for a secondary loss is thus involved, and must be added to the loss in the tailing of the first operation.

There are often good reasons in practice for using a middle-product-making machine, but it should not be overlooked that in principle, and for the conditions assumed, there is a definite objection to the system; and, therefore, in designing improvements over existing machines it is a point which will count in actual commercial results in the end. Questions of capacity, cost and expediency often influence the choice of the details of arrangement of mills; but the question under discussion is one of individual efficiency of work, which should be well weighed in the first instance. It is known that in modern concentration mills losses are often high; it is therefore no heresy to refer in detail to certain probable causes of loss in existing practice.

To sum up the conditions which seem from the above to be most favorable to the evolution of any new concentrator which may reasonably be expected to prove an advance over existing

machines, it can be said that there must be a large and smooth surface; a regular settling motion which does not jar the finest particles from their contact with the surface when settled, and a delivery of clean mineral without the production of a middle class. The provision of a proper settling motion insures the other necessary conditions of slight inclination of surface, and the minimum quantity of water in pulp and for washing purposes. For example: a revolving buddle, which as ordinarily used, will require an inclination on its surface of 1½ in. to the foot, can, with a proper shaking motion, work to better advantage with ½-in. fall, and will need less water.

Classification.

The necessity for classification before concentration is so marked in some cases and with some machines, that it is accepted by some people as a comprehensive dogma with full scientific warrant; but actual experience has shown very clearly that in certain cases it is distinctly undesirable, and in other cases it is so imperfectly applied as to largely neutralize its advantages.

Bearing in mind the limitation in this paper to the consideration of material finer than 40 mesh, it is a clearly demonstrated fact that machines of class III (shaking traveling belts) give better results both industrially and theoretically without classification. At intervals of some few years new teachers arise to point out that the treatment of an unclassified pulp is in contravention of elementary laws of scientific concentration on any machine. It will be worth while here to push this argument to a conclusion, to see what, if anything, science has determined, and what practice has demonstrated.

In what scientific formula has it ever been attempted to lay down the relative values of the functions of specific gravity, inertia, surface-resistance, speed of settling, friction of grains, rate of water flow, and adhesion to machine surface, in the case of a mixture of particles from 40-mesh aperture to impalpable slime, on a machine of class III type?

Such a formula might amuse a mathematician; it would be beyond the comprehension of the writer and of most people, and it would carry no conviction whatever in a practical sense. It is only necessary to fall back on certain results and details of actual operation, to show that some of the functions above named are so different relatively on different types of machines, as to make conclusions drawn from one machine inapplicable to the results to be expected from another machine. An illustration will help to make this clear. If a silver or copper coin be put on a machine of either I, II or IV class, it will be delivered with the concentrate. If the coin be put on a machine of class III it will not be delivered with the concentrate. Any scientific formula which correctly accounted for these opposed results could never be construed into a dogma on classification in general, since the effect of the function of inertia is quite different in the two cases.

It is known that classification is not necessary for a sample treated on the vanning shovel; and as this treatment is taken above as a standard of work and as a guide to investigation, it would seem to indicate in advance that classification is not a fixed law of perfection in concentration.

Furthermore, the experience of thirty years, on all kinds of ores, in many different mining fields, has established most definitely the fact that there is a distinct advantage in treating unclassified pulp on machines of class III. The coarser particles form a bed on the belt owing to its upward travel; and the fine slime instead of flowing down in unchecked waves over the belt surface (as it would do if treated alone) flows more slowly, by a filtering process in part, and so has better chances for settlement of the fine mineral.

In practice, the blind adherence to the belief in classification has led to some curious results. A very unsatisfactory method of classification is usually adopted, that of simple pointed boxes. By this system the coarser or bottom discharge is of necessity accompanied by a stream of the finest slime; and this last goes to a class of machine not adopted to its treatment. In some modern mills, arranged especially with a view of keeping coarse and fine apart for separate treatment, the first step, that of classification, is thus most imperfect. A step in advance is made in some cases by the introduction of upward flow classifiers, with a disadvantage from the continued dilution of the stream of pulp for the final treatment. The experience obtained in some of these mills has led to the supposed discovery that machines of class III are not proper machines for slime treatment, and they are therefore given an intermediate product of the classification system.

This conclusion is quite opposed to general experience; because machines of class III were originally developed for, and are continuously used for, the direct treatment of ores crushed fine in batteries, and containing small quantities of finely divided and brittle minerals, which are valuable enough to demand very close saving. Such ores, crushed under such conditions, must need close saving of slimes, and must make much slime value, and it would be a remarkable thing if a class of machine which is so successfully used for such work should be now discovered to be not adapted to slime saving. The explanation of this erroneous conclusion is simple, namely, that by attempting classification, and then experimenting with the very diluted slime alone, the machines of class III are not working under the best conditions; and they have, furthermore, very little capacity when so improperly limited as to size of pulp and when so overcharged with water.

In most of the large modern mills of the United States and Mexico, machines of class III are used for the finest sizes, and the slime losses due to imperfect classification already referred to are sometimes diminished by a retreatment on machines of class III of the tailings of class II machines. In some of these mills the finest overflows of classifiers or thickening boxes are run into settling pits, and treated as a concentrate for addition to richer products for the smelter; but such treatment is not available when clean products are essential to subsequent treatment, for example, with tin ore. Of late there has been a tendency to the use of fine screens in place of some of the water classifiers in a mill, to this extent avoiding the slime discharge of the ordinary classifier with the coarser particles, while making a separation by size alone, instead of by equal falling particles.

It is a very usual thing in mills to treat the tailing of one slime machine by another machine of the same or different construction; for example, buddles are often made double, or a convex surface may be followed by a concave one; and frames or canvas tables may be used in a long succession, each saving a little.

This double treatment can be used with great effect on machines of class III, treating unclassified pulp, but is very seldom attempted. The loss in the tailing of the first machine is in part saved by the second, and in some cases (for instance, with tin ores) where the pulp from the stamps is fine enough to go direct and unclassified to the first machine, the double treatment will be most effective as compared with a classification and treatment on several types of middle-product machines.

To get the best results from the second machine, it is advisable to use a large pointed box between, to get rid of some excess of water introduced by the first chamber; and an increase in belt surface on the second row will add to the efficiency. By this system the slime mineral has a double chance of being saved; there is no excessive dilution of the slime; clean concentrates can be produced, the losses due to rehandling of middle products are avoided, and the whole plant is the most simple and automatic in character.

If the tailing of the second row of class III machines contains enough fine mineral to be worth additional treatment, a properly arranged classifier will deliver this mineral in the fine overflow and in a condition suitable for treatment on revolving buddles or frames, or other wide surface appliances suitable for poor material.

When a screen is used in the battery too coarse for effective work by machines of class III, an additional finer screen can be used to take out one or more sizes of particles for machines of class II, or even for jigs.

* * *

In the discussion which followed, several speakers emphasized that the paper treated not with the concentration of slimes alone, but with the concentration of slimes with fine sands. It was also pointed out that where purely mechanical concentration failed other methods, like electromagnetic or electrostatic separation, or flotation processes, might help. But the author's paper was restricted to a discussion of mechanical separation by means of water.

The advisability of mechanical separation was also discussed by various speakers. Two speakers also expressed the hope that a method of concentration, based on centrifugal force, may still be developed.

Mr H. L. Sulman said that, considering the four factors alleged by the author to govern the deposition of slime-material, viz.: (a) Sufficient time for settling; (b) a smooth surface for final separation; (c) a gentle movement of the washing water; (d) a motion to assist in differential settling and separation; he thought they would be in general agreement as to the value of the first three for slime deposition. In regard to (d), however, this seemed a more debatable point; in most of the recognized types of mineral slime-savers that factor was absent.

Mr. Sulman thought that the majority of recognized slime-concentrators were included in the author's class I. Concerning the class III type, he said that it was a machine which had the undoubted advantage of dealing with a wide range of particles, often working better without the previous close classification demanded for the best results with other types of vanners, its bed of coarser concentrates acting as an intercepter for some of the slime mineral. It was admitted that vanners of this type III saved more slime-mineral than did machines in other classes, which dealt equally with concentration of coarser particles from sands. But he doubted that on slimes alone (i. e., without a bed of coarser concentrates) the class III type would equal some in class I as a saver of fine mineral particles.

Mr. Sulman also remarked that the class I machines, which gave the closest savings of slime-mineral per se, were provided with no differential motion. He concluded, therefore, that the author's contention as to factor (d) was not always borne out in actual practice; or it might be that the ideal machine, in which the closest slime-concentration was to be in combination with a different motion, had yet to be invented.

Mr W R Thomas spoke on the concentration of tin ores in Cornwall. They now had, in Cornwall, examples of classification before concentration and concentration before classification; in the former, in some cases, the spigot going on to the jerking tables and the overflow to vanners, while in the latter case the whole was taken on to the vanners; of one instance he knew, where the whole mill product, after going over the vanners, was classified into sands and slimes, and the sands were put over a jerking table. In Cornwall, they used either simple pointed boxes or upward current classifiers. His experience during the past few years had been that where the latter form of classification was in use, by varying the amount of water they could vary the character of the pulp on both the tables and the vanner; if one increased the amount of water which was coming from the upward jet, one could throw sands onto the vanners.

He had noticed that the tin ore recovered by treatment of the middle head from the jerking table was much finer than that recovered from the head from that table. The vanning shovel also showed that a small quantity of similar sized hard grained tin was found in the tailings from the first machine.

The catching of the coarser particles of mineral was not a difficult matter, but to catch the finer particles contained in the slimes was a most serious problem. To close a concentration was likely to increase losses. In some of the best cases, where there was comparatively little associated mineral, the practice was to concentrate it up to 25-35 per cent black tin, which on a pulp having an assay value of 28 to 35 lb. a ton meant a concentration of 20 to 1. To attempt to procure a clean concentrate would appear to result in a loss which otherwise might be obviated.

It was probable, however, that the main proportion of the loss was confined to the slimes, and it also appeared to be evident that both tables and vanners, with or without existing methods of classification, had failed to catch a fair proportion of the mineral values contained in the same. There, then, was the field for those studying the subject-matter of the paper, slime-concentration. The old school, who were responsible for, and who had so great an attachment for, the vanning shovel, originated the idea of the fixed hand frame, which had since given way to the automatic frame. They saw the necessity of running the slimes in a thin layer over a large area of inclined wood frames which secured an even flow, and where an acceleration or diminution of speed did not take place, as was the case with either the concave or the convex principle applying to both buddles and revolving tables. Owing to the comparatively low value of the slimes step concentration became a necessity. Many of the slimes were worth only perhaps 3 or 4 lb. of tin per ton. That had been followed up by applying a revolving table, either concave or convex, or a combination of both, for the treatment of the pulp from the frame, from which latter it was possible to obtain a fairly high grade concentrate.

The application of such settlers, referred to by the author for dealing with slimes, was very useful, and had to a large extent dispensed with the old slime pits, which were dirty, beastly things, and which took up so much space.

There appeared to be every reason to believe that a slime plant of that description, with automatic frames and revolving tables, systematically laid out, would catch a fair proportion of the tin escaping from the modern concentrators now in use, and that at an extremely low rate of working cost.

Mr. R. E. Commans thought that the types of machines best suited for the concentration of slimes and fine material are not those having a jerking or bumping motion, but machines of the shaking, traveling belt type—that is to say, machines on the lines of the well-known Frue vanner. When a small percentage of very fine material has to be separated, canvas tables of self-acting slime frames of large area, over which the pulp can be distributed in a thin film, might, as suggested by Mr. Thomas, be used with advantage, but in most cases it will be found that a sufficiently high percentage of the rich mineral can be secured if the de-watered pulp from the first set of vanners is passed over a second set.

Mr. S. J. Speak pointed out that the paper explained why different types of machines possessed particular advantages, and this might be amplified by two examples, from his own experience. Both cases were gold mills, working on sulphide ores; in one case good concentration rendered the tailings too poor for cyanidation; in the other case this was not so, but it was desirable to make a concentrate in order to save the galena in the ore. The first mill was fitted with eight Frue vanners and two jerking tables; and experience proved that it was best not to classify before concentration, and that better results were obtained by discarding the jerking tables altogether. The second mill had only Frue vanners, but as it was desirable to throw as much zinc blende and pyrites into the tailings as possible, they were obviously not suitable machines for the work, and jerking tables would have been much better.

Dr O. J Steinhart emphasized that the broad line underlying the whole principle of concentration was that of equal falling particles in water, and stood up for Rittinger's law.

Mr W S Welton emphasized the necessity to ascertain what the actual losses in slimes concentration amount to under present conditions. For a number of years he carried out a

series of experiments with a view to discovering the causes and amount of loss in concentrating ores. He found that from 8 to 20 per cent of the tailings were carried off by the water employed, and that this portion was very much richer than the sands recovered as tailings. In consequence, the assays of tailings gave a very imperfect idea of the amount of loss in concentration. The collection of the float tailings on a large scale was found to be impracticable, and the actual loss had to be calculated as the difference between the assay value of the feed and the values recovered, divided over the weight of feed, less the weight of concentrates. It was then found that the loss, calculated as a percentage of the assay value of the feed, was as follows, in a concentration to four times the value of the feed:

With a silver ore sands, 38 per cent of the value of the feed.

With a silver ore slimes, 55 per cent of the value of the feed.

With a copper ore sands, 29.7 per cent of the value of the feed.

With a galena ore sands, 16.98 per cent of the value of the feed.

It was found that the percentage of loss on the feed remained the same for any given degree of concentration of the values, whatever might be the value of the feed; but increased with greater degrees of concentration, when there was no regrinding of the ore during concentration—but with regrinding of the ore there was a considerable addition to the loss. The results in concentration determined experimentally by him were confirmed by the results of concentration on a large scale, and again, recently, by independent experiment.

Mr. J. H. COLLINS referred to the treatment of the impalpable slime of the tin mines. The only successful method of separating the particles of tin-oxide from the waste in true slimes has been by standing them upon flat and slightly inclined surfaces, from thin films of flow, without shaking.

The plant commonly employed with this object, consisting mainly of the well-known "self-acting" and "revolving" frames, is cheap in operation and moderately effective, but somewhat costly in erection and inconvenient.

The author, Mr. WALTER MCDERMOTT, in his reply, said that while he was referring to the difficulties of saving slimes, since they all admitted that the coarser particles could be saved, it was saved in certain cases in conjunction with fine sand. Part of the argument he intended to bring out was the question whether classification in certain cases was advisable, and whether machines of a certain type did not do better work when there was some finer sand mixed with it, and also that those same machines, by a double treatment, would diminish the loss which every machine must make in practice.

Mr. McDermott then replied to those speakers who had referred to frames and other simple appliances being used after other machines. He considered that that was no argument whatever, because if one used the other machines again after the frames one would still save something. Because one machine saved something from the tailing of another, that was no proof that the second machine was better than the first, because one could reverse the order with similar results, and put it again over the first machine and still save something. So that the argument drawn from the use of a simple machine like frames or blankets, or anything of that sort, after other machines, did not prove anything as to the relative adaptability of the two kinds of machines for saving fine slimes.

When large surfaces were required, treating poor material, it was natural that the cheapest kind of machines would be used. To run it over a series of frames over a large surface, and to spread the layers out thin, was, of course, to facilitate the saving of a certain amount of fine material; and, therefore, those simpler machines were always used after more complicated machines, although people did not think of using them to begin the concentration with; they used them simply as auxiliaries.

His argument was that it would be better to use a superior machine twice, rather than go to a less perfect machine which had the disadvantage of making an impure concentrate, and that, too, with the disadvantage often of a middle product. That particularly held in the case of material like tin, of which Mr. Thomas had spoken. It did not hold in the case of some copper ores, where the cost of smelting and the conditions as to fluxing enabled one to treat material as a concentrate, which, talking simply in abstract terms of close concentration, would not be considered a concentrate. Of course, if one ran the product of a stamp mill into large holes in the ground, one would save 95 per cent or 99 per cent, but it was not concentration.

Mr. McDermott also replied to Mr. Sulman, who had thought that machines of fixed surface (class I) are best for treatment of pure slime, because they are in general use for this purpose; and who had argued from this that a differential settling motion is not necessarily an advantage for a slime concentrator. One object of the paper was to dispute the correctness of some modern practice, so that the appeal simply to practice is not convincing; and furthermore, American practice does not support Mr. Sulman, because class III machines are very commonly used in preference to class I for slime.

In the paper it is suggested that while class 3 machines are preferable to those of class I, even on pure slime, the former machines will work to still better advantage if some fine sand accompanies the slime. There is a sufficient reason for the common use of fixed surface machines in the fact that a large surface must be used for efficiency in any slime treatment, and economy in first cost of plant largely influences practice. The subject under discussion, however, is simply perfection of work in concentration.

Mr. McDermott finally stated that he had no intention to object to classification in general, but rather to point out that in some cases of direct fine crushing it is unnecessary and disadvantageous; that in other cases of coarser crushing it is better to take out one or two sizes by screens without further waste classification; and that even with coarse crushing, classification may be carried further than is either necessary or advisable.

Standardization of Assay Values.

The committee for the standarization of assay and bullion values appointed by the Institution of Mining and Metallurgy has formulated rules for the guidance of assayers, metallurgists and mining men in reporting results of assays, metallurgical tests and similar operations where uniformity is desirable. This is for the purpose of avoiding ambiguity and of minimizing the possibility of misinterpretation or misuse of reports.

Reports of assay results, extraction tests, metallurgical and ore-dressing trials, etc., are sometimes so worded as to permit a wrong interpretation, and the committee urges the necessity of definite statements as to the nature and condition of the sample as assayed, and, where advisable, an indication of the method of assay. Thus, in the case of such metals as copper, tin, lead and antimony, where both "dry" and "wet" methods of assay are in use, results should be reported as obtained by the "fire" assay or the "wet" assay, with further details as to method, if judged advisable. Similarly it should be stated whether the sample was assayed in the condition in which it was received, or air-dried, or dried at — deg. C.

In reporting that an assay for such a constituent as tin, arsenic or tungstic acid corresponds to so much "black tin," "white arsenic," "wolfram" or other recognized salable product, the report should give the assay result for such constituent, together with a definite statement as to the figures used in calculating such salable product. Thus, an assay showing 1 per cent of metallic tin might be reported as 1 per cent of metallic tin, equivalent, if all could be extracted as such, to 1.5 per cent of "black tin," containing 66.6 per cent of its weight of metallic

tin. Similarly, in reporting results where assay values might be mistaken as representing available produce, it should be stated that such are merely "gross" assay values, a statement of the actual "commercial" values being added whenever possible.

In such matters as ore-dressing, ore-treating processes, furnace work, etc., where so much depends on the way in which results are reported, on the nature of the sample and on the method of taking it and in its relation to the other samples, the committee emphasizes the necessity for careful phraseology and for the inclusion of exact data, as also the necessity for care in preparing, describing, labelling, sealing and otherwise insuring the correctness and authenticity of samples submitted for tests and assays.

The following recommendations are submitted for adoption by the Institution:

(1) Assay reports shall state the exact condition of the sample when assayed.

(2) Assay values of gold and silver ores and products shall be represented in pennyweights and decimals, or in ounces and decimals, and not in ounces, pennyweights and grains. They shall be expressed in terms of fine gold and fine silver respectively, and not as "bullion."

(3) Assay values of alluvials shall be reported in grains and decimals of a grain of "fine" gold, or in pence (at 2d. per grain of fine gold), or cents per cubic yard. It is recommended that in the absence of specific information, 1 cu. yd. of ordinary alluvial, excluding boulders, be taken as equivalent to 3000 lb. (1.5 short tons).

(4) In reporting assay values of cyanide and other solutions, the results shall be given in parts by weight in a stated volume of the solution. In the case of cyanide solutions, the use of the "fluid ton of 32 cu. ft." is recommended. It closely approximates to 2000 lb. and is in common use.

(5) When it is necessary to state or estimate the money value of an ore, etc. (other than of gold), it shall be accompanied by the assay value, and the basis on which the former has been calculated from the latter shall be stated.

(6) Laboratory sieving tests shall be made with the I. M. M. standard sieves, or, when other sieves are used, the widths of the apertures shall be stated.

The following recommendations on weights and measures have already been adopted by the Institution:

(a) The word "ton" shall represent a weight of 2000 lb. avoirdupois (29,166.6 oz. troy). It is advisable to abandon the use of the words hundredweights and quarters, and to express fractions of a ton in pounds or in decimals of a ton.

(b) The word "gallon" shall represent the Imperial gallon measure of 10 lb. of water.

(c) Temperatures shall be expressed in degrees Centigrade.

(d) Returns of gold and silver shall be expressed in terms of fine gold and fine silver respectively, not as "bullion."

(e) Gold contents of ores, etc., determined by assay, shall be expressed in money values as well as by weights; and in this connection the value shall be taken (as a convenient constant) at 85 shillings, of $20.67 U. S. currency, per troy ounce of fine gold.

All of the foregoing recommendations are meritorious and worthy of general adoption. For the most part they are already observed by the more careful assayers in the United States.

Hypochlorite for Water Purification.—In the *Engineering Record* of June 25 Mr. Francis D. West remarks that we appear to be rapidly approaching the "age of hypochlorite." Since the successful test of Fuller and Johnson, at Boonton, N. J., several cities have adopted this method of purifying their water supplies. It is excellent as an extra precaution, but is not a permanent substitute for a carefully operated filter plant. But the investigation of the subject is not yet exhausted. Mr. West urges further study of the use of hypochlorite with rapid uncoagulated filters and further suggests that the test of oxygen consumed is of value to operators of water disinfection plants.

Natural Gas and Its Production.

By Harold A. Danne.

To an Australian in the United States, the phenomenon of naturally generated gas issuing out of the ground ready for use is one of the most remarkable sights that can be seen.

This natural resource wherever it occurs is gradually converting many districts from second-rate farming communities into thriving cities abounding in wealth and manufactures and all that such features mean to the cosmopolitan population attracted by them.

West Virginia abounds in widely distributed bituminous coal fields and in the petroleum industries, and particularly illustrates the preceding features, inasmuch as in this State are found some of the steadiest gas wells and thermally richest gas.

Without going into the history of this natural gas and its discovery in the West Virginia fields, which involve George Washington and some of his contemporaries, it suffices to say that after the pioneers had drilled the first few shallow holes and had overcome the very great obstacles due to imperfect drilling rigs, reels and tools and ignorance of geological conditions, the industry has almost continuously expanded, particularly since the introduction of long pipe lines connecting the fields with refineries and with large cities, such as Cincinnati, Pittsburgh, Cleveland, and numberless small ones, and the extension and exhaustion of the products are now threatened in a period variously estimated at from 20 to 50 years.

Australians are accustomed to the use of artificial gas at about 4 shillings per 1000 cu. ft., and regard it at even that high figure as economical and useful, so that the wonder is in the West Virginia fields that at about 2 pence per 1000 the gas will not be exhausted the soonor, more particularly since it is serving the various purposes of hundreds of large factories and 15,000,000 to 20,000,000 people.

With the advent of the modern high-efficiency large-powered gas engines in increasing numbers, the amount of gas used for generating power is also becoming a very serious factor.

As an important feature in this use of the gas it should be borne in mind that at the well mouth the gas is saturated very often with light hydrocarbons, such as gasoline, having passed through or generated with, in very many cases, the strata known as oil-bearing pools. It, therefore, gives a very high thermal efficiency, some of it running as high as 1134 b.t.u. per cubic foot.

When it is compressed, either purposely or during the pumping operation carried on to force it along some of the pipe lines, it deposits its rich contents, and its thermal efficiency is decreased thereby until when it reaches some of the distant cities it gives often only 900 b.t.u. per cubic foot.

Some operators who have purposely compressed the gas have stated that it is neither reduced in volume nor in thermal efficiency, which might be explained by saying that their measurements and other data were faulty. The apparatus used in this gasoline extraction industry consists of double coils of piping similarly built to the old-fashioned ammonia condensing coils. Through the inner coil the natural gas compressed at about 200 lb. to the inch is allowed to pass. Thence it passes into the space between the inner and outer coils after having been duly cooled and the reduction of temperature due to its consequent expansion in many cases amounts to 50° Fahr. and so affects the molecular state of the gas that the hydrocarbons are deposited, one might almost say pressed out, in considerable quantities. I have it on authority of one operator that this juice, or fat, as it is called, often amounts to 2 gal. per 1000 cu. ft. of gas at 72° Fahr.

The natural pressure, or rock pressure, as it is often called, has been turned directly to account in running steam engines and pumps without combustion of any kind. This is the more feasible as such pressure varies from 500 lb. to 1200 lb. per square inch.

A peculiar phenomenon is noticeable when an engine so driven

is reversed. The lack of resiliency of the gas makes action so quick as to be almost uncanny, and if not carefully done is liable to strain the working portions of the machine.

A State law has recently been enacted prohibiting these wasteful methods and also compelling the shutting in of wells not in use.

A fact well known to all gas operators is that the natural pressure all over the States invariably falls about 25 lb. whenever the wind is blowing from the northwest. Why, no one has yet explained.

In the natural wells, i.e., in wells which were not artificially drilled, the gas pressure almost invariably rises very greatly at or before seismic disturbances and was particularly noticeable before the earthquakes of Martinique and San Francisco. This fact should be borne in mind when the theory of production of the gas, as formulated below, is considered.

The gas closely approximates marsh gas, CH_4, in composition and requires about 11 cu. ft. of air for the complete combustion of 1 cu. ft. of gas.

It has been estimated that the entire output of this field is close on 1,300,000,000 cu. ft. per day, of which about 500,000,000 is used on or about the fields, as much more is sent out and the balance is wasted in very many ways.

One of the least reprehensible ways of wasting the gas is in the manufacture of carbon or lamp black. This is only to be excused when it is considered that otherwise the gas would be totally valueless at present owing to its distance from the main pipe lines, or if the well be shut down, gradually fill in and the flow would cease.

This carbon black is manufactured by burning the gas beneath iron funnels and introducing cold-iron plates into the flames on which the soot or black is deposited. This goes on day and night and the plates are scraped by various methods, the result being collected and sold.

In the glass industry, which uses an immense amount of gas, the molten glass is heated entirely by natural gas, making glass of a peculiarly good quality. It is also clean and efficient and cheap to install.

Another industry is that of zinc smelting, in which the clay retorts are heated by the same fuel. This industry very often fouls the air for a considerable distance around with arsenical and antimonial fumes and also with uncondensed zinc, and the manufacturers assert that no way so far has been found to recover these waste fumes, although where the Federal Government has put pressure on firms down in the South the corporations have been forced to adopt more scientific methods in this respect and have found that thereby they have increased their profits as well as decreased the nuisance.

Natural gas is also very useful in the manufacture of bricks and pottery, particularly so as strata of suitable clays are often found in the vicinity of the wells. By varying the gas flames in the kilns the product can be graded in color from the lightest Mackinaw shade to deep chocolate brown, possibly by varying the surface composition and combination of the silica with aluminium and iron.

The manufacture of electrodes is also very largely entered into. In this industry crushed coke mixed with oil-stills residues is pressed out and carefully placed in small kilns and the whole brought to a red heat for various periods by means of natural-gas blasts. This roasts the electrodes and gives them high tensile strength when used in furnace work at great heat.

Before closing it might be appropriate to mention something of the method of generation of these natural hydrocarbons.

In Bulletin 410 of the U. S. Geological Survey, George F. Becker reviews the evidence. In it Mr. Becker finds the combination of hydrocarbons and igneous action so often that all suggestion of fortuitous collocation seems out of court.

One widely accepted opinion is that of Mendeleef's theory of the decomposition of the carbides. The strongest reasons against this is that natural carbides have never been found even in the regions where hydrocarbons are most abundant. Magnetic declinations, however, due to iron deposits, are often found. Whether or not this is a proof of the carbide theory as some investigators have claimed is open to question.

The writer has proposed a theory which men on the field have pronounced at least probable, and it may be useful to tabulate the facts before we consider the theory. They are as follows:

That oil and gas are accompanied by strong evidences of igneous heat.

That they are accompanied by strata of coal.

That they are accompanied by layers of salt water, which salt water was really one of the original reasons for their discovery.

That they are also accompanied as low as 2000 ft. down by beds of water-worn gravel and shells.

That some of the supposedly exhausted gas fields have recently been redrilled and have yielded paying quantities of hydrocarbons.

That natural gas on pressure and cooling deposits liquid hydrocarbons.

That as previously mentioned they are affected by seismic disturbances.

From facts such as these the writer proposes the theory that all these hydrocarbons are generated from natural carbonaceous deposits, such as bituminous and other coal, by the natural heat of the earth's interior brought into proximity to such deposits by seismic disturbances, which action is still going on under enormous pressure and the hydrocarbons are being generated under the, among others, liquid pressure of the streams of water overlaying them, and on the release of some of the pressure and changes in the temperature of the hydrocarbons so generated, some of their contents are deposited in the dips of the strata and in the pockets known as oil-bearing pools.

The dips, fissures and tumbled nature of these strata would account for the eccentric behavior and irregular location, comparatively speaking, of these gas and oil aggregations.

In Kentucky there are wells and caves known as "breathing caves," from the fact that the well or cave is alternately giving out and taking in gases. It has been recently proved that the gases thus given out are explosive mixtures of hydrocarbons and air. The gases taken in are, of course, air. The cause of the breathing may well be brought about by the variable pressure of the water which seals most of these subterranean gases, and which is itself in a state of agitation, and shows that there is still violent internal action taking place among the strata containing this water, which is apparently advancing and receding to cause the so-called "breathing."

The different natural hydrocarbon centers may be roughly grouped as Russian, Californian, Texan, Pennsylvanian, which latter might also be called West Virginian. In all of these different bases appear in the hydrocarbons produced.

These different bases have been investigated and differentiated by Park and Worthing, among others, and the hydrocarbons constituting them have been investigated as to nature and structural formula.

In these three main sources of natural manufacture of hydrocarbons in the United States, namely, Texas, Pennsylvania and California, to locate still more accurately the main centers of generation, a suitable method is to find the group of wells giving hydrocarbons at the highest pressure and thermal efficiency. This method is the more likely to give correct results, as it is a well-known fact that the farther away from the source natural gas is piped above ground the lower is its thermal efficiency, and I hold it to be reasonable that the same fact is true below. Furthermore, as a general rule, the lower the pressure at which the hydrocarbons issue the lower is their thermal efficiency. In this manner a center of generation in each of these large fields is found to exist, whence the hydrocarbons have issued and are still issuing to be deflected and carried through considerable distances in the underground connected porous strata associated with all these hydrocarbons and known to drillers and operators as the various "sands."

These main centers will be the first to supply and the last to exhaust. Thus it seems the crust of the earth in some parts forms a cover with a liquid seal of salt water to approximately three vast natural gas-producer centers wherein the deposits of former ages have been and are being eternally distilled by the steady, well-modulated heat of the earth's interior for man's discovery and use.

Clarksburg, W. Va., and Melbourne, Australia.

Measurement of the Thermal Conductivity of Fireclay.

Bulletin 36 of the Engineering Experiment Station of the University of Illinois (Urbana, Ill.) contains an interesting paper on "the thermal conductivity of fireclay at high temperatures," the joint authors being Dr. J. K. Clement, of the Technologic Branch of the United States Geological Survey, and Mr. W. L. Egy, of the Engineering Experiment Station of the University of Illinois.

Although the investigation was undertaken to solve a specific problem—the thermal conductivity of fireclay at high temperatures—the method developed is of general interest in other lines of work where the question of thermal conductivity is of importance.

The thermal conductivity K of a substance at a certain point is given by the equation

$$K = -\frac{Q}{A}\frac{dr}{dT}$$

where Q is the heat (in calories per second) flowing in unit time across the area A at the point in question. The area A is assumed to be perpendicular to the direction of the flow of heat at this point. The differential quotient $-dT/dr$ is the temperature gradient at the point in question—that is, the differential quotient of the temperature taken in direction of the flow of heat (dr being an infinitely small length in direction of flow of heat, $-dT$ the corresponding temperature drop).

The most accurate method of measuring Q is to generate a known quantity of heat by means of an electric heating circuit, in such a manner that all the heat generated must flow through the substance to be tested. If a constant quantity of heat is generated until conditions have reached an equilibrium, then the quantity of heat conducted through the material per second must be equal to the quantity generated per second. This method may be used with a heating coil either in a hollow sphere or in a long cylinder. The latter form was chosen for these tests because of the experimental difficulties arising in the former.

In using this method, the assumption is made that there will be no escape of the heat longitudinally at the middle of the cylinder; that is, the exact amount of heat generated by a centimeter length of the coil, taken at the middle, must flow out through the corresponding circular section of just 1 cm thickness. To avoid errors from this cause, the length of the cylinder must be several times greater than its diameter.

The test pieces were made into cylinders about 40 cm in length and 12 cm in diameter, with a hole through the center about 3.5 cm in diameter for the reception of the heating coil. Four longitudinal holes, about 3 mm in diameter, were made, in which thermo-couples could be placed for the measurement of the temperature.

Applying the above equation to the case of a cylinder, the heat Q generated by 1 cm length of the coil in unit time is expressed by

$$Q = 0.2394\,EI/l \text{ calories},$$

where E represents volts, I amperes and l the length of the coil in centimeters. The constant 0.2394 is the reciprocal of the mechanical equivalent of a calorie expressed in joules. The area A perpendicular to the flow of heat at a distance r from the axis is $2\pi r$ per unit length. Substituting for Q and A in above equation, we get

$$K = -\frac{0.2394\,EI}{2\pi lr}\frac{dr}{dT}$$

Assuming that K is constant between the temperatures T_1 and T_2 and integrating, we get

$$K = \frac{0.2394\,EI\log(r_2/r_1)}{2\pi l\,(T_1 - T_2)}$$

where T_1 and T_2 are the temperatures at points distant r_1 and r_2, respectively, from the axis. For any given values of r_1 and r_2, the only variables in this equation are EI, the electrical watts dissipated in the coil, and $T_1 - T_2$ the difference in temperature between r_1 and r_2. Thus the expression may be reduced to

$$K = CEI/(T_1 - T_2),$$

where

$$C = \frac{0.2394\log(r_2/r_1)}{2\pi l}$$

Twelve test cylinders were investigated, which had been made by the Laclede-Christy Clay Products Company, of St. Louis, Mo. They were made up from four different mixtures, which were marked A, B, 1 and 3.

FIG. 1.—TEST PIECE SURROUNDED BY FIRE-CLAY JACKET.

Those marked A were of a dark reddish-brown color and contained no gravel. The structure appeared similar to sandstone. After heating in the test, these pieces were cracked in a great many places. The pieces marked B were also of a reddish-brown color, but were of medium coarse structure. They contained very small pieces of white gravel throughout the mass. Those marked 1 were a little coarser than the B cylinders and were brown in color. They contained a very small amount of gravel. Cylinders 3 were almost white and very coarse. They contained a large amount of gravel.

One cylinder of each composition was tested at temperatures ranging from 400° C. to 800° C., or 900° C. The remaining pieces were tested at only one temperature.

Fig. 1 is a general view of the apparatus with the test piece surrounded by the fireclay jacket, and with the thermo-couples in place.

Fig. 2 shows a longitudinal section of the furnace ready for use; *aa* is the test piece, surrounded by a large cylinder *bb* of fireclay, in order to get uniform radiation from *aa*, and also

to maintain higher temperatures. Coverings *cc* were placed over each end to prevent loss of heat in this direction. The whole was supported by an open framework of strap iron. A thermo-couple was placed in each of the holes *D* and *F*.

The heating coil was made of pure nickel wire about 1.8 mm in diameter, wound nine turns to the inch upon a ½-in. porcelain electric insulator tube. Commercial insulator tubes 20 in. long were taken for this purpose and the enlarged end cut off, leaving them 18 in. or 19 in. long. Small holes were chipped in the ends of each tube, and the ends of the wire pulled tightly through these holes, thus holding the wire in place. The holes were 16⅛ in. apart, allowing 145 turns of wire, or an equivalent length of 40.9 cm. Potential leads were fastened to the power wire where it emerged from the insulator, by winding one end of a piece of smaller wire tightly about it. The place where this connection was made had a rather high temperature, so nickel was used to prevent any thermo-electric effect at the junction.

The coil was at first supported in the cylinder by two small pieces of quartz tubing at each end (see Fig. 3). At the higher temperatures, however, the insulator softened and sagged down in the middle, and two additional pieces of quartz were placed under it, each one-third the distance from either end. The holes in the end of the furnace were packed with asbestos to prevent the escape of heat. The heating coil was made a few turns longer than the test piece to prevent cooling of the ends.

Current and voltage were read by Weston instruments. The current was taken from 110-volt mains and was regulated by means of two large rheostats. As it was necessary to maintain the current constant to one-tenth of an ampere for several hours at a time, and the line voltage varied considerably, a storage battery was placed across the mains.

The temperatures were measured by platinum-platinum rhodium thermo-couples, made by Heraeus. The e.m.f. of the couples was measured by a Wolff potentiometer and a galvanometer.

The heat which was generated at the middle of the coil flowed out along radial lines. This was tested by moving the outer couple *F* (Fig. 2) back and forth in a longitudinal direc-

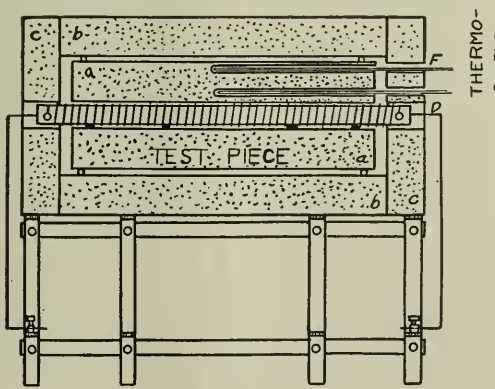


FIG. 2.—SECTION OF FURNACE.

tion and noting the reading in the various positions. This operation showed that the temperature was constant to 0.3° C. for a distance of 6 cm to 7 cm from the middle. As no heat will flow along constant temperature lines, there could be no longitudinal flow of heat in the 12 cm or 14 cm in the middle of the test piece, and hence the assumption is allowable.

The general method of operation was to start the furnace with a rather large current, about 25 amp, and gradually reduce this as the furnace approached the temperature desired. This took from three to five hours for the lower temperatures. The storage battery would then be placed in the circuit and the current kept constant for two or three hours more. When the outer couple *F* showed a temperature constant to one-tenth of a degree, indicating equilibrium conditions, readings were taken. The voltage and current readings were taken before and after the temperature readings in order to be sure there was no change.

Each cylinder at the close of the tests on it was broken across the middle and r_1 and r_2 carefully measured in the plane in which the temperature readings had been taken.

The accuracy of the values for *K* in this work is limited to the accuracy in measuring r_1 and r_2. The temperature readings are certainly accurate to 1.0° C., and very probably to 0.5° C. The voltage and current readings are accurate to 1 per cent. An error may be introduced in the values of r_1 and r_2 because of the uncertainty of the position of the thermo-junction in the hole.

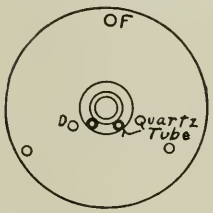


FIG. 3.—DETAILS OF QUARTZ TUBING.

If the couple were touching one side of the hole, it would take the temperature of that side, while r_1 and r_2 were always measured to the center of the holes. This error, however, would be a constant factor for the tests on any cylinder, since the couples were never disturbed after they were once placed in a cylinder until the completion of the tests on that piece. Thus the comparative values of *K* for any given specimen at different temperatures would not be affected by the values of r_1 and r_2.

The greatest error is caused by changes of current through the furnace. The supply voltage changed a great deal, and the batteries used were not large enough to maintain the current constant at all times. If the current should change just before a reading was taken, the temperatures would not correspond to the other readings, though they might not be changing at the time.

As to the results it is interesting to note that the conductivity of the two coarser fireclays was constant, while that of the other two, which were of finer structure, increased with the temperature.

For a test piece B the conductivity *K* (in gram calories per centimeter per second per degree C.) increases from 0.0021 at 350° C. to 0.0022 at 600° C.

For a test piece 3 the conductivity *K* was practically constant = 0.00264 between 400° C. and 700° C.

For a test piece A the conductivity *K* increased from 0.00245 at 500° C. to 0.0026 at 800° C.

For a test piece 1 the conductivity *K* was practically constant = 0.0036 between 400° C. and 750° C.

There were found, however, differences between the conductivities of cylinders of the same composition. These are probably due to differences in porosity. An investigation of the effect of porosity as well as composition of substances would, no doubt, lead to more definite results in regard to a comparison of fireclays.

The application of the above figures in practice is illustrated by the following example:

Knowing the thermal conductivity *K* of brick, the dimensions of the furnace, and the outside temperatures of the walls, the quantity of the heat transmitted through them may readily be calculated. Taking the specific case of a 210-hp Heine boiler at the University of Illinois, when working under full load—the area of walls exposed to the hot gases is about 364 sq. ft., or 338,000 sq. cm., and the thickness of the same about 20 in., or 51 cm. The average temperature of the inside of this area is approximately 1400° Fahr., or 760° C., and of the outside 150° Fahr., or 65.6° C., and $-dT/dr = 694.4/51$. If we now take the value of *K* as found for that test piece which was nearest like the brick in the setting—that is, $K = 0.0026$—and calculate the

heat conducted through the walls we have, using c.g.s. units:

$$Q = -KA\frac{dT}{dr} = 0.0026 \times 338{,}000 \times \frac{694.4}{51}$$

= 12,000 gram calories per second = 43,400,000 calories per hour.

In an appendix the method for calibrating thermo-couples for high-temperature measurements is described in great detail.

Continuous Pachuca Tank Agitation at the Esperanza Mill.*

By M. H. Kuryla.

The agitation equipment of the new cyanide plant of the Esperanza Mining Company, El Oro, Mexico, consists of two batteries of six 14 ft. 10 in. x 44 ft. 8½ in. Pachuca tanks. The first battery of six tanks, locally referred to as the "Upper Pachucas," treats the flocculent slime produced from the coarse crushing in 120 stamps, and is worked on the "intermittent" or "cycle" system originally installed. The location permits of a gravity discharge to the storage tanks for the Merrill filter presses.

The six "Lower Pachucas" treat the granular slime resulting from the regrinding of the battery sands in 10 4-ft. x 20-ft. tube mills. It is with these "Lower Pachucas" this article deals.

Original Intermittent System.

Each tank held 80 tons dry slime and 144 tons of slime per 24 hours; the time required to fill one tank was 6.4 hours. After the available length of agitation each tank was discharged through a 5-in. valve at the bottom and pulp raised to the filter press storage tanks by means of three vertical bucket elevators. These elevators were made of an 18-in. 10-ply rubber elevator belt 54 in. C to C of head and boot pulleys, with 16-in. x 7-in. x 5¼-in. type AA malleable iron buckets, spaced 18 in. C to C. The boots were made of cast iron and provided with accessible handholes for cleaning.

The housing, 2 ft. 6 in. x 5 ft. 6 in. inside dimensions, was built of No. 12 steel, with removable doors for repairs on belts, buckets, and pulleys. The original belt speed of 360 ft. per minute was raised to 540 ft. per minute to permit one elevator to be held in reserve. With the average condition of the elevators the time required to discharge one tank was about 3.8 hours.

On the assumption that one-half of the filling and discharging times resulted in net agitation on the entire tank charge, the net time lost in these operations amounted to 5.1 hours. The net agitation per charge, accordingly, worked out as follows:

Total possible agitation without filling or discharging loss ..... 38.4 hours
Loss in filling and discharging ..... 5.1 hours

Net agitation ..... 33.3 hours
Percentage of net agitation lost in filling and discharging, 15.3 per cent.

The yearly expenses connected with the operation of the three elevators figured out as follows:

	Mexican Currency.
Three elevator belts, each 12P', at $6	$2,160
Power	1,825
Regular attendance	730
Buckets and bolts	35
Repairs, labor	1,005
Repairs, material (exclusive of belts)	500
Total per year	$6,345

Present Continuous System.

The difficulties and expense experienced with the elevators made it highly desirable to deliver the pulp by gravity to the filter press storage tanks (collars of the storage tanks being 6 ft. below the collars of the Pachuca tanks and 50 ft. distant). The 6-in. pipe connections shown in Fig. 1 were put in Feb. 10, 1910, first as an experiment to demonstrate the applicability of the scheme. The particular information desired was with reference to comparative extraction and classification during agitation, as it was thought that the "sandy" product fed to the tanks might undergo considerable classification in passing from tank to tank. The temporary connections have proven so satisfactory that they have been retained in present daily operation.

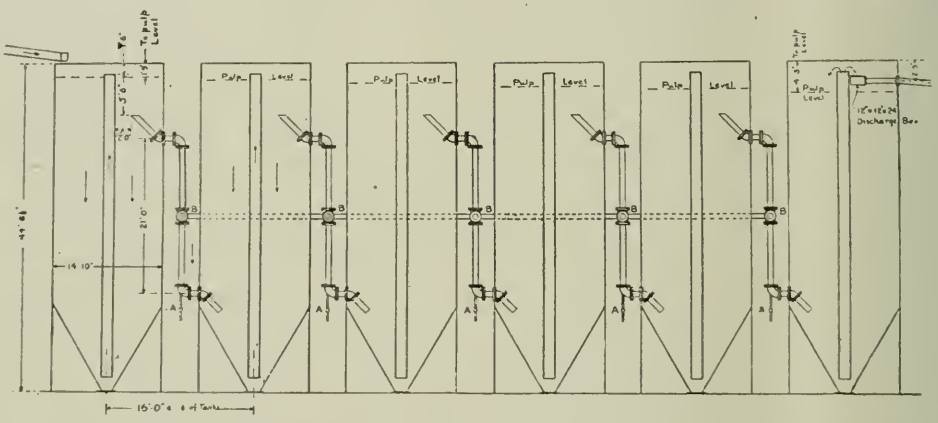


FIG. 1.—THE CONTINUOUS SYSTEM OF AGITATION AT THE ESPERANZA MILL.

The pulp is fed into the top of tank No. 1 at the rate of 300 tons dry slime and 540 tons solution per 24 hours and flows through the 6-in. pipes by gravity to the discharge box near the top of tank No. 6. The friction drop from tank to tank is about 6 in., making a total drop in the level of the pulp between tank No. 1 and tank No. 6 of 30 in. However, by utilizing the central tube as an air lift, and so placing the outlet into the final discharge pipe or launder that the overflow of the air lift will drop into the outlet, the difference in the level of the pulp in tank No. 1 and the level of the final discharge is reduced to 12 in. In fact, were it desirable to have the final discharge at a higher level than the first tank, the air lift in the last tank of the series could be raised sufficiently to permit of this being done (at an additional consumption of air, of course).

The intake into a tank being near the bottom, the pulp rises

* Paper presented on April 28, 1910, before the Instituto Mexicano de Ingenieros de Metalurgia.

through the air lift, as the current in the annular space around the air lift is in a downward direction. The discharge from the tank is placed 7 ft. below the collar of the tank and 2 ft. from the side of the air lift, in this way "sampling" the overflow of the air lift and producing a practically uniform transfer of pulp from tank to tank, as shown by sizing tests and solid liquid ratios in the following table:

SIZING TEST OF PULP.

		Discharge from Tank No.					
	Charge.	I.	II.	III.	IV.	V.	VI.
+ 200 mesh.....	14.7%	14.9%	15.2%	15.7%	15.3%	15.1%	14.9%
— 200 mesh.....	85.3	85.1	84.8	84.3	84.7	84.9	85.1
Strength of K Cy solution in percents—		.12	.112.	.110	.1075	.105	.1025 .100

Consumption of K Cy per ton of ore, 358 grams.

Gold Contents and Percentage of Extraction.

+ 200 Mesh.....9.7 gr.	46%	58%	61.5%	62%	63%	63.5%	
— 200 Mesh.....6.0 gr	68	75.5	78	80	82	83.5	
Gen. Sample.....6.7 gr	66	72.5	74.5	76	77	78	

Silver Contents and Percentage of Extraction.

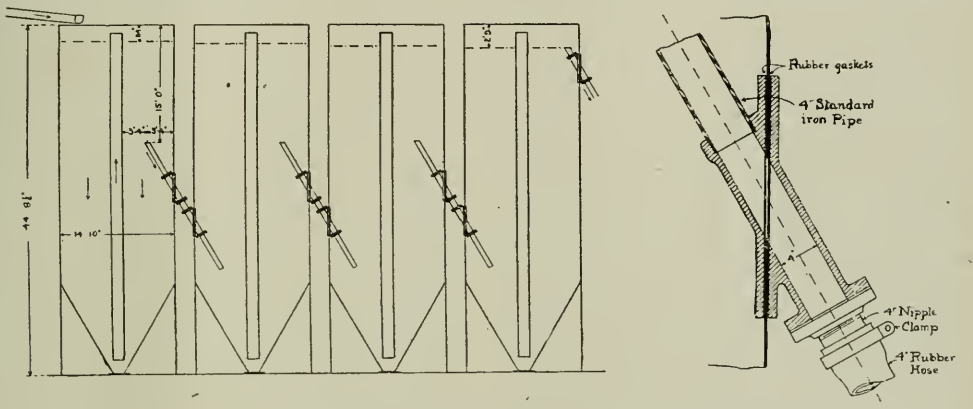
+ 200 Mesh.....75 gr.	9%	16%	22%	26%	28.5%	31%	
— 200 Mesh.....83 gr.	28	39	46	50.5	53	54	
Gen. Sample.....80 gr.	22.5	34.5	41.5	46	48	50	

Feed and final discharge, solid to liquid ratio 1 :1.80.

Sampling, six days at one-hour intervals.

Mr. Grothe thought that the mechanical part of the connections between the tanks, as shown in Fig. 1, could be improved upon, but the end discharge of the last tank was not open to objection. While the degree of sizing as reported by Mr. Kuryla is very satisfactory, the numerous curves in the pipes might be avoided. Especially the elbow at the discharge end of the connecting pipe is objectionable as it may become partially filled up by sediment, thus reducing the useful section of the pipe. The hydraulic grade between the successive tanks shows this to be the case. With a much smaller head the discharge through a 4-in. pipe should answer the purpose, if no obstruction existed. In the arrangement shown in Fig. 2 the possibility of clogging is entirely avoided. The pipes have an incline of 60 deg., and a moderate velocity of pulp keeps them clean. The connection outside the tanks consists of a flexible pipe, nullifying the effect of vibration and unequal expansion.

Mr. Grothe concludes that when the subject is better understood it will be found that continuous agitation is a great improvement on former practice. In fact, he can only see one



Section Details of Constructions.

FIG. 2.—THE CONTINUOUS SYSTEM OF AGITATION AS SUPPLIED BY GROTHE AND CARTER.

The gain of extraction in the continuous system as compared to the intermittent is 1.3 per cent of gold and 1.5 per cent of silver and the saving in cyanide 25 grams per ton of ore.

The saving in operating labor and the increase in length of agitation effected in the "Lower Pachucas" has made it advisable to install the continuous system in the "Upper Pachucas" even though, as already mentioned, the "Upper Pachucas" discharge by gravity to the filter press storage tanks.

* * *

In the discussion which followed Mr. A. GROTHE referred to some experiments made by him and Mr. J. Leslie Mennell as to the effect of continuous agitation in laboratory tanks of 18-in. diameter and 54-in. height, the results being published in the *Mexican Mining Journal*, of February, 1909. He pointed out that Mr. Kuryla's paper is of particular value, as it removes the matter from the experimental to the practical stage. From his observations it is clear that an important saving in time, coupled with a decrease in cost and a slightly higher extraction, is obtained. Theoretically, the extraction in the continuous system should always fall a little short of the possible extraction in the intermittent and the reason that in Mr. Kuryla's case, as well as in Mr. Mennell's experiments, there was an increased extraction must be due to the fact that formerly the ore (in the intermittent system) was, for want of tankroom, not agitated as long as when the six tanks were connected and advantage was obtained in the latter case from the time formerly lost by filling and emptying, which now became available for some additional agitation. It seems certain from his observations that with the same number of tanks there is a decided advantage in favor of the continuous system.

disadvantage in the method (if it is one), viz., that a successful sampling of the pulp, on which the efficiency depends, can only be effected in "Pachuca" tanks.

Notes on Chemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

The Iron and Steel Institute.

The spring meeting of the Iron and Steel Institute was held at the Institution of Civil Engineers on May 4 and 5.

The Duke of Devonshire's presidential address was mainly a remarkable comparison of international statistics of coal, iron and steel productions, and of populations during the last 40 years or so. (An abstract was given on page 314 of the June issue of this journal.)

First Day's Papers.

The papers by Mr. D. SELBY-BIGGE on **"Development in the Production of Electric Power,"** its Application and Bearing upon the Iron and Steel Industries, and by Mr. E. G. SEHMER and Dr. R. DRAWE on economy and design of modern **Reversing Rolling Mill Steam Engines** were discussed together. Abstracts of these two papers will be found in the Synopsis in this issue.

In opening the discussion on Mr. Selby-Bigge's paper Mr. T. Westgarth said the utilization of waste heat was a matter of great importance to them as iron and steel makers. The existence of waste heat was self-evident, and the great point was to ascertain the cheapest way of using it. At Barrow a saving of

1500 tons of coal per week had been effected by putting in gas engines for blowing the furnaces and producing electricity for the auxiliary machinery; and the surplus gas was taken to the boilers to raise steam for the steel-works engines. At South Bank the waste steam was taken to regenerators and used in low pressure turbines. Another firm sold their exhaust steam to a power company, who took it from the regenerators to low-pressure turbines. There were many ways of solving the problem of using waste energy, but it was not possible to lay down any general rule. Each case must be governed by its own conditions. Nevertheless there was a good deal to be said for the use of gas engines, because the surplus power from the coal after it had been used at the coke ovens and furnaces was probably more than could be obtained from the best engines and low-pressure turbines.

Mr. A Lamberton said that it was not possible for him to attempt to discuss Mr. Selby-Bigge's paper in the short time allowed, but he fully recognized its great value. The paper by Mr. Schmer and Dr. Drawe was also interesting, but it had some defects, and he would like to put some questions with a view of removing them. Some of the authors' figures seemed to institute comparisons between modern engines in Germany and a wasteful non-condensing engine referred to in a paper by Mr. Ablett last year. The authors stated that the Schmer engine showed an economy of 35 per cent to 50 per cent in steam, but their figures showed that their engine used 350 lb. of steam as against 1100 lb. That was not 50 per cent. He made it 68 per cent. Such a discrepancy tended to throw doubt upon the whole question. He controverted the authors' statement that the introduction of low-pressure turbines between compound engines and the condenser was not a practical scheme, and quoted an instance showing an economy of 17 per cent in favor of the combination as against the reciprocating engine.

Mr. C. H. Merz agreed with Mr. Westgarth that the problem at every individual works must be solved in accordance with its own circumstances. Instead of employing exhaust-steam turbines or mixed-pressure turbines they might use gas engines, or put in an entirely new plant of high-pressure turbines. He had installed turbines at various stations totalling about 200,000 hp; he knew how most of them were operating, and he thought that in many cases exhaust turbines and mixed-pressure turbines were recommended where conditions were not favorable. He thought that in many cases exhaust and mixed-pressure turbines were put in in consequence of the reluctance of the manufacturer to scrap his old plant, which the exhaust turbines allowed him to retain, when a new turbine plant or installation of gas engines would have been better economy. The great thing in the use of exhaust and mixed-pressure turbines was the vacuum, without which they were practically useless, and the requisite vacuum was not always attained. He could not agree with Mr. Selby-Bigge's figures as to interest and depreciation, which would only allow them to write off their plant in 20 years. Turbines could be bought to-day giving 50 per cent better results than they did a few years ago. He thought that manufacturers should not jump to the conclusion that exhaust or mixed-pressure turbines gave additional power for nothing or that they were the best things, because in many cases they were not, and he would advise them to wait longer and completely reorganize their plant.

Mr W. C. Mountain thought that, having regard to his experience of three collieries for several years, Mr. Selby-Bigge's figures as to cost per unit might be taken as correct. He had previously expressed his doubt as to electric winding being made commercially successful, and those remarks applied equally to electric rolling at present. The commercial aspect of these matters should be more fully considered. The lowest estimate a steam-produced horse-power could be put at was from $17.50 to $20 per annum.

Mr W. J. Larke remarked that one question which had not been gone into was the use of the waste gas from blast furnaces and coke ovens to raise steam for high-pressure turbines. He knew that, in respect to thermal efficiency, this was not the best way of using the available gas, but the main point was commercial efficiency.

Second Day.

The discussion was resumed by Mr. Williams who said they had used gas from blast furnaces to raise steam for the steel works, and part of the exhaust steam was then used for generating their electric power and for blowing. The results appeared quite satisfactory, and they were going to join up another plant. They had found no difficulty in maintaining a vacuum of 28 in. to 28½ in., and, of course, below 28 in. the loss would be very great.

Mr. J. Mulholland advocated the use of high-power gas engines in iron and steel works. Their prime cost was no greater than that of steam plant, while repairs and maintenance cost less; and he thought that the superiority of the gas engine in thermal efficiency was generally acknowledged by engineers.

Mr. Selby-Bigge briefly replied. He agreed that in certain cases gas engines deserved to be considered as well as other means of producing power. Gas engines had recently been very much improved and were now very different from those produced even three or four years ago. He corroborated Mr. Lamberton as to the erroneous nature of the statement at the end of Mr. Schmer's paper concerning the interpolation of a mixed-pressure turbine between the engine and the condenser. With regard to the question of difficulty in maintaining a vacuum he could support Mr. Williams' remarks. Personally, he has found that with mixed-pressure turbine plant 28½ in. was ordinarily obtained. They had got 29 in., but 28 in. was the most usual. Of course, it was necessary that the condenser should be big enough. The figures he had given for capital outlay were taken from an actual case.

Mr. Schmer, in the course of his reply, said they did not wish to give a definite decision on reversing rolling mills, whether driven by steam or electric power, but only to mention some new features, the publication of which they considered desirable. The figures given by one of their newest engines showed that they effected a saving of from 60 per cent to 65 per cent.

Electric Furnaces.

Dr. W. Borchers' paper on "The Girod Furnace" (abstracted in the Synopsis in this issue) was then read. The discussion was opened by Mr. J. H. Heap, who said it was a reproach to us in this country that we had left the experimental work on the electric furnace chiefly to Frenchmen, although conditions afforded great opportunities to steel manufacturers for experiments with it. In the Middlesborough district they had waste gases affording as cheap a source of power as could be had anywhere. Even with the best hydraulic schemes the cost was as high, or higher, than that of using waste gases in an economical way. He was sorry Dr. Borchers had not given a general resumé of the electric furnaces at present in use, which, he thought, would have great interest for steel manufacturers who would be glad to have component and independent statements on the merits of the various types. He could not quite follow the author's reasoning in his approval of the Girod furnace. The only point which he could see indicating a special feature was in the contact pieces, and he doubted if the arrangement for cooling would enable them to last very long. The author seemed to assume that circulation of the contents of the bath would occur, but in some special arrangements that had not been the case. It seemed strange that it should have been left to Sheffield, which was by no means an ideal place, to take up the electric furnace. To obtain ideal conditions for the electric furnace they must have a cheap power and cheap material, and Sheffield producers had been obliged to have expensive raw material.

Mr. E. H. Saniter wished to ask Dr. Borchers if the Girod furnace consumed 1 kw-hour less current to produce a ton of steel than the Héroult furnace?. Dr. Borchers seemed to con-

sider that the Girod furnace was better because there was only one roof-hole and one electrode, and because—so the author stated—the resistance of the bath increased the heat there. Now the resistance of molten metal was the source of only a small proportion of the heat generated. In the Héroult furnace the reaction area was three times as great as in the Girod furnace, and he wished to ask the author if the smaller reaction area were an advantage.

Prof. J. O. Arnold said he supposed Mr. Heap would credit Sheffield with understanding its own business. Mr. Heap was repeating an old fable as to converting Middlesborough iron into the highest class of steel. That had been a menace to Sheffield for at least a century, but it had never been accomplished. What they had to consider was the relative melting capacities of the crucible and the electric furnace. They wanted a melting furnace, not a refining furnace, and therefore adopted the Kjellin furnace, and still used the finest Swedish iron, which produced the highest quality of steel known in Sheffield.

Mr. F. W. Harbord observed that he had had opportunities of seeing all the various types of electric furnaces except the Kjellin, and from an inspection of the Girod furnace last year he judged that it worked very satisfactorily, but he did not think Dr. Borchers should state that it was the simplest form of furnace. That depended on the particular purpose for which the furnace was designed. The advantage of one roof electrode instead of two did not apply to large furnaces, in which they must have two electrodes, and he believed in some cases they had three or four. With furnaces holding 200 tons of metal sufficient circulation was obtained without special means, and in small furnaces the ordinary reactions produced adequate circulation. He should expect trouble with the bottom electrodes because metal and slag were liable to collect round them when tilting, and the slag had to be splashed out. Repairs would also be troublesome.

Mr. D. F. Campbell said the figures in the paper did not warrant the statement that the Girod furnace was the simplest. Comparing a 12½-ton Girod furnace with a 15-ton furnace working at South Chicago, the Girod type had four roof holes and 16 bottom electrodes—and he thought trouble was more likely to arise with the bottom holes than with those in the roof—while the Héroult furnace had three roof holes and the bottom was arranged exactly as in the best basic open-hearth practice. The only sources of heat were the electricity and the mechanical reaction. He had no doubt there was ample circulation in any electric furnace, and had ascertained it in the Héroult type by adding ferro-chrome. In the Girod furnace the steel was liable to get under the layer of dolomite and hasten the breaking up of the lining. In South Chicago the bottoms of the Héroult furnaces lasted from 4000 to 5000 heats, and the cost of roof was about 4 cents per ton. When melting cold scrap the bottoms were protected by several hundredweight of lime and mill-scale, but in the Girod furnace the scrap was in contact with the pole pieces. The paper gave the time necessary for a heat with the Girod furnace as eight hours, whereas with cold scrap in a 2½-ton Héroult furnace the time of a heat was only slightly over six hours.

Carbides in Manganese Steel.

The next paper, by Dr. J. O. Arnold and Mr. A. A. Read, was on "The Chemical and Mechanical Relations of Iron Manganese and Carbon." (See abstract in Synopsis in this issue.)

Mr. J. E. Stead opened the discussion, and said he had always been of opinion that carbides of some kind—he would not say definite carbides—did diffuse in steel when cooled slowly, but the existence of a certain thing after cooling was not proof of the condition during solid solution. Dr. Campbell had, some years ago, suggested there might be different carbides in steel, just as there were different compounds of hydrogen and carbon, and this view seemed plausible. He (the speaker) had done some work in the same direction as Dr. Arnold's researches, and their conclusions were practically identical. He would like Dr. Arnold to explain the different characters of pearlite represented in one of his photographs. Why was one granular, another lamellar, and a third described as troostitic?

Mr. W. H. Hatfield said the existence of a definite mixed carbide seemed to be established. According to the analysis a lower yield of carbide was accompanied by a higher percentage of carbon in the residue. He should like to have the result of similar experiments on supersaturated steel from which the carbide would come out without difficulty in larger masses.

Dr. Arnold replied that he could not explain the different phases of the pearlite. It was quite easy with the very fine lenses now produced for micrography to resolve so-called sorbite into laminæ. Besides the alloy was heated in a bar to a bright red for 36 hours, and then given three days to cool.

Tool Steel.

In the discussion on Mr. E. G. Herbert's paper on "The Cutting Properties of Tool Steel" (abstracted in the Synopsis in this issue), Mr. W. F. Beardshaw remarked that he hoped it would not be accepted that the author's method was the best possible one for ascertaining the relative values of tool steels, but judgment should be suspended until steel makers had had opportunities of testing the ingenious machine described.

Mr. King (Great Western Works) was of opinion that the conditions of test with the author's machine had little in common with those observed in machine shops in ordinary working. It seemed to him that the heat generated by the tool in the author's test was so small as not to compare even with that of a finishing cut in a tool shop.

Mr. A. Lamberton could not agree with Mr. King because it was shown that in some of the tests the tool was heated by gas jets to obtain workshop cutting conditions. The paper was of great value. The high prices asked for high-speed steel justified their asking the producers to specify the conditions under which it would give the best results.

Mr. Herbert, in his reply, said his objection to the usual working test was that it was carried out at one speed, and he had shown that variation of speed affected the result irregularly. They must test at all speeds to ascertain which was the best. In some cases a higher speed had increased the durability, just as his curve showed. Mr. King was under a misapprehension. The tests were made from low temperatures up to the highest the tool would stand.

Chromium Steel.

The paper on "Some Physical Aspects of 2 Per Cent Chromium Steels," by Andrew McWilliam and E. J. Barnes, was discussed with H. Moore's paper on "The A2 Point in Chromium Steel." Abstracts of these papers are given in the Synopsis in this issue.

Dr. W. Rosenhain questioned the correctness of Mr. Moore's differential heating curve. Indication of thermal change frequently occurred at a considerable interval from the temperature at which the bulk of the transformation took place. He considered that it had been shown that hardness was not maintained by quenching. It was really a case of the preservation of β iron by quenching, and the results Mr. Moore found in that case might naturally be expected.

Mr. E. H. Saniter asked Professor McWilliam if there were any specific reason for using so low a proportion of manganese in the steel. It seemed extraordinary that a temperature of 960° should be used for normalizing and annealing, while for quenching in water they had only 850°. If steel were heated to 1000°, allowed to cool, and then heated to AC1 and not to AC3, would there be any reorganization in its structure.

Professor McWilliam replied. He was surprised to hear Dr. Rosenhain express his belief in the existence of hard β iron. It was a question on which there was violent disagreement. How did Dr. Rosenhain define hard β iron?

Dr. Rosenhain, interposing, disclaimed having defined it. The reason for the hardness of hardened steel was the only practical point to be dealt with in connection with the hardness

of β iron. If it was only slightly harder than ordinary iron then it made no appreciable difference. Everyone knew that was the point at issue. In answer to Mr. Saniter, there were both practical and scientific reasons for keeping the manganese low, because they wanted to see the effect of chromium. Mr. Moore also briefly replied.

Iron and Steel Production in the Transvaal.

The Transvaal Mines Department has issued a report by Mr. F. W. Harbord on the prospect of manufacturing iron and steel in the Transvaal. Although there are good deposits of magnetic quartzites near Pretoria and at Airlie, which, if mixed with small quantities of richer non-siliceous ores existing in other localities, could be smelted to produce a pig iron suitable for steel making by the basic process, yet the supply of suitable limestone and coke, except at excessive cost, makes it improbable that the manufacture of iron and steel in the Transvaal is likely to be a commercial success. Labor does not present any insuperable difficulty, but the question of a market for the finished product deserves serious consideration.

A steel company in the Transvaal could hardly hope to secure more than 50 per cent of total imports to South Africa, which would mean about 50,000 tons per annum of all classes of materials.

If all the railways of South Africa would combine to take a fixed fairly large quantity of material per annum, the situation would be more promising, and an iron and steel works turning out 1000 tons of finished steel per week should then be able to live.

Mr. Harbord comes to the conclusion that any attempt to start the manufacture of pig iron and steel on a large scale would at present be premature; but, on the other hand, electric steel plant to deal with the large quantitiees of scrap at Pretoria and other places in the colony promises good results, both technically and commercially. After careful consideration of all local conditions, both as to quality of scrap and finished products required, he is of opinion that an electric steel furnace is by far the most suitable and will give the best results in actual practice.

Market Prices.

May, 1910.—*Copper* has remained fairly steady throughout the month; prices 4th, 11th, 18th and 26th, £56. 2. 6., £57, £56. 3. 9 £56. 8. 9., closing at £56. 13. 9.

Tin has been within £2 of £150, with upward tendency at commencement of the month, but steadying after the 20th. Closing at £150.

English Lead has remained flat, closing at £12. 18. 0.

Iron, Hematite, after a fall with the general depression, followed by a spurt, settled down to 66/3.

Scotch Pig has been lower and rather unsteady, within small limits. Closed at 55/10.

Cleveland Warrants, much the same as Scotch pig. Closed at 49/10½.

	£	s.	d.
Alum, lump, loose, per ton	5	12	6
Antimony, black sulphide powder, per ton	20	0	0
Borax, best refined crystal, per ton	16	0	0
Sulphate of ammonia, f.o.b. Liverpool, per ton	11	15	0
Sal-ammoniac, lump, firsts, delivered U. K.	42	0	0
Copper sulphate, per ton	17	10	0
Camphor, 1-oz. tablets		1	7
Caustic soda, 48 per cent ordinary	5	10	0
Bleaching powder, 35 per cent, per ton	4	5	0
Shellac, standard T. N. Orange Spot, per cwt.	3	10	0
Sulphur, recovered, per ton	5	0	0
Carbolic acid, liquid 98/99 per cent, per gal.			11½
Hydrochloric acid, per cwt.		5	0
Creosote, ordinary good liquid, per gal.			2¼
Naphtha solvent, 90 per cent, at 160° C.		1	4½
Platinum, per ounce, nominal	5	15	0

Rubber still maintains a high price, though inclined to settle down to a more fixed value. Taking the 6th, 13th, 20th, and 27th the prices were 10/7, 10/4, 10/6½ and 9/10, closing at 10/1.

It will be noticed that sulphate of ammonia and copper sulphate have fallen considerably; carbolic acid has dropped ½d., and platinum is quoted lower, but "subject to arrangement."

London, May, 1910.

Synopsis of Metallurgical and Chemical Literature

Iron and Steel.

Electric Power for Iron and Steel Works.—A paper presented by Mr. D. Selby Bigge at the recent meeting of the Iron and Steel Institute deals with "developments in the production of electric power; its application and bearing upon the iron and steel industries." He deals especially with the progress that has been made in the cost at which electricity can now be produced in iron and steel works having at their disposal waste gas, waste heat, and waste steam. One of the means by which a lower cost of production has been attained has been brought about by the advent of the mixed-pressure turbine. The author discusses at some length mixed pressure turbines and then refers to the fact that the introduction of the steam turbine has created a demand for cooling plant in cases where a natural supply of cooling water is not available, and where a good case exists for the introduction of mixed pressure or exhaust-steam turbine plant. There are several types of cooling plant on the market giving excellent results. The three forms most commonly met with are the natural draught cooling tower, the open-type cooling tower, and spray nozzles distributed over a pond. In England the circulating water can be reduced to a temperature of 80° Fahr to 85° Fahr under ordinary conditions with a natural-draught cooling tower. The open-type cooling tower will give better results, and circulating water temperatures of 75° Fahr may be obtained with its use if situated in a favorable position. This type of tower, however, requires more ground space than the natural draught type. Where ground space is not of great importance, and especially in open situations, spray nozzles are to be recommended, as excellent results are obtained by their use, and if properly installed, it is quite easy to maintain the temperature of the circulating water at the prevailing atmospheric temperature. Details are given about the following types of mixed-pressure turbines: Parsons, Ratean, Curtis, Bergmann, Dr. Laval, Zoelly, Westinghouse. Recent development in gas engines are then taken up (Mather & Platt, Nurnberg, Cockerill-Westgarth).

In discussing the commercial aspects of electric power schemes with the aid of waste heat, the author gives the following details of the electrification scheme, which has recently been carried out by the Fife Coal Company in connection with their collieries in Scotland. In this scheme the principle adopted has been to lay down mixed-pressure turbines at those collieries where the greatest amount of exhaust steam might be available, and feed the power thus generated into a common network of mains, from which power could be drawn off to supply the requirements of their other collieries in the district. The Fife Coal Company possess five mixed-pressure turbines, each of 1000 hp and one of 800 hp. The current is generated at 3000 volts at a frequency of 50 cycles. The turbines run at a speed of 3000 r.p.m., and in addition to the direct-coupled exciters upon the shaft, separately driven steam excitation sets are provided for emergencies. Each plant is complete with condensing and water-cooling arrangements. It is yet too early to gauge the total savings resulting from this comprehensive electrifica-

tion scheme, but at one of the collieries, where a 750-kw plant has now been in operation for a year, the cost of production, after allowing 12 per cent interest and depreciation upon the capital outlay, including a complete standby unit, and including superintendent's wages, oil, stores, waste, etc., and an allowance for live steam, amounts to 0.4218 cent per kw-hour, made up as follows:

Interest and depreciation at 12 per cent per year upon cost of complete station equipment, including standby unit ($47,000)	$5,640
Station attendants	800
Stores, oil, etc.	225
Coal for live steam and firemen's wages	3,420
	$10,085

Total load ..... 2,295,000 kw-hours.
Cost of production ..... 0.4218 cent per kw-hour.

At this particular colliery, the net saving which has been actually effected already exceeds $15,000 per year, but as the plant is not yet fully loaded up and more motors have to be added, the ultimate saving will be considerably increased. Brief notes are added on electric rolling mills, electric furnaces, transmission lines and voltages of installation. (From advance sheets.)

Rolling-Mill Steam Engines.—An Iron and Steel Institute paper by Mr. EDUARD G. SEHMER and Dr. R. DRAWE deals with the "economy and design of modern reversing rolling-mill steam engines." They believe that the electric drive of rolling mills, electricity being generated from gas power, is a mistake. They describe a simple single-lever gear for steam engines for rolling mills so designed that the steam consumption is a minimum. The output of the work of the engine is controlled by regulation of the cut-off, the proper amount of cushioning takes place, and the number of idle revolutions is very small. Reference is made to a 45-hour test on a rolling mill compound

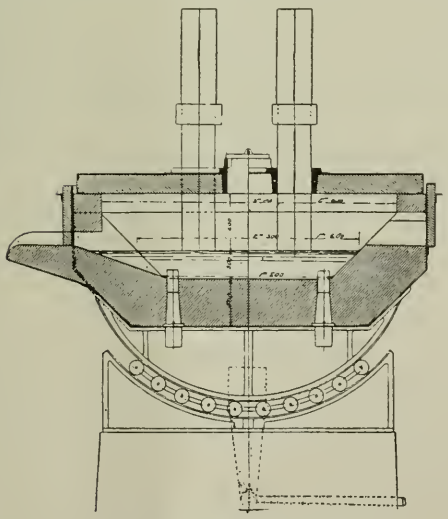


FIG. 1.—VERTICAL SECTION OF GIROD FURNACE.

engine recently carried out by the Commission on the Power Requirements of Rolling Mills. The mean pressure on the engine was 103 lb.; the mean temperature vacuum was 23.5 in. The average steam consumption was 350.77 lb. per ton of material rolled down to 9.222 times its original length. Under equal conditions, and with the same amount of elongation, the steam consumption was formerly 880 lb. to 1100 lb. per ton. The amount of steam consumed includes the whole of the feed water necessary for the engine, together with all the auxiliary engine and steam pumps for the condenser.

The authors believe that the advantages of steam turbines are often overrated and refer to experiments carried out at the "Union" works at Dortmund on engines whose exhaust steam was applied to driving rolling mills. The result showed that the steam consumption of a compound rolling-mill engine—

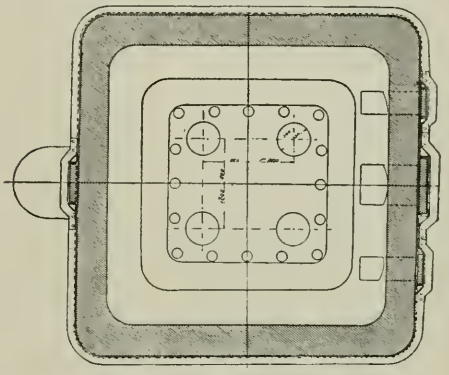


FIG. 2.—PLAN OF GIROD FURNACE.

that is, a very economically working one—was increased by 35 per cent by the interpolation of an exhaust-steam turbine between it and the central condenser. The experiments led to the conclusion that no economical advantage results from the use of such a turbine, for the chief reason that the extra 35 per cent of steam used in the reversing engine would be sufficient to develop the same amount of energy in a live-steam turbine as would be obtained by means of installing the rather complicated and costly exhaust-steam turbine plant. (From advance sheets.)

Electric Steel Furnace.—Prof. W. BORCHERS has written another paper on "The Girod Furnace," this time for the Iron and Steel Institute. He emphasizes again his opinion that "the Girod furnace is the simplest of modern electric steel furnaces, both as an experimental furnace, or as a smelting works furnace on the largest scale possible. In the author's laboratory it was being used with current of 30 kw to 36 kw, while elsewhere, for steel smelting, it is being used with currents of 300 kw to 1200 kw." Figs. 1, 2 and 3 are sections of a 12-ton Girod furnace. There are four upper electrodes and 16 "contact pieces" in the bottom. The furnace requires 1000 kw to 1200 kw at 70 volts to 75 volts. Taking into account a loss of 10 per cent to 11 per cent of the charge by oxidation and evaporation, the power required to make one ton of steel in this furnace is 800 kw-hours to 900 kw-hours starting with cold charges. The consumption of carbon electrodes amounts to 12 kg or 15 kg per ton of steel, including the short pieces, which remain in the holders when the latter have reached the lowest point at which they can withstand the radiating heat of the bath. The lining of the furnaces, mostly consisting of calcined dolomite, will withstand at least 80 charges. Thereafter the side walls, into which cavities have been eaten by the slag around the level of the molten bath, will require repairing, and these repairs will subsequently need to be effected at shorter intervals. The bottom of the furnace will, however, stand 120 to 150 charges without repair. The thickness of the bottom will by that time have been reduced by about 100 mm. Furnace linings will stand a considerably longer time if liquid charges are used. One furnace working in this manner was run for 200 charges without any repairs. The linings of the furnace covers require more frequent renewal. In the small furnaces they need repairs after 25 to 30 charges, and in the

larger furnaces after 20 to 25 charges. The other parts of the furnace are not worn out as fast as the linings and electrodes. The furnaces which have so far been put into operation still possess their original outfit. At the works at Ugine only scrap iron is used as raw material for steel making as well as for the manufacture of iron alloys. The larger part of the scrap is soft iron and steel, the smaller part pig and cast iron. Some notes are added on the equipment of the works

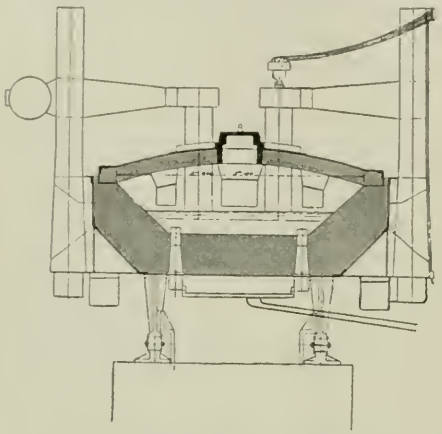


FIG. 3.—VERTICAL SECTION OF GIROD FURNACE.

of the Society Anonyme Electrometallurgique, Paul Girod. (From advance sheets.)

Carbides in Manganese Steel.—An Iron and Steel Institute paper by Prof. J. O. Arnold and Prof. A. A. Read deals with "the chemical and mechanical relations of iron, manganese and carbon." The authors made a number of experiments to determine the composition of the carbides separated from a series of annealed steels containing various percentages of manganese, the percentage of carbon being practically the same in each. The alloys were made as nearly as possible to contain about 0.9 per cent of carbon with ascending manganese. The color of the carbides obtained by separation ranged from bright silvery gray in the lowest to dull gray in the steel with the highest percentage of manganese. The percentage of carbon in the carbide was found in most cases somewhat higher than the theoretical amount 6.7 and also, in some cases, the total amount of carbon in the steel was not obtained as carbide. This is only to be expected when dealing with a substance like manganese carbide, which easily decomposes.

As the manganese rises so does the amount of iron replaced by manganese in the carbide go on increasing until a steel with 4.68 per cent of manganese is reached. From this point to a steel with 13.38 per cent of manganese, the manganese in the carbide remains stationary. With steel containing 15 per cent of manganese, the manganese in the carbide begins to increase again, and probably with a 20 per cent manganese steel a further replacement of iron by manganese takes place. It is somewhat difficult to say whether the double carbides are true compounds or simple mixtures of iron and manganese carbides. It would appear that the carbides from the steels up to 4.68 per cent of manganese are simply mixtures, and possibly a considerable amount of a true double carbide, $3Fe_2C$, Mn_3C is present in steels containing 11.21 per cent and 13.38 per cent of manganese respectively. In steels with 15 per cent of manganese and upwards, a double carbide containing a higher percentage of manganese, such as $2Fe_2C$, Mn_3C, or a mixture of carbides, may be formed, a point being reached resulting in the formation of another true double carbide. A mixture of carbide of iron and manganese, or a double carbide of iron and manganese, registers a lower percentage of carbon by the color test than carbide of iron per se. With steels up to 10.07 per cent manganese, the percentage of total carbon agrees very closely with the amount of carbide of iron present in the carbide, thus showing that manganese carbide gives no coloration with HNO_3, specific gravity 1.20. The increase of total carbon obtained by the color test with steels containing 11.21 per cent and 13.38 per cent manganese seems to point to the formation of a compound capable of giving a slightly higher color. Results of micrographic analysis are also given. (From advance sheets.)

Tool Steels.—An Iron and Steel Institute paper by Mr. Edward G. Herbert deals with "the cutting properties of tool steels." Tool steels are steels which can be made hard enough to cut metals. But hardness alone does not determine the quality of tool steel, since a tool should not only cut, but continue cutting for a considerable time before becoming blunt. The author has devised a tool-steel testing machine, the principle of which is shown in Fig. 4, where *A* represents the element on which the cutting tests are made: that is, a tube, usually of steel, ¾ in. in diameter and ⅝ in. bore, formed by drilling from the solid bar, and subjected to a heat treatment which renders it uniform in hardness. The tube *A* revolves about a vertical axis, while its lower end rests on a fixed abutment *B* of hardened steel. The sample of steel to be tested is made into a cutting tool *C*, which is ground to a standard angle and held in a vise *D* with its cutting edge radial to the tube. The vise *D* is mounted on knife edges *E* lying in the plane of the end of the tube. By means of a steelyard *F*, with a movable weight and scale, the tool can be caused to press upward against the edge of the tube with any desired force, and cut it away with a turning action. The tube is held in contact with the abutment *B* by means of a weight *G*, the downward pressure of which is always greater than the upward pressure of the tool. As the tube is turned away by the tool, it is fed downward by the weight *G*, so that the end of the tube, though it is constantly being turned away, is constantly held in contact with the abutment *B* and the point at which cutting takes place is stationary. A dashpot *H* is connected to the vise to prevent jarring or vibration. As the test proceeds, the cutting edge is gradually worn away, and as this takes place the vise tilts on the knife edges. The angle through which the vise tilts is proportional to the blunting of the tool, and this is actually measured in thousandths of an inch by means of the micrometer *K*, which is mounted on the pillar of the machine, and is brought to bear on a projection from the end of the steelyard, so as to measure its movements due to the wear of the tool.

Fig. 5 gives a view of the tool-testing machine. The tube

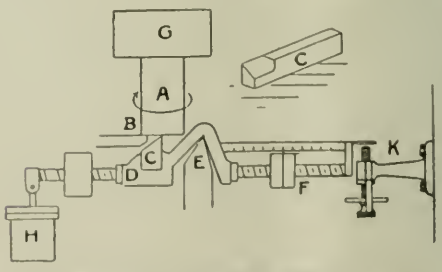


FIG. 4.—DIAGRAM OF TOOL-STEEL TESTING MACHINE.

is driven by a spindle, which receives its motion through cone and friction disk gearing, capable of giving it a wide and continuous range of speeds. A paper-covered drum, mounted beside the column of the machine, is driven through worm-and-spur speed-reducing gearing from the spindle. Vertical movements of the spindle are communicated through a fine chain to a pencil mounted on a vertical sliding bar. When the ma

chine is set in motion the pencil draws a diagram, in which horizontal distances represent the number of revolutions made by the tube, and vertical distances represent vertical movements of the spindle—i.e., the length of tube turned away by the tool. The slope of the line represents the rate at which the tube is being turned away, or, in other words, the feed or traverse of the tool in inches per 1000 revolutions of the tube. By means of this instrument tests can be made at varying speeds up to the highest speed at which the tool will cut, and the results plotted in form of "speed curves" in which the ordinates represent the durability of the tool and the abscissas represent cutting speeds.

A set of speed curves is shown in Fig. 6, in which *A* is the curve of a carbon steel. *B*, *B*¹ are both from the same high-speed steel differently hardened, and *C* is from another high-

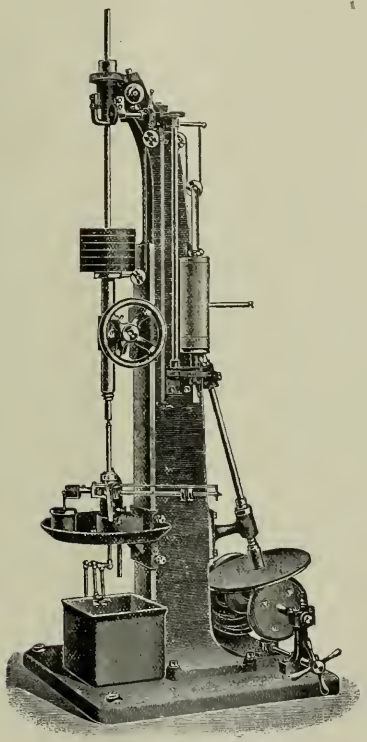


FIG. 5.—TOOL STEEL TESTING MACHINE.

speed steel. At the low speeds, up to 50 ft. per minute, the carbon steel is more durable than the high-speed steels; but at the higher speeds the carbon steel is (as might be expected) less durable. The fact that a high-speed steel tool will not keep its edge so well as a carbon-steel tool under a light cut at a low speed (a finishing cut) has long been familiar to engineers, but the fact that the durability of a high-speed steel tool working under a light cut, can be greatly increased by running it at a high speed, and that under these conditions—a light cut and a high speed—it will keep its sharp edge better than a carbon-steel tool, was not generally known. Perhaps the most striking and unexpected feature of the speed curve is this: that the durability of all steels without the exception is very low at low speeds (under light cuts) and that the durability of the cutting edge increases as the speed is raised. In considering these curious facts, it is important to bear in mind that we are here dealing with durability as measured, not by the length of time the tool will continue cutting before becoming blunt, but by the amount of metal it will cut away before becoming blunt. To turn away an inch of tube at 40 ft. per minute obviously takes twice as long as turning away an inch at 70 ft. per minute cutting speed, the thickness of the shaving being the same in each case. If the speed curves are drawn out on a time basis they assume a rather different form, as shown in Fig. 7. From this it appears that the maximum time durability of the carbon steel was actually higher than that of the best high-speed steel, but that this maximum occurred at a low speed. Neither the curves in Fig. 6 nor those in Fig. 7 give a correct view of the relative value of carbon and high-speed steels for the rapid removal of metal. The best steel for this purpose is not the one that will keep its edge longest when run at a slow speed, nor even the one that will remove the greatest amount of metal before becoming blunt. The engineer usually requires the steel that will remove the greatest amount of metal per hour without requiring too frequent sharpening. If we multiply the number of inches of tube turned away by the corresponding cutting speed, we obtain a quantity which may be termed the "duty" and which is proportional to the time rate of removing the metal, and to the durability of the tool. It is reasonable to assume that when this quantity attains a maximum the ideal working conditions have been reached. In Fig. 8 are shown duty curves obtained from the speed curves, Fig. 6, and this diagram shows the relative values of the steels for the rapid removal of the metal. Fig. 9 gives an even more striking comparison between the duty curves of A, a carbon steel; B, a good high-speed steel, and D one of the new vanadium steels. When the vanadium steels were first brought out it was generally supposed that they were not capable of working at higher speeds than the older tungsten steels. The tool-steel testing machine has shown that they are capable of working, and that they may attain their maximum durability at speeds which are much too high for the ordinary tungsten steels, and this has now been amply confirmed in practice.

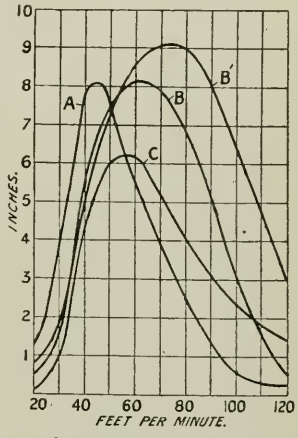


FIG. 6.—SPEED CURVES. (A, CARBON STEEL; B, HIGH-SPEED STEEL; B', SAME TOOL REHARDENED; C, HIGH SPEED).

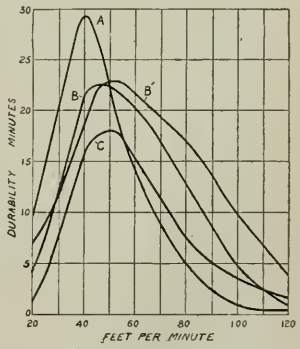


FIG. 7.—SPEED CURVES ON TIME-DURABILITY BASIS.

The author then discusses at some length the causes of the differences in the durability of the cutting edge of a given

steel at different speeds. The durability of the cutting edge cannot depend on the rubbing of metal against metal alone It seems that it depends also to a large extent on the evolution of heat due to the rubbing and causing a rise in temperature of the cutting edge. It seems that each speed on the testing machine corresponds to a definite temperature of the cutting edge and that the durability of a tool (of a given shape and cutting a given material) depends entirely on its temperature. From this point of view the speed curve is really a temperature curve. While the above curves are simple in form, more complicated curves often appear, showing two peaks with a distinct depression between the two. This corresponds to an almost total loss of durability at a particular speed which may be intermediate between two speeds, each of which gives the tool the highest durability of which it is capable. Unaccountable failures of cutting tools are among the commonest of workshop experiences and may be frequently due to working at the critical temperature of low durability. The concluding chapters of the very long paper deal with the effect of temper on the durability of carbon steels and the effect of percentage of carbon on durability, and finally the effect of the cooling process on the durability of high-speed steels. (From advance sheets.)

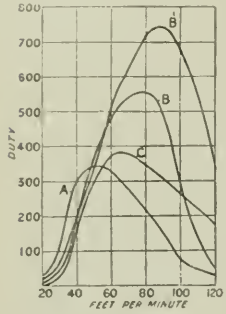


FIG. 8.—DUTY CURVES.

Chromium Steels.—An Iron and Steel Institute paper by Prof. A. McWilliam and Mr. Ernest J. Barnes deals with "some physical properties of 2 per cent chromium steels." Tensile tests, the effect of heat treatment, heating and cooling curves, and micro-photographs are described and discussed. Another Iron and Steel Institute paper by Mr. Harold Moore deals with "the A-2 point in chromium steel." The following conclusions are reached: The position of Ac1 is progressively raised by additions of chromium to steel, and if sufficient chromium (3 per cent) is present, another critical point occuring at about 777° C. and below Ac1 is indicated by heating curves. The identity of this lower critical point with the magnetic change point proves it to be Ac2. The occurrence of Ac2 below Ac1 is held to support the view that Fe_3C is insoluble in β-iron. The method of determining the magnetic change point, described above, is simple and accurate (From advance sheets.)

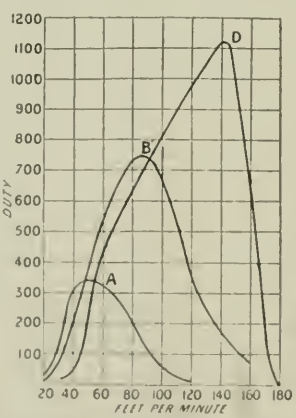


FIG. 9.—DUTY CURVES. (A, CARBON STEEL; B', HIGH-SPEED STEEL; D, THE NEW STEEL.)

An account of the discussion which followed the different papers, abstracted above, at the recent meeting of the Iron and Steel Institute, will be found in the letter of our London correspondent in this issue.

Alleged Lack of Uniformity of Electric Steel.—In this Journal, January, 1910, Gronwall had claimed that in many cases in practice there was found lack of homogeneity and uniformity of steel produced in electric furnaces from cold charges. This was claimed to be due to differences of temperatures within the furnace. In *Stahl und Eisen* of April 20, V. Engelhardt replies to this attack as far as the induction furnace is concerned and gives some interesting notes on the automatic motions which are always observed in induction furnaces. They are indicated in the adjoining diagram, in which 1 represents a simple induction furnace, 2 a Roechling-Rodenhauser furnace operated with single-phase current and 3 a Roechling-Rodenhauser furnace operated with three-phase currents. In all of them there are two different kinds of motions. First, the rolling motion perpendicular to the surface of the steel bath and directed toward the inner wall of the crucible. Second, a rotary motion parallel to the surface of the bath, at a lower speed. The motion is not so strong that the steel and slag would get mixed up, but is strong enough to produce absolute uniformity of the steel This has also been confirmed in actual practice.

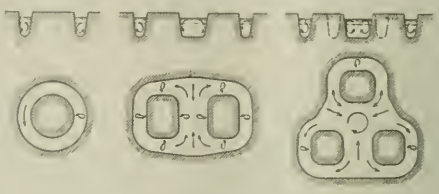


FIG. 10.—MOVEMENT OF CHARGE IN INDUCTION FURNACE.

Titanium in Steel.—An article by W. Venator in *Stahl und Eisen* of April 20 gives the results obtained in practice in a steel works in Osnabrück, in Germany, with the use of titanium in the manufacture of basic open-hearth steel. The molten steel was divided into two halves, to one of which titanium was added, the additions varying between 0.038 per cent and 0.14 per cent. Detailed comparative tests were then made between the steel treated with titanium and the untreated steel. The ingots of the titanium-treated steel were found to be perfectly sound in rolling. The chemical analysis showed that the titanium was present only in traces in the steel, it having passed chiefly into the slag. The mechanical tests showed conclusively that the treatment with titanium is advantageous, as it toughens the steel: it also has an effect on the microstructure, as is shown by the appearance of the fracture. The results of the mechanical tests are given in detail. The author thinks that the success of the titanium treatment is mainly due to very effective dioxidation. That the titanium also removes nitrogen from steel needs to be proven. The titanium acts simply as a purifying, not as an alloying, agent. It is stated that in a crucible steel works favorable results were obtained by adding metallic titanium mixed with aluminium file to the steel before pouring. It was found that the aluminium had no disadvantageous effect on the steel It apparently facilitates the diffusion of the titanium

Gold and Silver.

Chlorine in the Cyanidation of Silver Ores.—The development of the electrolytic cell for the cheap production of chlorine is giving rise to a number of investigations looking toward the use of this element in the extraction of the precious metals It is well known that chloridizing roasts can be utilized for treating refractory silver ores, but recent investigators tend more and more to adopt more direct methods of treatment. Writing in the *Pacific Miner* for May, 1910, D. Mosher suggests a scheme for chloridizing silver ores containing antimony and other base combinations by the direct application of chlorine in a chlorine-proof tank, subsequently extracting the silver with cyanide. His idea is to first slime the ore and treat the pulp in the ratio of one of slime to one of water heated to 120° Fahr. to 150° Fahr., by passing chlorine into the sludge until tests show the conversion of the sulphides into

chlorides. Caustic soda formed in the cell which produces the chlorine may be used to neutralize any acid salts formed in the reaction. After dilution and subsequent thickening of the chloridized slime, silver and gold are to be extracted by cyanide in any customary manner. The construction of the chlorine-proof tank is suggested as follows: An outer steel shell lined with reinforced concrete, attached to which is a double layer of acid-proof brick set in acid-proof cement.

Cyanide Practice in Western Australia.—An outline of the treatment of concentrates at five mills operating on the "Golden Mile," Kalgoorlie, Western Australia, is given by W. M. VON BERNEWITZ in the May 21, 1910 issue of the *Mining Journal* (London). Wet crushing, concentrating, roasting and cyanidation are the main features of the process. Edwards Simplex and Duplex roasters are used at four of the five mills, the other using the Merton Improved. At all of the plants except one a sweet roast is made. At the other, 4 lb. to 5 lb. of salt are added to the roaster charge and a poorer roast is made. Following the roasting process the product is treated in grinding pans so that about 80 per cent passes a 200-mesh screen. Agitation with weak cyanide solution, filtration and washing in filter presses, and precipitation in zinc boxes complete the steps in the process. The amount of flue dust formed is stated to be very low, one plant getting about 0.4 per cent of the raw tonnage roasted. The cyanide solutions in which the slime is agitated vary in strength from 0.03 to 0.1 per cent KCN. Air agitation is used for about 12 to 36 hours. Filtration is carried on under 75 lb. to 100 lb. pressure, and washing at from 50 lb. to 90 lb.

Chlorination and Charcoal Precipitation.—An interesting description of the Edwards Pyrites Works, Bendigo, Victoria, Australia, is given by DONALD CLARK in the *Australian Mining Standard* for April 27, 1910. Sulphide concentrates are roasted in Edwards Duplex and tilting furnaces. The capacity of the Duplex roaster is given as about 14 tons of clean pyritic concentrates in 24 hours, using about 4.5 tons of wood. The cooled roasted ore is mixed with about 10 per cent water and charged into chlorination vats. Chlorine is forced in at the base of the vats under slight pressure and allowed to flow as long as it is absorbed. From 24 to 48 hours are required for complete solution of the gold as chloride, after which water is turned into the vats at the top and allowed to carry the gold chloride in solution through the customary gravel-sand filter. The gold solution is pumped to suitable storage vats, whence it flows through the precipitating jars. The latter are in three series arranged in steps. Each jar is 3 ft. in height with a diameter of 1.5 ft. An earthenware pipe is set vertically in the center of each jar, and the space between this and the sides of the jar is packed with lumps of clean charcoal. The pipe may also be similarly packed. The gold solution is made to flow at the rate of about 3 liters per minute downward through the central pipe, upward through the charcoal outside and thence to the next jar. After passing the series of three jars the solution is conducted through two large carcoal-filled vats before it is allowed to run to waste. The precipitation is said to be remarkably efficient, and 90 per cent of the gold will be precipitated in the first series of jars. The charcoal is slowly burned and the ash melted for recovery of the gold. Chlorine is generated from manganese dioxide, salt and sulphuric acid.

Copper.

Bessemerizing Copper Matte and Copper-Nickel Matte.—A study of the behavior of copper-nickel mattes in the Bessemer converter, extending over a period of five years, is contributed to the April *Bulletin* of the American Institute of Mining Engineers by DAVID H. BROWNE, metallurgist of the Canadian Copper Company. Nickel was formerly regarded as an element replacing iron in copper matte, but as a result of an extensive investigation the author announces certain conclusions, as follows:

1. Nickel is not an element replacing iron in matte.
2. Nickel-copper alloys act in the matte-blow like one metal.
3. Nickel-copper alloys follow, in the matte-blow, exactly the same laws as govern the behavior of copper alone.

In arriving at the laws governing the behavior of copper during the blow, and in determining the relations of copper to sulphur and iron, the author makes use of data previously contributed by other investigators, notably Mathewson. By recalculating Mathewson's analyses showing the "rate of elimination" of matte impurities, to express "ratios" of iron and sulphur to copper present in matte at a given time, the author shows that the curve of iron elimination is a regular mathematical curve, and that, on the whole, sulphur disappears in a straight line, with a curious jog or reversal when the iron is down to 5 per cent. The conclusion is, therefore, reached that "relative elimination" is a fixed factor, no matter how much the "rate of elimination," or blowing time may vary. This suggests the idea of the laws governing the action of copper matte in the Bessemer converter. Experiments at the plant of the Canadian Copper Company show that in bessemerizing copper-nickel matte, nickel does not follow the iron into the slag; that only in the last stages of the blow does it oxidize; that when it does pass into the slag it carries more than its own weight of copper with it, and that it is impossible to blow all of the nickel out of the matte. The elimination of the sulphur and iron is, therefore, a direct function of copper-nickel in such mattes, and not of either metal separately.

Vacuum Concentration.—The application of the Elmore process to hitherto wasted tailings at Sulitelma, Norway, within the Arctic Circle, is an interesting development resulting from experiments begun several years ago. H. HOLMSEN and H. N. REES describe the plant in the May *Mining Magazine*. At present a plant of 12 Elmore vacuum concentrators is treating 500 tons of tailings per day, producing 1000 tons of concentrates per month. The annual production of metallic copper from this plant is from 800 tons to 1000 tons. The vacuum plant (Fig. 11) treats tailings from jigs and concentrating tables in which there was formerly a loss of about 40 per cent copper. This material is delivered to a large spitzkasten, the coarse underflow from which is reground in Krupp ball mills, the overflow going to settling tanks. The overflow from the latter goes to waste. The crushed product from the ball mills, the fine from the spitzkasten and the slime from the settling tanks are delivered to a rotary distributor which accurately divides the pulp into equal parts for the separate units of the Elmore plant. The consumption of oil is small, amounting to 2.7 lb. per ton of material treated. No acid is used. Formerly sulphuric acid was used at the rate of 3 lb. per ton of material treated, and later experiments with sulphurous acid prepared by passing roaster gases through water also gave satisfactory results. Neither is used now, although slightly better results might be obtained by the use of acid.

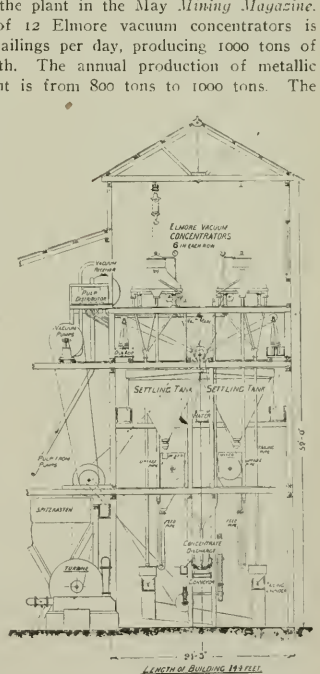


FIG. 11.—VACUUM PLANT AT SULITELMA.

The following figures indicate the efficiency of the process without the use of acid:

	Per Cent Copper.
Tailing from jigs and tables	0.8 to 1.4
Concentrates	6.0 to 8.5
Final residue	0.15 to 0.3
Extraction	Over 80 per cent.

The labor employed is one man for each floor per shift of eight hours, viz., at the separators on the top floor, at the mixers on the first floor and at the concentrate discharge on the ground floor. The plant has had constant operation, losing but 25 hours in eight months.

Converting High-Grade Mattes.—In the March 5, 1910, issue of the *Aust. Min. & Eng. Rev.*, W. A. Longbottom describes the successful bessemerizing of matte carrying 60 per cent to 70 per cent copper. Owing to a shortage of suitable low-grade material the furnaces of the Electrolytic Refining & Smelting Company become filled with high-grade matte, and although the copper content of the material was far higher than was considered advisable for treatment in the Bessemer converter, it was decided to attempt the experiment. The converters are 10 ft. 6 in. by 7 ft. 6 in., and the air pressure available was only 9 lb. per square inch. The greatest trouble experienced was in keeping the tuyeres clear, and a man was kept constantly punching to accomplish this purpose. The slag was very viscous and at the time of skimming it was necessary to rabble the slag off from the converter. The linings withstood the operation much better than with the more fiery low-grade matte. The time required to blow the slag averaged from 50 to 60 minutes, but the time required to blow to blister was relatively long, being as high as 2 hours and 50 minutes to 3 hours. The best results were obtained with a charge of about 6 tons of matte, and it was found that if this quantity was much increased the time required to blow to blister was greatly increased, probably on account of the low air pressure and the loss of tuyeres due to slagging. The author recommends the alternate treatment of high and low-grade mattes in the converter, rather than mixing two mattes of extreme grade. An experiment of the latter kind gave poor results.

Vortex Copper Converter.—In the ordinary form of copper converter the air is introduced at the angle of greatest resistance and the force of the air current is spent largely in overcoming frictional resistance due to impact in encountering the wall of liquid matte. Thus the power stored up in the many air currents is not used to great advantage. In the May 7, 1910 issue of the *Eng. & Min. Journ.*, Herbert Haas describes his suggestion for a vortex converter, in which the power of the air currents is used to set up a rotation of the converter contents and thereby insure a more thorough mixing of the various constituents and a more thorough and rapid oxidation of the iron and sulphur. He uses a spherical converter blown from the bottom. The tuyeres are so located and designed that the air currents are directed forward, outward and upward, thus following the line of least resistance and imparting a motion to the mass. This would cause even corrosion of the lining, eliminate the necessity of punching the tuyeres and prevent the formation of noses. A still further advantage would lie in the thorough mixing of the constituents Cu_2S and Cu_2O, with consequent reduction of copper. Owing to the difference in specific gravity of these constituents they tend normally to separate. The vortex converter has never been placed in operation and is offered as a suggested improvement.

Electrolytic Refining.—In five issues of the *Aust. Min. Stand.*, from Feb. 16 to March 16, 1910, inclusive, H. Schroder describes in detail the Australian practice in electrolytic copper refining as carried out at Wallaroo, Lithgow and Port Kembla. The author has been officially connected with each of the three plants. Only a few points can be selected from the details given in the series of articles. At Lithgow the tanks are constructed of Oregon pine with hardwood studs and wrought iron straps and stays. They are lined with 6 lb. lead and have an additional wooden lining of redwood, the bottom of which is in the shape of a tray 6 in. deep. The pine, hardwood and the top board of redwood are dipped in paraffin, which lengthens the life of the tank.

The busbars rest on porcelain insulators, and the dead ends of anodes and cathodes rest on glass blocks. The busbars are of sufficient cross-sectional area to carry at least 500 amp per square inch. The composition of the electrolyte is about 12 per cent $CuSO_4$, 12 per cent free H_2SO_4; the specific gravity is 1.16 to 1.18, and the temperature 120° Fahr. Flow is maintained at the rate of 3 gal. to 4 gal. per minute. A small percentage of soluble chloride is also carried. Current density employed is from 14 amp to 18 amp per square foot. The starting sheets are deposited on rolled copper plates coated with a mixture of oil, black grease and kerosene, or tallow and graphite. In stripping the cathodes care is exercised to withdraw a uniform quantity from the tanks daily, proportionate to the theoretical quantity which should be deposited. The Australian custom is to withdraw from 90 per cent to 95 per cent of the theoretical amount. Anode scrap amounts to 10 per cent to 15 per cent of the anodes loaded. Ninety per cent of the ampere efficiency is obtained.

Anode slime is treated in one of three different ways. First, it may be treated with sodium nitrate and sulphuric acid to dissolve the copper; then with metallic copper to precipitate silver, after which it is washed, dried and smelted with sodium bicarbonate. In the second method the slime is screened and then boiled with sulphuric acid, with subsequent treatment as in the first method. The third method is similar to the first, except that whereas the sodium nitrate and sulphuric acid are previously mixed together before being added to the slime in the first method, they are added separately in the third. In the treatment of the Dore bullion obtained from the smelting of the slime, two of the plants use the old Moebius process of electrolytic refining, while the third uses the Balbach cell.

The cost of electrolytic refining of copper under Australian conditions is given as about 24 s. ($6) for electrolyzing only, per ton of cathode copper, in a plant producing 35 tons per day. To this must be added 2 s. 6 d. ($0.62) per ton for retreatment of scrap. The cost of refining Dore bullion depends very largely on the size of the plant. With a small two-cell Moebius plant capable of depositing 3000 oz. of silver per week, the tetual bullion treated was 12,600 oz. at a cost of 0.146 d. (0.3 cents) per ounce. To this must be added the cost of melting, which was 0.103 d. (0.206) cents per ounce, or a total cost of 0.249 d. (0.5 cents) per ounce.

Tin.

Analytical Methods.—Sodium peroxide has proved to be a most useful reagent in analysis, especially in decomposing those minerals which are refractory to ordinary agents. Cleanliness and rapidity are also favorable factors in its use. In the March, 1910 *Journal* of the Chem. Met. & Min. Soc. of South Africa, James Gray gives a résumé of fire and wet methods for assaying tin ores. Fire methods are rightly dismissed as being inaccurate at the best. Of the wet methods (eight being given), the one involving fusion with sodium peroxide, extraction with water and solution with hydrochloric acid, is reported as being the most satisfactory and giving the best results. The details of the method follow. From 0.5 gram to 2 grams of the finely powdered ore are fused from four to six times the same weight of sodium peroxide in an iron crucible. On cooling, the melt is extracted with water. This solution is treated in a flask with an excess of HCl, heated to about 90° C., and reduced with iron nails for 40 to 60 minutes after the solution has become colorless. The flask and contents are then rapidly cooled, a piece of marble added to maintain a neutral atmosphere, and when cold, a little starch paste is added. The solution is then titrated with iodine solution which has been previously standardized against pure tin. A blue color indicates the end point. The author gives figures showing that the presence of varying quantities of arsenic,

antimony, bismuth, tungsten and titanium have little influence on the accurate results obtained by this method. A bibliography is appended.

Chemical Engineering.

Manufacture of Sulphuric Acid.—The Ducktown Sulphur, Copper & Iron Company, Isabella, Tenn., operates a pyritic smelting plant treating pyrrhotite and manufactures sulphuric acid from the blast-furnace gases. W. H. FREELAND and C. W. RENWICK, respectively, former general manager and present general manager of the company, describe the sulphuric acid plant in the May 28, 1910, issue of the *Eng. & Min. Journ.* Blast-furnace gases from a pyritic smelter are not converted easily into sulphuric acid on account of the varying composition and temperature at which they are delivered, and the flue dust which they contain. The Ducktown company first experimented with the ferric oxide contact process, but found that it was not adapted to the conditions existing in copper smelting. The chamber process, later installed, has proved very satisfactory. The gases to be treated have approximately the following composition: SO_2, 3.5 per cent; CO_2, 3.5 per cent; SO_3, trace. The temperature of the gases frequently varies over 200° C. in 8 or 10 minutes. Flue dust is settled out in flues and dust chambers, and finally in specially constructed dust chambers in which a system of channel irons is arranged to catch the fine particles which have not settled out before. The control of the temperature is accomplished by leading the gases through a greater or less length of a kite-shaped flue leading to the special dust chamber. When the gases are hot they are directed through the entire length of the flue and when cooler through a shorter length. There are four entrances from flue to chamber, through which the gases may pass. Having been thus cleaned, the gases are passed through Glover towers, lead chambers and Gay Lussac towers. A high oxidation of the gas is obtained in the Glover towers, fully 40 per cent being converted into SO_3. The plant has a rated capacity of 160 tons of 60° B. acid per day, but has yielded 200 tons when necessary.

In the *Mining Journal* (London) for April 30 and May 7, 1910, Prof. GIUSEPPE ODDO, of the University of Pavia, discusses the possibilities of manufacturing sulphuric acid from rock sulphur. About two years ago he suggested the idea as a solution of the crisis existing in the Sicilian sulphur market. It was not well received locally because it opposed some traditional customs and empirical ideas. Nevertheless the idea has been worked on and considerable advance has been made in the theory and practice suggested by Professor Oddo. His proposal is to use the fine sulphur rock, which is unsuited for melting, and the poorer grades of melted sulphur which now exist in large stocks on hand, to make briquettes carrying 50 per cent available sulphur. This will utilize one material that is now going to waste and another for which there is little or no demand. The use of sulphur mineral for making sulphuric acid has several notable advantages over pyrites, which Professor Oddo briefly recapitulates as follows:

(a) A purer current of gas which will make possible still further purification by the contact process, and the manufacture of a purer acid for certain purposes.

(b) A current of gas richer in SO_2, in the ratio of about 1.3:1, which will give proportional economy in plant and operation.

(c) Longer life of the lead chambers in the same proportion and for the same cause.

(d) Facility and regularity of combustion, with an economy of about one-half in manual labor, with 50 per cent briquettes.

(e) Less loss of sulphur in combustion which will only be brought about by a limited formation of calcium sulphide and sulphate. This will be negligible since sulphur will begin to react with the gypsum and limestone in the rock only at a temperature above 420° C., and then very slowly.

(f) Smaller weight of cinder compared with the percentage of sulphur. The best pyrites with 50 per cent sulphur give 68 per cent, while 40 per cent and 45 per cent pyrites give 74.9 per cent and 71.8 per cent, respectively. Briquettes will produce only 50 per cent cinder.

Taking local conditions into consideration, Professor Oddo has worked out the economic features of the industry and has shown that rock sulphur can compete easily with pyrites in the manufacture of acid, and that a concerted action is all that is necessary to open this new industry and put an end to industrial misfortune in the Sicilian sulphur market.

Electrochemistry.

Fixation of Atmospheric Nitrogen.—The old question whether the forced combination of the nitrogen and oxygen in air by means of electric arc discharges is simply a thermal phenomenon (representing the equilibrium at the existing high temperature and independent of the method of producing this temperature) or whether specific electrical effects also play a part in this formation, is again discussed in an interesting paper of Dr. W. HOLWECH in *Zeit. f. Electrochemie*, June 1. Haber and Koenig have formerly shown that at the high temperatures employed in the commercial processes in operation in Norway the reaction seems really to be a purely thermal phenomenon; however, they also have shown that at lower temperatures it is possible to produce more nitrogen oxide than would correspond to the thermal equilibrium. The present author has made experiments with very short direct-current arcs between a cold anode and a hot cathode. He used a water-cooled silver anode and as cathode an iridium sheet which was brought to incandescence by an electric current from an independent source. In order to facilitate the starting of the arc and to reduce the cathode drop of voltage the iridium sheet was covered with calcium oxide. He produced up to 80 grams HNO_3 per kw-hour. The shorter the arc the higher the concentration at minimum speed of the gas. The highest concentrations obtained were 9 per cent NO and more in air. The voltage drop per unit length in the arc was investigated and it was found that although the anode is cold and free from electrode vapor and the cathode is hot, the voltage drop per unit length remains constant within the arc with the single exception of the immediate neighborhood of the two electrodes. It was found that the higher the voltage per unit length the greater is the output and the maximum concentration which can be obtained. This is true as long as the temperature of the arc is not too high. If the temperature is raised the thermic phenomenon hides the electrical phenomenon and it is no longer possible to obtain the same maximum concentration as before. In the commercial furnaces the voltage drop per unit length is less than 10 volts per centimeter. In the author's experiments it was between 100 volts and 200 volts per centimeter. The output in practice is about 60 grams to 75 grams of HNO_3 per kw-hour. The output in the author's experiments is about the same. On the other hand, the maximum concentration which can be obtained is higher in the author's experiments than in practice. The author thinks that in the commercial furnace the thermal phenomenon predominates, while in his experiments the electrical phenomenon predominates in the formation of NO. In his direct-current arcs the temperature was so low that the points of the finest iridium wire did not soften.

Electrolytic Refining of Bismuth.—In the electrolytic refining of lead by the Betts process the anode slime contains silver and bismuth. In *Zeit. f. Electrochemie*, of April 15, F. FOERSTER and E. SCHWABE propose to subject this silver-bismuth to a further refining process in a silico fluoride solution of bismuth as electrolyte. Pure bismuth is deposited on the cathode. The lead contained in the anode passes into the electrolyte. The silver remains back at the anode. The bismuth deposited on the cathode is free from lead.

Alkali Bromide Electrolysis.—A paper by F. FOERSTER and J. YAMASAKI in *Zeit. f. Electrochemie*, April 15, deals with details of the ionic mechanism of the electrolysis of alkali

bromides. Experiments which were made with a well-stirred electrolyte showed that the primary anodic reaction is the discharge of bromine ions, sometimes combined with a discharge of hydroxyl ions. By secondary reactions which are sometimes purely chemical and sometimes electrochemical bromate, etc., is formed. The effects of variations of current density, temperature, concentration of bromide and alkali were investigated, both with platinized platinum anode and with a polished platinum anode.

Potassium Permanganate.—In *Zeit. f. Electrochemie* of March 1 P. Askenasy and S. Klonowski discuss the electrolytic production of potassium permanganate from solution of potassium manganate. The authors have specially investigated the possibility of getting along without a diaphragm. They used an iron-wire cathode and a large concentric iron sheet as anode. The best amp-hour efficiency was obtained with a cathode current density of 0.9 amp to 0.7 amp per square centimeter in a 7 per cent to 10 per cent manganate solution in about 10 per cent potassium hydroxide. The authors also made some experiments with diaphragms and finally substituted platinum anodes for the iron anodes. Recommendations are added on the commercial manufacture of potassium manganese. As far as the electrolytic part is concerned it is recommended to use an iron vessel, which serves as anode, and iron cathodes without a diaphragm. The cathodic current density should be 85 amp per square decimeter, the anodic current density 10 times smaller. Distance between anode and cathode about 20 mm; 20,000 amp-hours are required for 100 kg of manganate. The voltage is 2.8. About two-thirds of the manganate are changed into permanganate; 1 kg of permanganate requires about 0.7 kw-hour.

Nickel Wire Furnace.—In *Zeit. f. Electrochemie*, May 15, R. Lorenz and G. von Hevesy record their experience with resistance furnaces, using nickel wire. They employ nickel wire covered with asbestos. The life of such a furnace if heated to not more than 700° C. is long, but at higher temperatures the nickel is destroyed. This is due to the formation of nickel oxide on the surface of the wire; this then enters into the interior and the nickel gets brittle and breaks. If higher temperatures up to about 1200° C. are to be used, it is necessary to protect the nickel from the oxygen of the air. This can be done by an air-tight construction.

Recent Metallurgical and Chemical Patents

Gold and Silver.

Cyaniding Apparatus.—The rapid dissolution of gold and silver from complex sulphide and silicious ores, especially those of low grade, is the object of an apparatus patented by P. A. Brangier, of Agnew, Cal., and Royer Luckenbach, of Philadelphia, Pa. The general principle involved is the flow of a thin stream of the pulp over inclined surfaces and the application of heat and compressed air at certain points for the purpose of increasing the solvent action of the cyanide. The apparatus is in the form of a box about 10 ft. long and 3 ft. wide and as high as desirable. The box contains a series of superimposed plates inclined in opposite directions from one end of the box to the other. The upper two-thirds portion of each plate is heated by steam and the lower third is convex with air pipes at the sides. As the pulp flows through the apparatus by gravity, traversing first one plate in one direction and the next in the opposite direction, it first flows over the heated portion onto the convex portion. Here the pulp tends to run to the sides of the plate and is thoroughly aerated and scoured by the action of the compressed air issuing from the pipes at the sides. Flowing onto the next plate below, the pulp is treated again in the manner described and so on to the outlet of the box. The succession of plates offers a compact form of apparatus with large operating surface for dissolving the values during the passage of the pulp through the machine. (959,301, May 24, 1910.)

Cyaniding Under Pressure.—Facility and efficiency are claimed for a cyanide barrel recently patented by Benjamin O. Webb, of Los Angeles, Cal. The apparatus consists of an enclosed and revolvable tank or barrel which can be made air-tight. It contains a perforated floor which can be used as a filter medium, and has means provided for injecting air or water under pressure. The object of the invention is to provide in one apparatus means for agitating and aerating pulp under pressure, for filtering under pressure when solution of the ore values is complete, and for washing out the filtered pulp. In operation, the ore pulp is introduced through a manhole which is then tightly closed. Through suitable valves oxygen or oxidizing gases are then introduced under a pressure suited to the operation and the apparatus. The barrel is then revolved and the pulp agitated with the compressed oxidizing gases. Projecting knobs on the inside surface of the barrel aid in the complete agitation of the pulp. When this operation is complete the barrel is stopped in what may be called its upright position, viz., with the manhole up and the inside filter floor horizontal. The operation of filtering the pulp may now be effected by introducing more air and allowing the filtered solution to be drawn off through suitable stop cocks. The pulp may be washed if necessary, after which it is discharged by turning the barrel over and washing the sand out through the manhole by means of streams of water introduced through the same valves used to carry the oxidizing gases. The idea of cyaniding under pressure in the presence of oxidizing gases may add to the efficiency of the cyanide process, particularly with reference to refractory ores. (956,655, May 3, 1910.)

Accelerated Settlement of Slime.—The separation of the solid and liquid components of wet slime or ore pulp is effected by Horace G. Nichols, of Ymir, British Columbia, on the principle of acceleration of settlement due to continuous removal of fine slime as it settles. The accompanying Fig. 1 illustrates the device for this purpose. The slime pulp is introduced through pipe *I* to a settling chamber *B*, the walls of which converge downwardly to the opening *B'* directly above an endless traveling belt *D* moving in the direction shown by the arrows. As the slime settles it is continuously moved upward out of the tank discharged at the uppermost point to which the

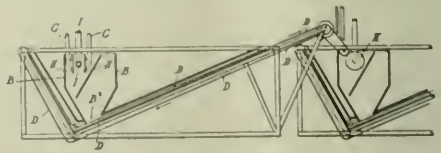


FIG. 1.—APPARATUS FOR THE ACCELERATED SETTLEMENT OF SLIME.

belt travels. The clear, or partly clarified, solution is withdrawn near the surface of the settling chamber *B* through two suction nozzles *H* connected with vacuum apparatus by pipes *G*. Provision is made for periodically cleaning the suction nozzles from accumulated slime by means of a reverse current of air or water. The apparatus may be set up in series of two or three, so that the discharged slime from the first may be subjected to a wash and subsequent dewatering in the second and third. This would be the procedure in case slime was being cyanided. The pregnant solution would be withdrawn in the first apparatus discharging slime carrying about 22.5 per cent moisture to the second, where it would be washed and, if neces

sary, again washed in a third tank. Clear liquid from these weakened cyanide solutions would be drawn off as described above. The apparatus has been in use at the Queen mine in British Columbia, where a plant handles 100 tons of dry slime in 24 hours. (958,272, May 17, 1910.)

Continuous Decantation.—In contrast to the many filtration schemes—vacuum and pressure—for treating sand and slime pulps, the continuous decantation process recently patented by W. L. HOLMS, Guadalajara, Mexico; L. M. GREEN, London, England, and J. A. PATTINSON, Zacatecas, Mexico, is an effort to get away from the necessity of vacuum appliances by making the decantation process mechanically rapid and efficient. The device consists of a series of tanks, say three, with internal mechanism for thickening the pulp. The thickened pulp from the first tank is drawn off at the bottom and pumped to the second and likewise to the third, from which it is discharged to waste. Passing from tank to tank in the opposite direction to the flow of pulp is a supply of wash water which is so introduced as to thoroughly wash the solids free from any dissolved values. The overflow from the third tank is returned as wash water to the second, and from the second to the first. In this way the flow of pulp is against the flow of wash water, the sand or slime being discharged from the bottom of the third tank, while the enriched solution overflows the rim of the first. Each tank contains suitable baffle partitions, agitators, and collectors, and pumps are provided for transferring pulp and solution from one tank to the other. The scheme is ingenious in its detail and may prove to have merit as a means of handling fine pulps. (957,775, May 10, 1910.)

Vacuum Filter.—A recent addition to devices for continuous vacuum filtration is that patented by MARTIN E. HILTNER, of Deadwood, S. D. The invention consists generally of an endless horizontal filter belt divided into sections, traveling around pulleys with means provided for alternately bringing each section into connection with vacuum and pressure pipes. A feed box suspended above the belt at one end of the machine supplies the pulp to be filtered. As each compartment of the belt receives its supply of pulp, a three-way valve connected with the same is actuated by a trip which establishes communication with the vacuum pipe. Each compartment then passes a second trip which actuates the valve so that communication is established with the pressure pipe, through which air or water may be forced to discharge the cake. The filtered solution is piped to suitable tanks prior to recovery of the contained values. (957,558, May 10, 1910.)

Copper.

Treating Impure Matte and Ores.—An impure copper matte, containing lead, is usually a product from most lead blast furnaces. This matte is sometimes partly roasted and then resmelted in the blast furnace, reducing some of the lead and concentrating the copper into a higher grade matte. This process may be repeated until as much of the lead is recovered as is deemed practicable, when the resulting matte is bessemerized. The object of a process recently patented by HERMAN MASCHMEYER, of Hoboken, Belgium, is to eliminate the lead and other impurities and to obtain comparatively pure copper without the repeated roasting and smelting before mentioned. Lead-copper ores may be treated in the same manner as prescribed for the matte. The process comprises two operations: First, a Bessemer roast with added silica; second, smelting in a reverberatory furnace. The first process may be carried out conveniently in Huntington-Heberlein pots. The roast will partially desulphurize the matte or ore and the silica will combine with the lead to form lead silicate. In the reverberatory furnace the lead silicate will go into the slag and remain unreduced. Reaction between copper sulphide of copper oxide will precipitate copper. Thus the separation is designed to be effected in one operation of smelting. Impurities, such as arsenic and antimony, would be partly volatilized and partly slagged off. A further step in the process consists in adding to the mass to be smelted an excess of copper oxide material, which has an oxidizing and refining effect. (957,231, May 10, 1910.)

Tin.

Reducing Cassiterite.—A novel process for the reduction of tin ore, differing radically from the usual process of smelting in reverberatory furnaces with culm, is that patented by ROBERT W. SCOTT, of London, England. The process consists in reducing cassiterite to tin by the action of producer gas. The application of the process is through the use of what may be termed tube mills in which the producer gas and cassiterite are brought in contact with the application of external heat. Reference to Fig. 2 will make the invention clear. An iron tube *a* is provided with screw caps and tubes at its ends. Stop cocks are provided to govern the flow of gas through the tubes. A furnace is constructed with two compartments, as shown, the first being the fire-box compartment and the second a drying and preheating compartment in which the waste heat from the first is utilized. These furnaces are provided with tubular supports and driving gears into which the iron tubes may be placed. The cassiterite is first dried and preheated in tubes in the

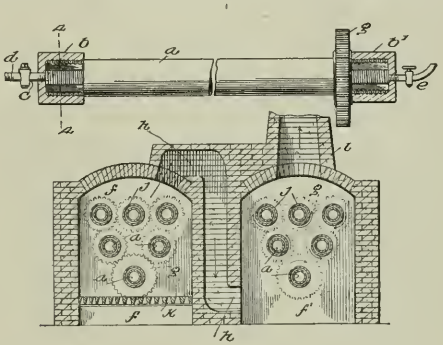


FIG. 2.—APPARATUS FOR REDUCTION OF CASSITERITE.

second compartment of the furnace, after which the tubes are withdrawn from their supports and transferred to the first compartment. Here connection is made between each tube, at the point *d*, and a source of producer gas. The gas is turned on until it will burn at the point *e*. The tubes are then set in rotation and kept revolving for the necessary length of time to complete the action of the gas on the cassiterite. The gas is kept under a sufficient heat pressure to keep the tubes constantly full of gas. Metallic tin is the product of the reaction. (957,111, May 3, 1910.)

Zinc.

Regenerative Furnace.—An improvement in furnaces for zinc distillation of the type generally known as "reverse regenerative," or more properly "reverse recuperative" type, is the subject of a patent granted to Dr. FRANZ MEYER, of Englewood, N. J. (now of Uerdingen, Germany). The principal feature of the invention consists in dividing the combustion chamber into two long flue-like compartments which communicate with each other at one end of the furnace. This permits of a complete reversal of the flow of air for combustion and of the products of combustion through the two compartments of the furnace proper, and through the recuperative flues. Fig. 3 represents a vertical cross-section of the furnace and Fig. 4 a horizontal longitudinal section taken just below the retorts, or on the floor of the combustion chambers. In Fig. 3 *AA'* and *BB'* are the recuperative flues through which air for combustion may be directed alternately by means of a reversing valve at the inlet, not shown; *ee'* are flues conducting the air from the recuperative flues to the combustion chambers

CC. Fig. 4. In one position of the reversing valve, which may be of the Forter type, air enters through the recuperative chambers *A.A'*, and by way of *e* into the combustion chamber *C*, where it mingles with gas issuing from the jets *n* in the first combustion chamber. It then passes through the communicating passage *p* into the second combustion chamber *C'*, where additional combustion of gas takes place. The products of combustion then pass through *e'* and through the recuperative chambers *BB'*. When this course has been pursued long enough to abstract the heat from the recuperative chambers *AA'* and to heat the chambers *BB'* by the waste gases, the flow of air is reversed. In this way the air for combustion is preheated alternately in the recuperative chambers. The gas for combustion is preheated by passage through the chambers *B' B*. (958,150, May 17, 1910.)

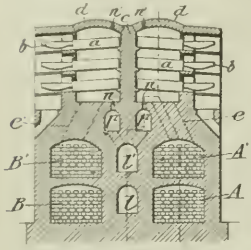


FIG. 3.—VERTICAL SECTION OF REGENERATIVE FURNACE.

Nickel.

Reduction of Nickel Hydrosilicate Ores.—Mr. Adolphe Chalas, of Philadelphia, Pa., patents a process of reducing silicious ores, such as garnierite, which contains nickel as a hydrosilicate together with a large proportion of magnesia. The other reducible metals contained in such ores are iron and a very small proportion of cobalt, manganese, and chromium, while the other refractory gangue materials, besides silica and magnesia, are very small proportions of alumina and lime. The hydrosilicate ores are characterized by their high combined water contents, averaging about 10 per cent by weight. They very easily absorb water, being of a clayey texture, and always retain a large amount of hydroscopic water, averaging over 20 per cent by weight. The present process essentially comprises three steps.

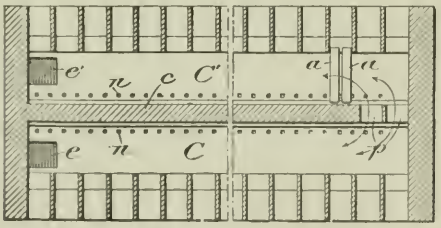


FIG. 4.—HORIZONTAL SECTION OF REGENERATIVE FURNACE.

The first preparatory step consists in briquetting a mixture of the nickel ores, serving as a binding material, with fine ores of high iron contents, such as magnetite sands and concentrates, flue dust, iron scale, purple ore, or the leached residues of roasted nickel copper pyrites.

In the second step metallurgical treatment begins. These briquettes are reduced in an ordinary blast furnace, with a sufficient quantity of lime to give a basic slag, producing a nickeliferous pig iron which is largely separated from the slag by gravitation.

Finally, the third stage of the process consists in chilling the molten slag by running it into water, crushing it, and passing it with water through a magnetic separator, by which practically every particle of nickel-iron alloy is recovered. The residual slag contains no nickel, owing to the lack of affinity of this metal for anhydrous silica, and to the fact that the iron, being more oxidizable than nickel, protects the latter from oxidation and scorification. (959,460, May 31, 1910.)

Furnaces.

Ore Roasting Furnace.—The reverberatory type of roasting furnace, both hand and mechanically rabbled, has been the subject of many patented improvements of various value. Letters patent have recently been granted to C. E. Mark, of Chicago, Ill., for an improved hearth claimed by the inventor to promote the roasting process and render it more efficient. As described by the inventor, the hearth is made of fire brick, or other suitable refractory material, perforated at regular intervals to permit a slight blast of air to be admitted from below while the charge is being rabbled either by hand or mechanical means. The idea is that by this intimate contact of air with the roasting ore, the oxidation of the latter will be more rapid and complete than in the usual form of furnace. The inventor's furnace may be oil-fired if desired, or gas may be used. Practical operation would determine whether such a furnace would make an excessive amount of flue dust, and whether the apertures in the hearth could easily be kept free from agglomerations of sintered ore. (952,680, March 22, 1910.)

Melting Furnace.—In the type of furnace known as the "pot furnace," used for melting metals and alloys, Alfred Fisher, of Chicago, Ill., has made certain improvements. In one instance the improvement consists in providing means for the even distribution of heat to the pot by means of a flame directed in an ascending spiral direction. Provision is made for an intimate mixture of air and gas, which passes around the furnace several times. The object of a second invention is to provide in a practical manner for the operation of a plurality of melting furnaces, so that the heat can be regulated in each and all of the furnaces independently of the operation or non-operation of any one of them. Provision is also made for the ready removal from the bottom of the furnace of a broken crucible or molten metal resulting from the breaking of a crucible. (957,873 and 958,384, May 17, 1910.)

Dross Furnace.—For the purpose of recovering metal from dross, George L. Walker, of Pueblo, Col., has devised a furnace in which the dross receptacle consists of a conical caldron designed to be rotated on its ends as trunnions. The caldron is contained in any suitable form of coal or gas-fired furnace, and is thus kept within the heated interior without being exposed to the open air. A conical stopper holds the dross in place in the caldron, which is revolved so as to facilitate the settlement of globules of metal from the dross. As the metal gathers it runs out of the lower end of the caldron. In this form of dross furnace, oxidation by the air is reduced to a minimum and the sweating effect is magnified. (949,311, Feb. 15, 1910.)

Gas-Cleaning Apparatus.

Smoke and Gas Condenser.—An improved form of condenser for use in connection with smoke, gases and vapors, which the inventor states has given satisfaction in practice, is the basis of a patent of Samuel G. Merrick, of Philadelphia, Pa. The apparatus consists of a casing of metal, or other suitable material, divided by a partition into a receiving and a delivery chamber having respectively an inlet and outlet for gas. A shaft is mounted for rotation in this casing, and carries a tubular member having a spiral partition running through it. In Fig. 5 the description thus far given will be apparent. The receiving end of the revolving tubular part has the shape of a frustum of a cone and is adapted to receive a large volume of gas. As the tube rotates and the gas is advanced toward the delivery end, it is compressed into smaller volume by reason of the constricted size of the tube at that end. This is an important feature of the invention in that it continues the compressing and purifying which would otherwise cease at the end of the frustum. Baffle plate 13 is arranged as shown for the purpose of breaking up the liquid in the tank and effecting a thorough mixture with the gas. In operation the casing is partly filled with water as shown. Smoke, gas and vapor enter

through the pipe 3 and are conveyed through the revolving tubular member. During its confined passage the gas is subjected to a thorough washing and condensation and issues free

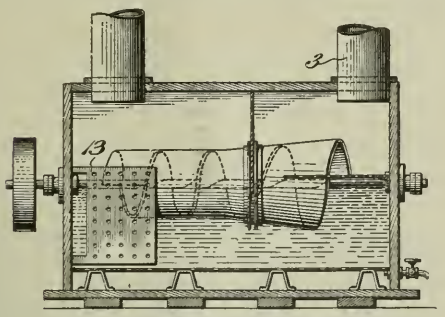


FIG. 5.—SMOKE AND GAS CONDENSER.

from solid particles and soluble constituents. (946,146, Jan. 11, 1910.)

Cleaning Blast-Furnace Gas.—Gravity and centrifugal action are the forces utilized in a device for cleaning gases from iron blast furnaces, according to a patent granted to JOSEPH MORGAN, of Westmore Borough, Pa. The device might also be applied to the removal of valuable flue dust from the gases from other metallurgical furnaces. Provision is also made for reheating the gas before use, and for washing it if necessary. The details of the cleaner may be seen in Fig. 6. The gas is first drawn from the top of a furnace down to a settling chamber, where, by reason of the expansion and decreased velocity of the gas, the larger dust particles separate out by gravity and may be withdrawn at the bottom of the chamber. From here the gas is led to the cleaning apparatus illustrated. Entering at 8 the gas passes through a spiral passageway formed by a helical partition. During this passage the particles of dust will fall into traps 15, where they will remain until withdrawn through suitable gates or valves. During this passage, also, the gas may be sprayed with water from the pipe 22, both water and solid particles then settling into the traps 15. Having passed the length of the spiral passageway, the gas now returns through a central pipe as shown by the arrow. Here the gas is reheated prior to use. (948,062, Feb. 1, 1910.)

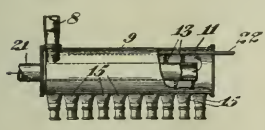


FIG. 6.—GAS CLEANER.

Miscellaneous Metallurgical Processes.

Basic Lead Sulphate Pigment.—The manufacture of basic lead sulphate pigment by volatilizing and oxidizing galena ores has usually resulted in a yield of about 80 per cent of a product which has not always had a good color. In addition to these drawbacks a certain amount of lead is usually reduced, thus making the process inefficient in the points of yield and color of product. J. B. HANNAY, of Cove Castle, Scotland, and S. B. WILSON, of Westminster, London, England, have been granted patent on a process for converting lead sulphide ores into basic lead sulphate, in which they claim a high yield of good white pigment of constant composition, without attendant reduction of pig lead. In a blast furnace supplied with a hot-air blast a column of coke is heated to about 1550° C. with a temperature at the top of the furnace of about 1250° C. Galena is fed onto this hot mass, with the result that globules of galena become surrounded with an atmosphere of the same composition. The furnace being thus filled with vapors of lead sulphide and carbonic oxide, and the liquid galena being surrounded by gaseous lead sulphide, there is no opportunity for reduction of lead. The mixed vapors pass from the furnace to a combustion chamber where a large excess of cold air is admitted. Combustion of the lead sulphide and carbonic oxide ensues and cooling is effected by the large excess of air. The lead fume now passes to a water-cooled condenser where it is collected in conical hoppers. The inventors have provided means for feeding the furnace with hot dry coke and dry galena in such a manner that the balance of flame temperature in the apparatus is not disturbed by feeding. They state further that they have obtained 111 parts of basic sulphate of a composition between $3PbSO_4 \cdot PbO$ and $2PbSO_4 \cdot PbO$ from 100 parts of galena containing about 5 per cent of impurities. The product is ready for use and requires no subsequent treatment of any kind. A furnace 36 in. internal diameter will produce 4 tons of lead sulphate in 24 hours. (955,396, April 19, 1910.)

Briquetting Fine Ore.—Flue dust, fine ore, waste iron, etc., are prepared for smelting by teatment in one of the methods recently patented by MAX GLASS, of Vienna, Austria. In the first process, the fine material is mixed with a solution of calcium chloride and water glass, when the following chemical reaction occurs:

$$Na_2SiO_3 + CaCl_2 = CaSiO_3 + 2NaCl$$

The silicate acts as a binding agent. Borax may be added to briquettes of fine iron metal to prevent rusting. The reagents are used in the following strengths and proportions: 55g $CaCl_2$ to one liter water, 175g Na_2SiO_3 to one liter water. The borax solution will contain 40g to 50g per liter. The second process of the same inventor contemplates the use of calcium acetate and aluminium sulphate, with an admixture of quicklime and borax if desired. The reactions are as follows:

$$3Ca(CH_3COO)_2 + Al_2(SO_4)_3 = 3CaSO_4 + Al_2(CH_3COO)_6$$

If lime is present, considerable heat is formed by its combination with the water, and under the action of this heat aluminium acetate is further converted into an insoluble basic salt which has a binding action. The reagents used in this process are: 28g CaO mixed with 60g CH_3COOH to slightly acid reaction, and diluted to one liter with water. 60g $Al_2(SO_4)_3$ are dissolved in one liter water, to which are added 50g to 60g powdered lime. Borax solution may be added as indicated above. One liter of the binding agent is mixed with every 100 kg of the material to be briquetted. The process produces briquettes which need no further heating. (958,700-1, May 17, 1910.)

Electric Furnaces.

Purification of Bauxite.—Mr. Lewis E. Saunders, of the Niagara Falls works of the Norton Company, has devised a process in which he makes use of the strong reducing properties of aluminium carbide to secure a nearly complete elimination of the metals or impurities associated with the alumina, and he also makes use of the known fact that silicon is capable of preventing the absorption of large proportions of carbon by such metals as iron and titanium. In his process of purifying silicious bauxite or clay he adds to it a proportion of carbon, largely in excess of that required to reduce the iron, silicon, and titanium therein, and fuses the mass in an electric furnace, whereby a considerable portion or even all of the alumina is converted into aluminium carbide. This carbide is capable of mixing with molten alumina or of dissolving or diffusing therein, and when so dissolved or diffused, or when in a molten condition, is a most energetic reducing agent, by the action of which, supplementing that of the carbon, the oxides of the associated impurities are reduced with substantial completeness, at the same time the reduced metal or alloy is by its content of silicon prevented from absorbing or combining with such proportion of carbon as would bring about an amount of resolution of diffusion capable of contaminating the bath to a considerable degree. The reduced metals are allowed to settle from the molten bath of aluminium carbide, or alumina and aluminium carbide, and are removed therefrom by tapping. The molten bath, substantially free from metals other than aluminium, is then oxidized, for instance, by blowing air through

the molten mass in the furnace or after tapping by showering the molten mass through air. This reaction is strongly exothermic and the fluidity of the bath is increased by the heat set free, whereby the operation of blowing and any pouring or tapping operation is rendered easier. The result is that a homogeneous mass of commercially pure alumina is obtained. (960,712, June 7, 1910.)

Reduction of Aluminium Ore.—A process for the reduction of aluminium ore is described by Mr. Frank J. Tone, of the Carborundum Company, of Niagara Falls. It consists of two steps. In the first step the aluminium ore is mixed with the proper amount of carbon and reduced to aluminium carbide. In the second step the aluminium carbide is used as a reducing agent in place of carbon for the reduction of ores of various elements to alloys of aluminium and these elements or for the reduction of aluminium ores to elemental aluminium as the case may be. For example, in the manufacture of a silicon-aluminium alloy corresponding to the formula $8Al.3Si$, aluminium carbide and silica are taken in the proportions expressed by the chemical equation:

$$2Al_4C_3 + 3SiO_2 = 8Al.3Si + 6CO \qquad (1)$$

and subjected to heat in an electric furnace, obtaining the silicon-aluminium alloy as the final product. If the element aluminium is to be produced as the final product a mixture of aluminium carbide and aluminium oxide is made in accordance with the following equation:

$$Al_4C_3 + Al_2O_3 = 6Al + 3CO. \qquad (2)$$

The advantage of dividing the process into two parts is that the reduction of aluminium ores to aluminium carbide can be accomplished with much greater efficiency than the direct reduction of the ore to the element aluminium. The reason for this is that furnace processes can be adopted for the production of the carbide which are highly efficient and entail little or no loss by volatilization. Of the six parts of aluminium produced in equation (2), four parts are the result of the highly efficient first reduction to the carbide and two parts are the result of the less efficient second step of the process. The amount of CO gas evolved per unit of aluminium produced in equation (2) is also much less than in the reaction in which the oxide is directly reduced by carbon. (961,913, June 21, 1910.)

Graphitizing Carbon.—A process of Mr. Frank J. Tone of the Carborundum Company, relates to the "electric heating of carbon, for the purpose of baking, purifying, polymerizing, graphitizing, or otherwise changing its character by heat treatment." The essential feature is the use of an arc furnace instead of a resistance furnace. In Fig. 1 the charge of carbon to be converted into graphite is indicated by 3, while 4 and 5 are the two electrodes. The upper electrode is surrounded by a shield 6 of carbon, which is insulated from the electrode by a refractory insulating layer 8. This shield 6 prevents contact between the charge and the electrode so as to avoid shunting of the current around the arc. An increase of efficiency over resistance heating is claimed, due to the greater amount of energy which can be generated and supplied in an arc furnace to a given volume of charge in a given time. Not only is the carbon converted in graphite, but any impurities, such as silica, alumina, iron, calcium, etc., are volatilized. (961,912, June 21, 1910.)

Electrode Connection.—Mr. Charles A. Keller, of Paris, France, has devised the following method of making connection between a carbon electrode and the conductor. The metallic conductor comprises a cast iron or steel sheath enclosing a hollow copper core through which cooling water circulates. A recess is formed in the head of the carbon electrode and in this recess the end of the conductor is inserted, the conductor being smaller than the recess in the electrode. Molten copper is then poured into the recess around the end of the conductor, which is thus physically connected in a permanent manner to the electrode. Recesses are formed in that portion of the conductor which lies within the carbon electrode and these recesses are entered and partly filled by the molten copper, "an improved contact being thus obtained and air gaps being provided which tend to prevent the formation of induced currents." (961,139, June 14, 1910.)

Alloy for Resistor Wires.—Mr. Wilbur B. Driver, of East Orange, N. J., patents the addition of aluminium (1 to 9 per cent) to alloys of at least two other metals for the purpose of producing wire for resistor purposes; for instance, to copper-nickel alloys, copper-manganese alloys, copper-nickel-manganese alloys, and iron-nickel alloys. An alloy containing copper and manganese and 5 per cent of aluminium and an alloy containing copper, nickel, and manganese and 5 per cent of aluminium, are especially mentioned in the claims. (961,217, June 14, 1910.)

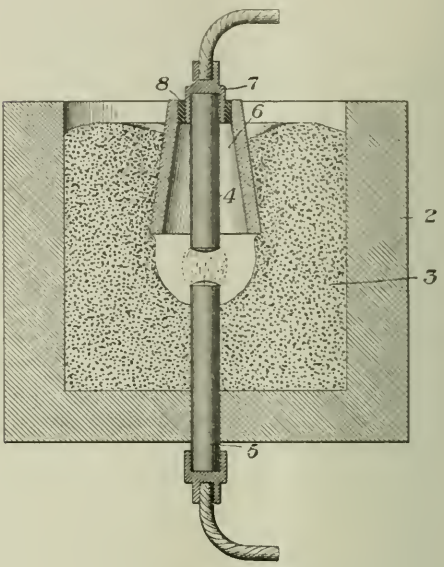


Fig. 7.—Arc Furnace for Graphite.

Electrolytic Processes.

Metallic Sodium.—Mr. Robert J. McKnitt, of Niagara Falls, proposes the following method of producing sodium from sodium chloride (assigned to Roessler & Hasslacher Chemical Company). The sodium in sodium chloride is first alloyed with lead or tin, by some such method as electrolyzing fused sodium chloride with a fused lead cathode, as in the Acker process. The lead-sodium alloy is then heated in a partial-vacuum chamber under reduced pressure. The sodium distills off and is condensed in a separate condenser. The process may be made continuous by conveying the molten alloy from the electrolytic furnace to the receiver, distilling off the sodium and returning the remaining lead to the furnace. In this case but little heat will have to be supplied to the receiver, as the temperature of the alloy in the furnace is usually about 800° C., while sodium in vacuo boils off readily below 800° C., it being necessary simply to maintain the temperature while reducing the pressure. Of the claims which are broad and do not refer to sodium, the first one may be quoted. It relates to "the method of reducing a metal from its compounds which consists in dissolving, alloying or chemically combining said metal with another metal or mixture of metals less volatile than the metal desired, submitting the resultant solution, alloy or compound to a pressure lower than that due to the atmosphere and to a temperature suitable for the distillation of the metal desired." (959,785, May 31, 1910.)

Cathode for Diaphragm Cells.—A patent of Mr. ELMER A. SPERRY (assigned to Hooker Electrochemical Company) relates to a metallic cathode plate for use with such electrolytic cells in which a diaphragm is placed in contact with one face of the cathode, as in the Townsend cell. In such cells, and particularly in case a high current density is employed at the cathode, it is desirable to secure a substantially even distribution of the current over the face of the cathode plate, as otherwise irregularities of action may ensue, resulting in a decreased efficiency of operation. The perforations of the cathode should preferably aggregate 50 per cent or more of its exposed area. These perforations have heretofore usually comprised uniformly distributed circular apertures. While the electrical conductivity of a plate of this character is substantially uniform in all directions, it has been found that such apertures tend to clog or fill in practice when in contact with impure liquids, the deposit accumulating in the lower portion of each aperture to a height dependent upon the thickness of the plate; after a certain amount of material has collected, any further deposit is washed out and eliminated by the escaping gases and percolating liquid. The difficulty may be largely overcome and other important advantages secured by elongating the apertures in a vertical direction. In this case the deposit, being dependent upon the thickness of the plate, does not rise higher than in a circular orifice, and thus clogs or fills a much less proportion of the area of the perforations. These elongated perforations or slots possess a further advantage which is independent of their vertical or other position, in that for a given percentage area of aperture in the plate they afford, as compared with circular apertures, an increased active cathode area; this results from the fact that the walls of the perforations are cathodically active to the extent that they are wet with the brine or other solution undergoing electrolysis, or are in contact with the saturated diaphragm; and a far greater length of wall is provided by the use of the elongated apertures. It is desirable that the cathode plate should be as thin as is consistent with strength and the considerations of electrical conductivity. In order that it may be made thin and at the same time may effect a substantially even distribution of the current it is essential that the current should be conducted from the cathode at those edges which are transverse to the slots or elongated perforations. In this way a direct and short path is afforded for the current from any portion of the plate to the conductors. (961,549, June 14, 1910.)

Gold Refining.—An important modification of the well-known Wohlwill gold refining process (in use in the United States mints and elsewhere) is patented by Dr. HEINRICH WOHLWILL, of Hamburg, Germany. The essential feature is that the development of chlorine at the surface of the anode can be entirely prevented by employing a non-symmetrical alternating current (a superposition of direct and alternating current) instead of the usual pure direct current. The process is therefore made applicable to the refining of gold carrying a large percentage of silver. Other advantages are a materially shortening of time and reduction of expense. The process was already described in some detail in our February issue, page 82. (961,294, June 21, 1910.)

Miscellaneous.

Tungsten Powder.—Metallic tungsten in powder form is obtained by reduction of tungstic oxide by the following method of Dr. ELIHU THOMSON (assigned to General Electric Company). The tungstic oxide powder is fed uniformly from a hopper into an inclined heated tube in which it is moved forward and downward by a revolving Archimedean screw. Air is kept out of this tube and hydrogen gas is passed through it in a direction opposite to that of the movement of the powder. At a temperature between 600° C. and 800° C. reduction takes place according to the equation $WO_3 + 6H = W + 3H_2O$. Only a small part of the hydrogen in the tube is used up in this reaction; the hydrogen-steam mixture having been withdrawn is therefore freed from the water vapor first by condensation in a water-cooled condenser and then by contact with metallic sodium, hydrogen being regenerated in the latter reaction from the steam. The unused hydrogen, together with the regenerated hydrogen, is carried back into the reduction tube. (960,441, June 7, 1910.)

Treatment of Wool.—Messrs. R. VANDATTE and B. LAGYE, of Antwerp, Belgium, patent details of an apparatus for removing the yolk and the grease from wool, the wool being subjected to the action of an electric current while it is moved about in a vat filled with yolk water. This process differs from similar ones by the electric current being conducted to the wool under treatment through rakes which, while their supports convey the current to the wool, also insure the carrying of the wool along the vat and its subdivision or opening out therein. (961,468, 1910.)

Ageing Liquors.—Mechanical details of an apparatus in which a liquor is simultaneously subjected to the action of gases and to the action of an electric current for the purpose of ageing, are the subject of a patent of Mr. J. SEITZ, of Chicago. (961,167, June 14, 1910.)

Recent Progress in Wood Distillation.

BY M. KLAR.

In the March issue of this journal (Vol. VIII, p. 155) Mr. Charles L. Campbell discusses the wood-distilling industry in the United States, with reference to present methods, market prices of the products, prospects of the manufacture of acetic acid and methyl alcohol from the crude products of wood distillation, and finally gives a review of improvements which have been devised and introduced by me and my firm, F. H. Meyer, Hannover-Hainholz, Germany, during the last years in wood distillation in various countries, including the United States and Canada.

The information given there was essentially based on my book, Technologie der Holzverkohlung (2d edition, 1910; published by Julius Springer, Berlin) and on prospectus 290 *a* to 290 *c*, 1909, of the firm of F. H. Meyer. But there are some inaccuracies on practical details, which I wish to correct.

Thus my tar separator (American patent 868,347) has not yet been used in practice for kiln plants, because the enormous quantities of gas in this process would necessitate the design of apparatus of enormous dimensions and therefore very high first cost.

To treat the crude wood vinegar of the kiln plant for the production of acetate of lime and wood alcohol, I employ a continuous distillation process, which differs, however, from those used heretofore in the United States in so far as the latent heat contained in the first distillation is repeatedly utilized. The result is that the cost of distillation with the methods usually employed at present, is reduced by about 70 per cent.

The statement of Mr. Campbell that my continuous drying apparatus for gray acetate of lime is operated with superheated steam, is also incorrect. In my evaporation and drying process (which is the first entirely continuous and mechanical process in practical use) nothing but exhaust steam is employed, so that the production of the gray acetate of lime from its solution costs practically nothing.

In the following I intend to supplement Mr. Campbell's article by showing in detail the relative advantages and disadvantages of the processes hitherto used in the United States, and my new processes.

Distilling of Wood.

Three methods have hitherto been practised in the United States, employing kilns, retorts or ovens respectively.

(a) Kilns have the disadvantage of insufficient yield.

(b) With retorts, that is, small horizontal iron retorts, the yields are poor, the fuel consumption high, and wages high.

(c) In ovens, that is, in large horizontal iron retorts,

charged with wood with the aid of carriages, good yields are produced, but the costs of repairs are very high, on account of the unfavorable profile of retort and carriage, while the fuel consumption is also high.

On the other hand, in the new process of F. H. Meyer, large horizontal retorts of 8 to 9 cords capacity are only used, charged by carriages. However, the retorts and the carriage have a profile which offers a far larger resistance. For heating wood, producer gas is used, whereby not only the whole length of the retorts is uniformly heated, but a certain amount of acetate of lime and wood naphtha is obtained as by-products from the fuel (wood). These by-products cover the cost of the fuel. Coal producer gas, furnace gas, natural gas can be used instead of wood producer gas. The preheating of the gas as well as of the air used for burning, and of the iron carriages with the charge of wood, entails no expense with the F. H. Meyer retort

This system has the great advantage that repairs of carriages and retorts are very rare, even after many years of working. By the regular heating with gas, a considerable saving of fuel, an easy regulation of the fire, and thorough regularity in the heating of the retort are obtained. The length of life of the retort is thereby prolonged; besides the yields are increased. Through the production of byproducts out of the fuel, in case wood producer gas is used, the fuel really costs nothing, since these byproducts cover the expenses for the fuel. Besides the saving of fuel, the retorts are preserved, the wood charring process is shortened, and the production is increased.

Distilling of Crude Liquor.

In the process hitherto used in the United States, the cooled crude liquor was distilled in simple copper stills, connected with a condenser. This distillation has in view the removal of the tar, dissolved in the crude liquor, in order to produce acetate of lime of 80 per cent.

The disadvantage of this usual method is that considerable quantities of steam are required for the distillation, amounting almost to 50 per cent. of the whole quantity of steam, which is required in wood distillation.

On the other hand the process of F. H. Meyer employs two methods for separating the tar from the crude liquor.

(a) Between the retort and the condenser the tar separator (American Patent No. 868,347) is inserted, whereby the whole quantity of tar, which is contained in the wood distillation products, is removed before the condensation of the crude liquor. This one can be neutralized on leaving the retorts without a special distillation.

(b) The crude liquor is distilled by connecting several distilling apparatus with each other, and these are kept under different pressures. The latent heat of the vapors formed in the first apparatus distills the crude liquor in the second apparatus and the vapors of the second apparatus distill the crude liquor of the third apparatus.

The advantages of these methods may be summed up as follows:

(a) By the apparatus (a) 868,347, which is already in operation in the United States, the whole consumption of steam which heretofore was required for the crude liquor distilling, is rendered unnecessary. The disagreeable cleaning of the primary stills and the retort condensers is also done away with. Further, there is a considerable saving of fuel, wages, apparatus and space

(b) The apparatus (b) can be worked by waste steam, and in this case do not require any fuel at all, if sufficient waste steam is to be had. In case no waste steam is available, the apparatus permits a saving of fuel of 63 per cent., compared with the consumption in former American practice

Manufacture of Crude Wood Alcohol from Neutralized and Distilled Crude Liquor.

In the process used so far in the United States, this was done by distilling the neutralized liquor in stills until the distillate became free from alcohol. In order to accomplish this, 30 per cent. had to be distilled off, which contained about 10 per cent. of wood alcohol. After a second or a third rectification of this weak wood alcohol, the commercial product was obtained.

The disadvantage of this method is that the three successive periodical operations require a heating to the boiling point each time; they require a large number of apparatus, much supervision and attendance, and involve a very great consumption of steam, while the losses are considerable.

On the other hand, in the new process of Mr. F. H. Meyer the solution of acetate of lime containing wood alcohol, which is obtained by neutralizing the distilled crude liquor, is separated in a single continuous operation and with only one apparatus into a commercial crude wood alcohol and a solution of acetate of lime free from wood alcohol, while without expense the acetate solution containing wood alcohol is preheated before entering the apparatus.

The advantages of this new method are considerable saving of steam, considerable saving of space, considerable saving of wages and simplification of the working, and practically no losses.

Evaporation of Acetate of Lime Solution.

This was hitherto done in open pans with double bottom, heated until a crystalline paste was formed. In this process crusts form on the heating surfaces.

The disadvantage of this method is that the forming of crusts results in an exceedingly high consumption of steam which is not at all in proportion with the evaporated quantity of water. Further, considerable manual labor is necessary in taking out the crystallized paste and removing the crusts. For this reason these open evaporating pans are a very expensive and very troublesome apparatus in a wood distillation plant.

On the other hand, in the new process of Mr. F. H Meyer the evaporation of the acetate of lime solution does not take place in open pans, but continuously in closed vessels, and only to such an extent as not to allow any crystals to separate. Thereby the vapors of one evaporator heat the solution in the other one, so that in connecting three evaporators, one needs to supply steam only to the first one. The further evaporation of the concentrated solution of acetate of lime, with formation of crystals, is carried out in revolving apparatus; the acetate is dried on its surface and is continuously removed by mechanical scrapers and transferred continuously to the final dryer.

The advantage of this new process is that in using several evaporators which work in combination together, one needs only about 37 per cent of the steam hitherto required for the concentration, as the latent heat is utilized three times. The apparatus works continuously, needs but little space, and not much of attendance. The heating surfaces of the revolving apparatus are constantly kept free from crusts by the scrapers. The saving of steam and wages is considerable, and the troubles of taking out the crystals and scraping off the crusts by hand are entirely done away with.

Drying of Acetate of Lime.

This was carried out heretofore on an open drying floor, heated by direct fire or fire gases with manual labor. The chief disadvantages are that a great many workmen were required, who usually do not like the work, so that there is the constant trouble of finding new labor. As a matter of fact, the unpleasant smell is very annoying and the workrooms are covered with dust.

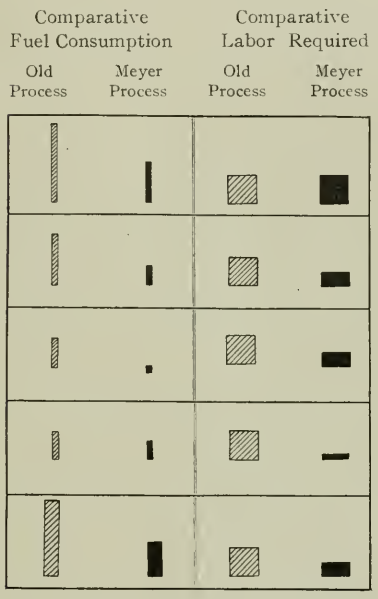
On the other hand, in the new process of Mr. F. H. Meyer this final drying operation is carried out in a continuously working apparatus, the dried acetate of lime being directly put into sacks without any intervention. The evaporator, the revolving dryer, and the final dryer work together in an abso-

lutely continuous matter; the acetate of lime solution enters the evaporator and the dried acetate leaves the final dryer, there being no manual labor.

This continuously working apparatus necessitates even for the largest plants only one man per apparatus. The disagreeable work on the open dryer is avoided; the work is a pleasant one and the required space is greatly reduced. The acetate of lime enters the apparatus on one side and leaves the same on the other side as acetate ready for the market, without any manual labor and using waste steam only for heating.

The following table gives a comparison of the consumption of fuel and the number of workmen needed in the older process used in the United States and in the new process of Mr. F. H. Meyer.

The table shows that a wood distilling plant operated by the system of Mr. F. H. Meyer is altogether different from


Comparative Fuel Consumption
Comparative Labor Required
Old Process
Meyer Process
Old Process
Meyer Process
Nature of Process
1. Distillation of wood in large retorts (ovens).
2. Distillation of crude liquor.
3. Manufacture of crude wood alcohol from acetate of lime solution.
4. Evaporation and drying of acetate of lime solution.
Summary giving the comparative totals.

former plants. This is especially true if wood-gas producers are employed; they originated in Hungary, but have been considerably improved. These apparatus produce the heating gases for the retorts and the boiler out of the cheapest waste wood. The producers are continuously charged with wood. While the wood falls down, the gas is gradually removed and the whole carbon, which the wood contained, is changed into producer gas, whereby the wood is carbonized at the same time by the hot generator gases.

The products escaping from the gas producer consist of crude liquor vapors, wood gas, and wood producer gas. These products pass through a condenser in which the crude liquor is condensed, which is worked up to acetate of lime and wood alcohol. The remaining gases are utilized, as mentioned above, after being preheated without cost, to heat the retorts and the boiler. We have, therefore, simply an apparatus heated by gas with all its advantages. The quantity of gas obtained by this operation is large enough to work up the crude liquor out of the producer itself as well as the crude liquor coming out of the ovens or retorts.

With these apparatus the fuel costs practically nothing, if the price of the wood is not too high. They are specially useful in using cheap kinds of wood, which are not fit for carbonization. This should prove a great advantage in many places.

To sum up the characteristic features of the new Meyer-Klar wood distillation process, the following types of apparatus are used and have proven successful in commercial practice:

(1) Retorts of 8 cords capacity, which are charged and emptied by means of carriages and are heated by gas. The method of firing has been carefully worked out in practice, wood-producer gas or any other gas being employed. The whole length of the retort is heated perfectly uniformly. On account of the special profile of the retort the troublesome repairs, resulting in interruption of operation, are reduced to a minimum. The cost of operation is reduced, the life and the yield are increased, and the fuel consumption is very materially decreased.

(2) Use of preheaters for the wood which is to be subjected to distillation. In these apparatus the wood is preheated and dried without expense. Other advantages of this simple and cheap method are a smaller consumption of fuel in the distillation which follows, and increase of output of acetate of lime, wood alcohol, and charcoal.

(3) The use of special devices in the chambers for cooling the charcoal, whereby the time of cooling is reduced to 24 hours, while only one cooling chamber is needed instead of two or three as heretofore.

(4) The use of preheaters, the operation of which entails no expense, for preheating the gases and air before entering into the fireplace, whereby considerable quantities of fuel are saved.

(5) The use of wood-gas producers, operated with waste wood. They furnish not only the gases required for heating the retorts and boilers, but it is also possible to recover the acetate of lime and alcohol in sufficient quantities to pay for the wood consumed in the producer. Hence, there is practically no expense for the heating—that is, in this case for the fuel for the whole installation.

(6) Washing of the wood gases and the heating gases before combustion, whereby the output is increased.

(7) Separation of the tar before the condensation of the

wood vinegar vapors by means of Klar's tar separator. This renders unnecessary the usual distillation of the wood vinegar in primary stills, so that the main quantity of the heating vapors used in a distillation plant is saved.

(8) Use of wood vinegar distillation apparatus, repeatedly utilizing the latent heat, instead of primary stills, in cases in which tar separators cannot be employed, as, for instance, in kiln plants. These apparatus operate absolutely continuously and, compared with primary stills, they permit a saving of 70 per cent of steam, if steam under pressure is used, or of 100 per cent if exhaust steam is used.

(9) Use of continuously operated rectification apparatus for the production of crude wood naphtha from the calcium acetate solution. In a single operation (instead of three or four, as employed heretofore) commercial crude wood naphtha is produced, of about 90 per cent, mixible with water.

(10) Use of apparatus operated perfectly continuously and mechanically for producing dry acetate of lime from its solution, with the use of only steam or exhaust steam for heating. The solution of this problem has been tried repeatedly in this country, but always without the desired success in spite of considerable expense of money. After many years of experimental work I and the firm of F. H. Meyer have solved this problem, and these purely mechanical lime acetate dryers have now been in successful operation for a number of years in quite a number of works. Those who know the difficulties and complications of present methods of drying lime acetate solutions and the high cost of fuel and labor will appreciate the advantages of our apparatus, which is supplied at one end with dilute acetate solution and which yields at the other end commercial gray acetate of lime directly into the bags, while only a relatively small percentage of the heat employed in usual processes is needed in the new process, and one or two men in a shift can take care of the drying of more than 30,000 lb. per day.

All the processes and special apparatus noticed above have been developed by us in almost 15 years of work in wood distillation. All of them are in practical operation in numerous wood distillation plants in all countries, erected by the author and the firm of F. H. Meyer.

In his article Mr. Campbell speaks of the good prospects of the manufacture of acetic acid and methyl alcohol from the raw products of wood distillation. On the basis of my personal experience in wood distillation plants in all countries (United States, Canada, Germany, Hungaria, France, Belgium, Sweden and Russia), I confess that I cannot share this optimism. In all these countries experience has shown that refining plants for the production of methyl acetate, acetone, methyl alcohol, allyl alcohol from crude wood naphtha are profitable only on a very large scale.

There are very few wood distillation plants with an output large enough to operate a refining plant with profit.

In the United States, in Canada, in Hungary, briefly, in all countries with sufficiently large output, there are central refineries. These can be operated with much greater profit, as they are better able to utilize the intermediate products; further, the material losses are less; they can adapt the product better to the needs of the consumers with respect to quality; the expense of labor and fuel is less, and there are general commercial reasons with respect to freight expenses and market situation.

Moreover, at present, only continuous apparatus are used for the refining of crude wood naphtha. But these can give the best results only when operated on a large scale. Further, a crude wood naphtha refinery requires continuous control by a scientifically trained chemist, who can expect a salary which would be too much for a smaller refinery.

All these points seem to indicate that the refining of crude wood naphtha is not always certain to produce a reasonable profit. On the contrary, on the basis of practical experience, it may be said that only under specially favorable conditions refineries can operate with a reasonable profit on a smaller scale.

Similar remarks may be made on acetic acid plants of smaller size. In this case, concentrated sulphuric acid is needed and its relatively low market value cannot stand considerable freight expenses.

Further, the present American producers of acetic acid, all of which use apparatus of the firm of F. H. Meyer, are also producers of sulphuric acid and this point should not be overlooked in considering the possibility of competition. Acetic acid manufacture also requires the employment of a scientifically trained chemist.

More favorable are the conditions for the manufacture of acetone. In the United States and Canada about 40,000 lb. of acetate of lime are treated per day by the process of the author for the manufacture of acetone and other raw materials are not required.

The above notes are intended to sketch briefly the features of wood distillation plants of modern type. In conclusion, it may be worth while to emphasize again that in case of a projected new plant or the rebuilding of an existing plant, it is necessary to first make sure that a reasonable profit can be obtained, and that the projected methods and apparatus really represent the latest developments of practice.

Convention of American Foundrymen's Association and Exhibition of Foundry Supplies.

The fifteenth annual convention of the American Foundrymen's Association and associated societies was held in Detroit in the week beginning June 6. What has been said of every convention of this association during the last years, was again true, namely, that the attendance was better than ever before and so was the exhibition of foundry supplies and machinery.

Mr. Arthur T. Waterfall, in his presidential address, referred to the permanent mold and to the electric furnace as the most important problems of to-day before the foundrymen.

The secretary-treasurer, Dr. Richard Moldenke, emphasized that liberal mindedness and broad ideas of the individual foundrymen of this country had procured for the association its present high standing. He gave an account of two recent trips to Europe, and pointed out two noteworthy differences in foundry practice in this country and Germany. In Germany they make the castings entirely for the work intended, no matter whether the machine shop kicks about hard castings. If they want wear, they make the castings accordingly and do not allow any "excessive" machine shop costs to stand in the way. Dr. Moldenke thinks a little of this method would do good in this country, where we are often too much after soft castings.

The second point of difference is that the better brands of pig iron used in Germany are all of the low-silicon varieties, as they still believe in the necessity of importing metal that has the name for making strong castings. When the Germans will recognize that other pig irons will do the same, American and German practice will get closer together.

Two papers on oxy-acetylene welding were presented, one by Mr. Steelman Stephenson, of Detroit, on acetylene-oxygen repairs in the foundry; another, an illustrated lecture by Mr. Henry Cave on the methods of the Autogeneous Welding Equipment Company, of Springfield, Mass., who use the process of the Davis-Bournonville Company.

A shockless jarring machine was the subject of a paper by Mr. Wilfred Lewis, of Philadelphia.

The premium system of compensation in the foundry was heartily recommended in a paper by Mr. W. J. Power.

Rejected steel castings in steel foundries were the subject of a paper by Mr. Samuel D. I. Emerson, who has reached the conclusion that in steel foundries rejections are principally due to accidents or to inefficient methods in core or molding

departments. Ninety per cent of rejections are mechanically avoidable, without reference to metallurgical difficulties. With adequate record, systematic and continuous attention rejections can be reduced to an average of 3 per cent. In order to secure the continuous attention required, a separate organization with pay largely dependent on results is most effective.

Mr. P. KREUZPOINTNER presented the report of the committee on industrial education. He emphasized the necessity of recognizing the social side of the problem of industrial education by the business men of this country. If our poorer European friends and competitors do not hesitate to expend enormous sums for the extension of their splendid system of industrial schools, then our industries may likewise consider such sacrifices as a profitable investment. Since we are not in a position for some time to come to increase the compulsory education age to 16 years, the employers ought to unite in establishing a standard of admission to work, which standard the school is to furnish and certify to, on dismissal of the pupils. This would compel the school people to get out of their isolation and to study the educational questions, not only from the academic standpoint, but also from the social and industrial standpoint.

Thus, on the pupils leaving school properly equipped we have to deal with two classes—the one class which cannot or will not go to school any longer, the other class going to school two years more. The former, going to work, ought to be taken care of by an arrangement between the school and the employer, the latter sending his employees under 16 years of age to school during a given number of hours during the week, or have them instructed in the shop, if desired. This is where the employer must make sacrifices. An outline of what should be required was given.

Dr. R. MOLDENKE presented a paper on specifications for foundry coke. This is printed in full elsewhere in this issue

Mr. BENJ. D. FULLER discussed in a general way the methods of raising the efficiency of a foundry. Mr. THOS. D. WEST spoke on the personal equation in accidents. Foundry transportation methods were discussed in two papers by Mr. DAVID GAEHR and Mr. A. W. MOYER.

Dr. JOHN JERMAIN PORTER, for the committee on standards for iron castings, presented a paper on "chemical standards for iron castings." It is probably the largest collection of analyses of iron castings ever gathered into one table. Naturally, it cannot be abstracted.

Before the brass founders, Mr. E. S. SPERRY spoke on fluxes as applied to the brass foundry; Mr. C. R. STEVENSON on brass foundry costs and statistics; Mr. JESSE L. JONES on the selling of brass foundry refuse; Dr. Jos. W. RICHARDS on the electric power required to melt brass, bronze, etc.; Dr. J. J. PORTER on co-operative courses in metallurgy; Mr. CHARLES T. BRAGG on modern foundry progress; Mr. HUGH MCPHEE on mounting patterns on molding machines; Mr. C. P. KARR on analysis of lead in brass alloys; Dr. W. F. HILLEBRAND on standard samples of brass, and Mr. H. M. LANE on magnesium for deoxidizing aluminium alloys.

Major JOSEPH T. SPEER, of the Pittsburgh Valve, Foundry & Construction Company, of Pittsburgh, is the new president of the American Foundrymen's Association. Mr. N. K. B. PATCH, of the Lumen Bearing Company, of Toronto, Canada, is the new president of the American Brass Founders' Association. Mr. GEORGE R. RAYNOR, of the Carborundum Company, of Niagara Falls, is the new president of the Foundry and Manufacturers' Supply Association. Mr. ROBERT B. THOMSON, of the Pitts Company, of Buffalo, is the new president of the Associated Foundry Foremen.

Exhibition.

As on former occasions, the chief interest centered in the exhibition. This was greater, more instructive, and more interesting than ever. We can give here only a few of the most notable exhibits.

The **Monarch Engineering & Manufacturing Company,** of Baltimore, exhibited a complete line of furnaces, operated by oil or gas, for the heating and melting of non-ferrous metals, ladle heaters, bolt heating furnaces, blowers, etc.

The **Rockwell Furnace Company,** of New York, showed their oil, coal or gas-fired melting furnaces, special burners, and complete furnace equipment.

The **Hawley Down Draft Furnace Company,** of Chicago, exhibited their line of melting furnaces, including the Schwartz furnace, the Reyelbee furnace, etc.

The **J. D. Smith Foundry Supply Company,** of Cleveland, exhibited brass and aluminium melting furnaces, core ovens, sprue cutters, molding machines, and sand-blast equipment.

The **Anthony Company,** of New York City, showed a fuel-oil crucible tilting furnace for non-ferrous metal melting in operation; also high and low-pressure oil burners, etc.

The **Hauck Manufacturing Company,** of Brooklyn, N. Y., showed several types of oil-burning appliances, portable heaters, skin drying burners, cupola lighters, kerosene torches, forges of various types, etc.

Mr. **J. B. Wise,** of Watertown, N. Y., showed melting furnaces for non-ferrous metals.

The **Kroeschell Bros. Company,** of Chicago, Ill., exhibited their Kroeschell-Schwartz tilting and stationary crucible furnaces, ladle heaters, etc.

The **American Blower Company,** of Detroit, Mich., exhibited an electrically operated cupola blower in operation; further electric forge blowers, ventilating and heating apparatus, etc.

The **Detroit Foundry Supply Company,** of Detroit, Mich., had a general exhibit of foundry supplies and equipment, including brass furnaces, oil burners, core ovens, plating dynamos and platers' supplies of a very diversified character, molding machines, etc.

Another general exhibit of foundry supplies and equipment was made by the **Central Foundry Supply Company,** of Columbus, Ohio.

Resting rooms for foundry men with general foundry supply exhibits had been fitted up in the booth of **T. P. Kelly & Co.,** of New York City, and in the booth of the **S. Obermayer Company,** of Cincinnati, Ohio.

Walter MacLeod & Company, Cincinnati, Ohio, exhibited melting furnaces, oil burners, drying molds, sand-blast machines, etc.

The **J. W. Paxson Company,** of Philadelphia, Pa., exhibited a general line of foundry supplies, cupolas, sand blasts, magnetic separators, overhead tram rails, etc.

The **Whiting Foundry Equipment Company,** of Harvey, Ill., showed photographs of foundry equipment, cupolas, ladles, elevators, turntables, core ovens, cranes and various labor-saving devices for foundries.

The exhibit of the **Northern Engineering Works,** of Detroit, Mich., included the Newten cupola, a working model of a traveling crane, trolley cranes, hoists, foundry ladles, tramway systems, foundry elevators, etc.

The **Detroit Hoist and Machine Company,** Detroit, Mich., exhibited pneumatic hoisting machinery.

Victor R. Browning & Company, of Cleveland, Ohio, exhibited hoisting and conveying machinery for foundry use.

The **Ingersoll-Rand Company,** of New York City, exhibited compressed-air appliances for foundry use, air compressors, pneumatic sand hammers, chipping hammers, drills, motor hoists, etc.

The **Q. M. S. Company,** of Plainfield, N. J., exhibited air hoists, hand-power traveling cranes, trolleys, steel foundry saws, etc.

The **Calumet Engineering Works,** of Harvey, Ill., exhibited tumbling mills, ladles, trolleys, cupolas.

The **Hanna Engineering Works,** of Chicago, Ill., exhibited Rathbone multiple molding machines, screen shakers, mold dryers, etc.

The **Thomas W. Pangborn Company**, of New York, exhibited high-pressure sand-blast system, sand-separating systems, sand dryers and air separators.

The **Standard Sand and Machine Company**, of Cleveland, exhibited machinery for mixing and handling sand.

Frederick B. Stevens, of Detroit, Mich., exhibited foundry facings and general foundry and platers' supplies.

The **United States Graphite Company**, of Saginaw, Mich., had in their booth, which was fitted up as a resting room, an exhibit of plumbago foundry facings.

Stanley Doggett, of New York City, exhibited foundry facings and parting compounds, manganese dioxide, ferromanganese, ferrosilicon, etc.

The **Joseph Dixon Crucible Company**, of Jersey City, N. J., exhibited a complete line of graphite products, including crucibles, muffles, retorts, stirrers, foundry facings, etc.

The **Jonathan Bartley Crucible Company**, of Trenton, N. J., had a large exhibit of standard and special crucibles, and other foundry supplies made from graphite.

The **Carborundum Company**, of Niagara Falls, exhibited carborundum products, like grinding wheels, carborundum cloth and paper, also carborundum fire sand, which is used as a refractory.

The **Detroit Testing Laboratory**, of Detroit, Mich., showed electric furnaces for laboratory work, photomicrographs of metals, and charts giving the results of chemical and physical tests of metals.

Rogers, Brown & Company, of Cincinnati, Ohio, exhibited samples of pig iron, coke, fluorspar, and showed an interesting map exhibiting with electric flash lamps of different colors their sales offices, the blast furnaces, and the coke properties in which the company is interested.

S. Birkenstein & Sons, of Chicago, Ill., exhibited samples of ingot copper, pig lead, spelter, block tin and general brass founders' alloys.

The **Michigan Smelting and Refining Company**, of Detroit, Mich., exhibited yellow and red brass ingots and sample castings; also solders, babbitt metals, and various new nonferrous metals or alloys.

Hickman, Williams & Company, of Chicago, Ill., exhibited samples of pig irons, ferromanganese, ferrophosphorus, ferrosilicon, and silico spiegel.

The exhibit of the **Goldschmidt Thermit Company**, of New York City, included materials and appliances for the application of the thermit process in foundry practice; especially titanium-thermit cans for purifying molten iron and steel; nickel thermit for the treatment of cast iron; thermit for welding and repairing of flaws in castings. Demonstrations of the use of thermit in foundry practice were made. The exhibit of this company also included samples of the different metals and alloys made by the aluminothermic reaction, especially chromium, manganese, ferrotitanium, manganese-copper, manganese-zinc, and manganese-tin, all of these being free from carbon.

Analyzed Samples of Bureau of Standards.

The Bureau of Standards, Washington, D. C., announces that it is ready to distribute the following analyzed samples:

No. 6a Iron D, replacing the original D of the American Foundrymen's Association.

No. 24 A vanadium steel with between 0.1 and 0.2 per cent vanadium At present only a provisional certificate of analysis can be issued with this steel.

No. 25 A manganese ore, for total manganese and available oxygen

No. 26 "Crescent" iron ore, for alumina, lime and magnesia

No. 27 "Sibley" iron ore, for iron, silica and phosphorus.

No. 28 "Norrie" iron ore, for manganese (low)

A new schedule of fees goes into effect July 1, 1910, and thereafter the only discount allowed will be 10 per cent on four or more samples, alike or different.

Two new circulars relating to the bureau's standard samples have been prepared for free distribution. Of these, Circular 25 contains general information regarding all samples, and Circular 26 relates to the methods used by the bureau chemists and many others in analyzing the above manganese and iron ores.

American Peat Society.

The fourth annual meeting of the American Peat Society will be held in Ottawa, Canada, on July 25, 26 and 27.

Through the liberality of the Canadian Government and the great efforts of the president of the American Peat Society, Dr. Eugene Haanel, director of Mines of the Department of Mines, a very attractive program has been arranged.

The Canadian Government has a fuel testing station at Ottawa, Canada, which is equipped for the testing of peat only for power purposes. The plant consists of a double-zone peat-gas producer, wet coke scrubber, tar filter and dry scrubber, a four-cycle, single-acting Koerting gas engine of 60 hp, a 50-kw direct-connected Westinghouse direct-current generator, and a resistance stand for absorbing the generated energy. The plant will later be utilized for running the machinery of a metallurgical laboratory.

The peat for the operation of the producer is supplied from the peat bog owned by the Mines Branch, Department of Mines. This bog is situated at Alfred, on the Canadian Pacific Railroad, near Caledonia Springs, about 40 miles distant from Ottawa, and has an area of about 300 acres. The Anrep process for manufacturing air-dried peat is employed, the plant installed having a daily capacity of about 30 tons of air-dried peat.

Both the peat fuel testing plant at Ottawa and the peat plant at Alfred will be visited by the American Peat Society during the meeting. The professional program of papers is quite full, papers having been announced by Dr. E. Haanel, Mr. A. Dobson, Mr. A. Anrep, Jr., Dr. J. H. Pratt, Dr. T. A. Mighill, Mr. F. J. Bulask, Mr. E. V. Moore, Mr. F. H. Nagel, Prof. C. A. Davis, Dr. J. McWilliam, Mr. P. Heseltine, Mr. M. Joltz, Mr. J. C. Schumacker, Prof. R. H. Fernald, Mr. R. Schorr, Mr. O. E. Moulton, Mr. J. H. Van Glahn, Mr. William Eichoff, Mr. J. N. Hoff, Mr. R. Runson, Mr. J Bordollo and Dr. O. Zwingenberger.

The hotel headquarters are at the Russell House. The chairman of the local reception committee is Mr. Arthur J. Forward, Ottawa, Canada.

Electrolytic Cyanidation and Amalgamation.

By Elmer Ellsworth Carey

Inventions and processes are frequently announced many years before the world is ready for them. The electrolytic cyanide process was described by Rae in 1867, but to-day only one part of that process—electrodeposition—is in use. The electrolytic system of separating and refining metals is now rapidly superseding all other metallurgical systems, and the electrochemical system will before long be used extensively in the metallurgy of gold. By the use of electricity the percentage of gold won will be increased, the cost of extraction diminished, and the area of mining operations greatly enlarged.

As soon as the principles of electrochemical metallurgy taught by the various colleges, institutes, technical schools, etc., become better recognized in practice, important modifications and changes will be made in milling operations As cyanidation made slow progress till a new generation of engineers had been trained in the use of the chemical system, so the electrochemical metallurgical methods will languish till the schools have supplied the necessary electrochemical engineers.

It has been known theoretically for 40 years that the amalgamating action of mercury and the energy of the cyanide

solution could be increased by the introduction of certain electrochemical conditions, but many difficulties had to be overcome before this knowledge could be used in a practical manner. Within the last 10 years the science of electrochemistry has been placed on a firm foundation, and the troubles of the early inventors have been surmounted. Low-voltage generators are now manufactured that produce the required current at a low cost; by the use of Acheson graphite, all anode trouble disappears; and the time of treatment has been reduced to a few hours, while in some systems the values pass into solution and are deposited simultaneously.

Metallurgical science took a great step forward when the chemical (cyanide) system was introduced, and the next important step will be the adoption of the electrochemical system. Many troubles, obstacles and difficulties of cyanidation disappear when electrochemical conditions are introduced; values which ordinarily are refractory readily pass into the solution; a weaker solution than usual may be used; the solution is constantly regenerated; the loss of cyanide is reduced; the values are automatically recovered in the form of amalgam; the refining processes, with their attendant losses, are unnecessary, and the cost of installation greatly reduced.

In the *Mining World*, Jan. 23, 1909, may be found a description of the Parks electrocyanide process. The process here described is extremely simple, effective and inexpensive, and has been found successful with ores which are not amenable to the usual cyanide treatment. In the works on cyanide practice by Bosqui, Wilson, Smart and others will be found descriptions of other systems of electrocyanidation, all of which have similar basic features, differing only in detail, modes of construction, etc. Some years ago electrocyanidation was successfully used for several months at the Delamar Mine in Idaho, and a description of the process there used will be found in Bosqui's work.

The recovery of values from a cyanide solution (after the usual leaching process) by electrodeposition has proved successful in many mills in South Africa; the Siemens & Halske process of electroprecipitation was introduced several years ago on the Rand, and despite the primitive apparatus used, the high cost of installation, and the 5 per cent royalty, many companies have been using the process for a number of years.

To-day the special process in question is obsolete, as far better methods are known; Acheson graphite is much better for anodes than the clumsy anodes used in the German system, and amalgam plates can now be used instead of the lead cathodes. With the introduction of discoveries, methods, machinery, and accessories which have been placed at our disposal by American inventors and investigators in the electrochemical field, the time has now arrived when the art of cyanidation can be completely revolutionized. Some of the advantages that may be expected from the improved system are mentioned in an article of electrochemical amalgamation and cyanidation in the *Engineering Magazine* for December, 1909, as follows:

"Turning to electrochemical lixiviation, the claim is made that under electrochemical conditions the cyanide solution is much more active than usual; values pass into the solution which are lost in the ordinary systems; the time of treatment is reduced and the extraction increased, while there is a decrease in the operating cost; precipitation by electrodeposition goes on simultaneously with the leaching process and the usual troubles are eliminated; a molecule of cyanogen after taking up a particle of gold, is released as nascent cyanogen, having greatly increased affinity for gold and silver. The solution is constantly regenerated, the loss of cyanogen being negligible. The larger particles of gold are amalgamated directly without passing into the solution. In electrocyanidation the pulp and solution are agitated in a shallow tank having the bottom covered with quicksilver (cathode); revolving stirrers cause the pulp to travel with a circular motion over the quicksilver for an hour or two; this is equivalent to passing the pulp over several miles of mercury surface. During this period the values are amalgamated directly, or pass into the solution and are precipitated by electrolysis, the extraction averaging 97 per cent. All ores, so far as known, are capable of successful treatment by means of electrocyanidation."

Regarding electroamalgamation, it may be said that experimental work relating to electrochemical amalgamation began in 1859, the first patent being issued to Henry Brevoost, of San Francisco; some 75 patents for various electro-amalgamating devices have been issued by the United States Patent Office, but only within the last three years have machines been constructed which meet the requirements of mining men. The various engineering difficulties have been overcome and an electro-amalgamation device is now constructed which is very simple, inexpensive, has large capacity, is easy to operate, skilled attendance not being necessary, and which extracts all values not encased, regardless of refractory elements which may be present in the gangue.

Such a device works equally well with mill pulp or screened placer material; in placer operations all rusty, coated and black sand values, as well as platinum, are amalgamated; fine, float and beach gold is readily amalgamated in the electrolytic device. Those who wish an authoritative statement as to the value and scope of electrochemical amalgamation are referred to a paper by Warnford-Lock mentioned below, and also to the patent issued to Sir William Crookes.

Those who wish to further investigate electrocyanidation are referred to an article by John R. Parks in the *Mining World*, Jan. 23, 1909; an article on "Electrochemical Amalgamation and Cyanidation," in the *Eng. Mag.*, December, 1909, and the following books: "Cyanide Process," by E. B. Wilson; "A Handbook of Practical Cyanide Operations," by Wm. H. Gage; "Practical Notes on the Cyanide Process," by F. L. Bosqui; "Electric Smelting," by Dr. W. Borchers, and "Cyaniding Gold and Silver Ores," by H. Forbes Julian and Edgar Smart.

Additional references for data regarding electroamalgamation are as follows:

"Electrochemical Amalgamation," C. G. Warnford-Lock, E. M., M. L. I. M., Vol. 1, page 205, *Proceedings* of the (London) Institution of Min. and Met.

United States patent No. 462,535, issued to Sir William Crookes for an electrolytic amalgamating process; the following numbers refer to other United States patents covering devices for saving gold by electrolytic methods: 328,532, 307,081, 342,421, 492,711, 590,524, 947,957, 947,958, 757,557.

Other articles on electro-amalgamation have appeared in the *Mining Journal* (London), May 15 and June 12, 1909; *Mining World*, March 27, April 17 and June 5, 1909; *Pacific Miner*, April and August, 1909, also February, 1910; *Electrochemical and Metallurgical Industry*, May, 1909; METALLURGICAL AND CHEMICAL ENGINEERING, March, 1910, page 116 and 157; *Scientific American*, Aug. 7, 1909; *Scientific American Supplement*, Nov. 6, 1909; *Mexican Mining Journal*, October, 1909, and January, 1910.

SAN JOSÉ, CAL.

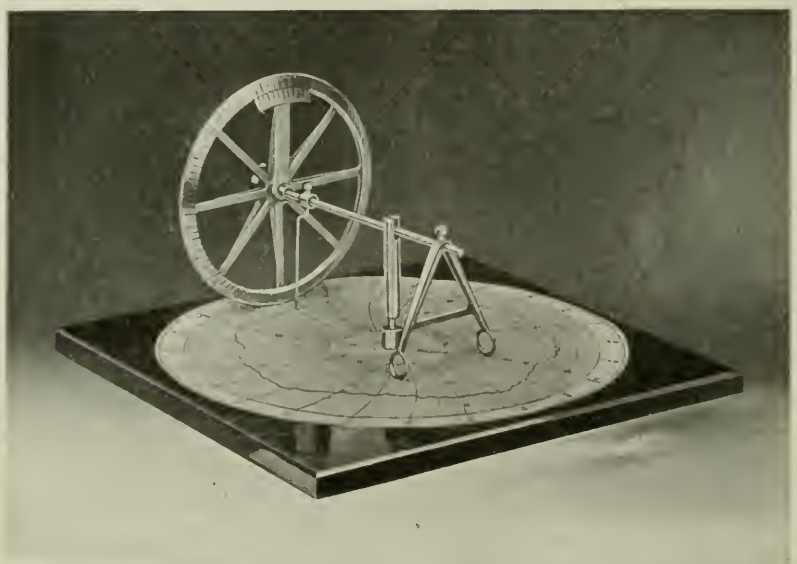
Commission of Conservation of Canada.—The Commission of Conservation of Canada (Hon. Clifford Sifton, chairman, James White, secretary) has just issued the report of the first annual meeting held at Ottawa from January 18 to 21, 1910. It is a very interesting volume of 216 pages. Besides the inaugural addresses by Hon. Clifford Sifton and by the Governor-General of Canada, Earl Grey, it contains papers by Dr. B. E. Fernow on scientific forestry in Europe, by Dr. J. W. Robertson on the conservation of agricultural resources, by Dr. Eugene Haanel on possible economies in the production of minerals of Canada, by Hon. Frank Cochrane on the conservation of the natural resources of Ontario, by Hon. Adam Beck on the conservation of the waterpowers of Ontario, by Mr. C. E. Contlee on the water wealth of Canada, etc.

Averaging Instrument for Circular Chart Records

The Bristol Company, of Waterbury, Conn., have for the last 20 years made a specialty of manufacturing automatic recording instruments for pressure, temperature and electricity. As this class of recording instruments using circular charts have come into general use there has developed a demand for a simple device to quickly determine the average of the record made on such charts, and the integral value for the whole 24 hours, or for the time covered by the record.

This long-felt demand has been filled by the instrument illustrated herewith, which is based upon a fundamental plan as worked out and patented by Prof. W. F. Durand, of Stanford University, and is constructed in accordance with a novel design recently patented by Wm. H. Bristol, president of the Bristol Company.

The instrument can be applied for averaging records of any kind on circular charts having uniform graduations, as, for instance, records of watts, amperes, temperature, pressure, etc.



AVERAGING INSTRUMENT FOR CIRCULAR CHART RECORDS.

Recording instruments equipped with circular charts are therefore made available for a number of applications for which it was previously thought necessary to use the instruments recording on straight lines or strip record charts. Recording differential pressure gages are coming into use for measuring velocities and volumes of liquids, air or gas flowing in mains, and this integrating device will prove of value for quickly obtaining total volumes of flow for any given period of time.

The simple construction of the instrument is shown in the accompanying illustration. A wooden base with a metal socket is provided for supporting and centering the chart. The socket holds a rotatable pin with a vertical slot at the top to receive the bar which carries the integrating tracer point and triangular support. The vertical groove in the rotatable pin allows the integrating wheel to roll on the chart with uniform pressure due to its own weight.

The integrating wheel is 6 in. in diameter, the rim being graduated into 100 numbered equal spaces, and is fitted with a vernier, which makes it possible to easily read with the naked eye to one-tenth of one division on the integrating wheel. The wheel is of such large size that it is not necessary to supply any counting device for the number of revolutions. The number of complete revolutions cannot be more than two, even for a record of maximum size on the large 12-in. charts.

To operate the instrument the thumb and forefinger of one hand are applied to the base of the triangular support, which is moved radially, so as to cause the tracer point to continually follow the record curve, while the chart is turned with the other hand.

By referring to a line plotted on a sheet of cross-section paper furnished with the instrument, for the particular record curve that is to be measured, the total reading for the entire 24 hours may be taken off directly.

A full explanation of the theory upon which the operation of the instrument depends has been given by Professor Durand in a paper presented at the New York meeting of the American Society of Mechanical Engineers in 1908. This may be briefly summarized as follows:

In applying the instrument it is necessary to have a uniform radial scale, from which it follows that equal increments in the length of the radius correspond to equal increments in the watts, amperes, temperature or whatever quantity is measured.

The integrating wheel being carried at right angles to shaft passing through the center does not turn and give a reading when the tracing point is moved on a straight radial line, but if the tracing point is made to follow a record, the integrating wheel will revolve and the amount of the revolution will correspond to the total of the circumferential elemental components of the record curve, the radial elemental components of the record having no turning effect on the integrating wheel.

As the lengths of the arcs of concentric circles for given angles or for the entire circumference are proportional to their radii, it is evident that the amount of turning of the integrating wheel, and the reading obtained thereon, will be proportional to the average radius of the record traced.

The instrument is adapted for integrating charts with either straight or curved radial time arcs. The correction necessary for radial time arcs which are curved may be made, after completely tracing the record, by returning the tracing point to a point on the chart having the same radius as the starting point, the movement of the tracing point being along an arc corresponding to the curved radial time arcs.

American Society of Civil Engineers.—The forty-second annual convention of the American Society of Civil Engineers was held in Chicago from June 21 to 24. A very enjoyable series of entertainments and excursions had been arranged, to Gary, Ind., to the drainage canal and its power plant and various others. Illustrated lectures were given by Mr. A. Noble on the New York tunnel of the Pennsylvania R. R., and by Mr. J. W. Smith on the New York water supply system.

NOTES.

American Iron and Steel Institute.—The first regular meeting of the American Iron and Steel Institute, organized in 1908, was held on May 27, 1910, at the Waldorf-Astoria in New York. The first annual meeting will be held in October and will last two days. The officers elected are: E. H. Gary, president; P. Stackhouse, W. L. King, Charles M. Schwab, vice-presidents, Edward Bailey, treasurer; W. J. Filbert, secretary.

Mica Production in the Custer district of the Black Hills, South Dakota, is being carried on by the Westinghouse Company and the Chicago Mica Company. The former ships its products to Denver mills for dressing.

The Steptoe Smelting Company, operating one of the large copper concentrators and smelters in the Ely district, Nevada, is reported to be running all (eight) sections of the concentrators, the full battery of the McDougal roasters (16) and three out of its four reverberatory furnaces.

The Cananea Consolidated Copper Company, in Cananea, Sonora, Mexico, is increasing its roasting plant and has placed an order with Allis-Chalmers Company for six McDougall furnaces. These are exactly like the four furnaces which they now have in service. The furnaces have an outside diameter of 18 ft. 6 in. and are 32 ft. high. They are the six-hearth type.

Census of Electric Central Stations.—According to figures just issued by the Census Bureau of the Department of Commerce and Labor, the number of commercial and municipal plants increased from 3620 in 1902 to 4714 in 1907, which is an increase of 30.2 per cent. The application of the same rate of increase to the estimated number (50,000) of isolated plants in 1902 gives an estimate of 65,000 for 1907. Of the 4714 central stations reported in 1907, 227 had no generating equipment, while 113 had more than one power plant, 337 generating plants being reported from these 113 stations. There were in 1907 upward of 30,000 individuals, companies, corporations and municipalities, exclusive of isolated electric plants, which reported the generation or utilization of electric current in commercial enterprises. These industries represent an outstanding capitalization of $6,209,746,753, of which $1,367,338,836 is credited to electric central stations, $3,774,722,096 to electric railways, $814,616,004 to commercial or mutual telephone companies, and $253,019,817 to telegraph companies.

Electric Heating for Compressed Air Pumps.—The possibilities of electric devices for the most varying purposes in connection with metallurgy seem to find attention in Shasta County, California, the classical district in which the electric furnace has first been tried in this country for iron-ore reduction. The latest application of electric heating is due to Mr. H. A. Sutliffe, of the Bully Hill copper mines, who uses a simple electric heating device in connection with pumps operated by compressed air. The object of heating is simply to raise the air pressure just before the compressed air is utilized. It is claimed that this is so successfully done that each electric heater which is operated with a current of 40 amp from the 110-volt supply circuit and therefore consumes 4.4 kw, saves something like $6 a month. The device consists of a resistance-wire coil around the air pipe, provision being made, of course, for proper insulation and for disconnecting the resistance coil when the pump in question is not working.

Gary, Ind.—The first plant of the American Sheet & Tinplate Company, at Gary, Ind., on the lake shore just west of the Indiana Steel Company property, is expected to be finished by March, 1911, with 2000 employees. The American Bridge Company is also well along with its new Gary shops. Both plants will use about 30,000 kw. This power will be obtained from the great gas-engine driven generating stations at Gary and South Chicago (working in parallel) of the U. S. Steel Corporation.

PERSONAL.

Mr. **Geo. F. Duck**, formerly consulting engineer of Pittsburgh, has succeeded Mr. R. L. Herrick as Western editorial representative of *Mines and Minerals*, with headquarters in Denver, Col.

Dr. **W. A. Hendry**, manager of the Hendryx Cyanide Machinery Company, of Denver, has removed to New York City, where the business of his company will in the future be conducted. The new offices of the company are at 107-109 William Street.

Mr. **R. L Herrick**, who has been the Western representative of *Mines and Minerals* for several years, has resigned his position and will engage in business near Cincinnati, Ohio.

Mr. **E. M. Johnson** recently accepted a position as assistant superintendent with the Mineral Point Zinc Company, Depue, Ill.

Mr. **J. M. McClave**, metallurgical engineer of Denver, Col., is engaged in experimental work near Fairview, Col.

Mr. **Charles M. Palmer**, formerly in charge of the B. & M. laboratory in Butte, Mont., has been transferred to the smelter at Great Falls, Mont., where he will be engaged in work on the smoke and fume problem.

Mr. **H. F. Parshall**, the distinguished consulting engineer of London, has received the degree of Doctor of Science from Tufts College.

Dr. **Jos. W. Richards**, of Lehigh University, has sailed for Europe to attend the International Mining and Metallurgical Congress in Düsseldorf, but will be back early in July.

Mr. **J. B. Risque**, of Salt Lake City, who has been general manager of Utah Consolidated Company for the past three years, has resigned his position, to take effect Sept. 1. Mr. Risque will continue to engage in mining and metallurgy after his resignation, with headquarters in Salt Lake City.

Mr. **W. J. Sherwood**, manager of the Rambler Copper & Platinum Company, Holmes, Wyo., recently spent a few days in Denver, en route to the company's head offices in Chicago.

Mr. **W. G. Swart**, representative of the American Zinc Ore Separating Company, with headquarters in Denver, recently spent several weeks in Montana on professional business.

Mr. **Theo. F. Van Wagenen**, who has been spending some months at Zacatecas, Mexico, on professional business, has returned to Denver.

Dr. **G. B. Waterhouse** has been appointed metallurgist of the Lackawanna Steel Company, to succeed Mr. H. Cook.

Mr. **R. H. Wolff**, of New York, representative of the Héroult electric steel process in this country, sailed for Europe on the *Mauretania* on June 22, to be abroad for about two months.

NEW BOOKS.

Albe, E. E. Fournier d'. The electron theory. A popular introduction to the new theory of electricity and magnetism; with a preface by G. Johnstone Stoney; with front and diagrams in text. Third edition. 354 pages. Price $1.25 net. New York: Longmans, Green & Co.

Moore, F. J. Outlines of organic chemistry. A book designed especially for the general student. 324 pages. Price $1.50, net. New York: John Wiley & Sons.

Sexton, A. H. Fuel and Refractory Materials. 364 pages. Price $2.50 net. New York: D. Van Nostrand Co.

Sharp, J. Modern Foundry Practice. Second edition revised. 769 pages. Price $8.00. New York: Spon & Chamberlain.

BOOK REVIEWS.

Modern Assaying. By **J. Reginald Brown.** Edited by **F. W. Braun.** 145 pages. Cloth. Price, $1.50. Philadelphia: J. B. Lippincott Company.

The sub-title of this book describes it as "a concise treatise describing latest methods and appliances," but it may be said to be more complete in the latter respect than in the former. It illustrates the great difficulty of presenting a technical subject in a "popular," or untechnical manner, at the same time confining its descriptions of apparatus to that of one manufacturer Technically, the volume cannot have as great a value as some other treatises on the same subject, although it may find acceptance with a class of operators who are not deeply versed in the technical refinements of their work, and who are content to follow directions without a knowledge of the underlying reasons, causes and effects. Methods are given for the fire assay of gold, silver and lead; electrolytic assay for lead and copper, and wet methods for lead and copper. The retorting of amalgam is also described.

* * *

Tables for the Determination of Minerals. By **Persifor Frazer,** D.Sc., and **Amos P. Brown,** Ph.D. 125 pages. Flexible leather. Philadelphia: J. B. Lippincott Company, Price, $2.50 net.

This is the sixth edition of the authors' translation of Prof. Albin Weisbach's *Tabellen.* The original tables were compiled by Dr. Weisbach for the use of his students at Freiberg, and were based on determination from outward appearance and physical characteristics. While a complete classification of minerals cannot be made on this basis, nevertheless it has been found more convenient in general use than a more elaborate system based on chemical composition.

Weisbach divided minerals into three systems, viz.: those of metallic luster, those of non-metallic luster which give a colored powder, and those of non-metallic luster and colorless streak. These three systems have subdivisions so that it is comparatively easy to classify a mineral in a short time by observing closely its physical characteristics.

The translators have retained Weisbach's system and have added to it accurate chemical formulae of the minerals. Dana's system of nomenclature, Mohs' scale of hardness and von Kobell's scale of fusibility are used throughout the work.

The volume should be of great value to students and to those who find it necessary to identify minerals in field work. An excellent index adds to the value of the work.

* * *

Metallography. By **Cecil H. Desch,** D.Sc. (Lond.), Ph.D. (Wnrzb.), Graham Young, Lecturer in Metallurgical Chemistry in the University of Glasgow. London and New York Longmans, Green & Company. Price, $3.

This book is divided into 18 chapters which deal with the following headings: Introduction; The Diagram of Thermal Equilibrium, Solid Solutions or Mixed Crystals; Ternary and More Complex Systems; Metals which are Only Partially Miscible in the Liquid State; Practical Pryometry and Thermal Analysis; The Preparation of Micro-Sections; The Microscopical Examination of Prepared Sections; The Crystallization of Metals and Alloys. Undercooling and the Metastable State; Diffusion in the Solid State; The Physical Properties of Alloys: Density—Thermal Expansibility—Hardness—Electrical Conductivity—Thermo-Electric Power—Magnetic Properties; Electromotive Force and Corrosion; The Construction of the Equilibrium Diagram, The Molecular Condition of Metals in Alloys and the Nature of Inter-Metallic Compounds; The Plastic Deformation of Metals and Alloys; The Metallography of Iron and Steel; The Metallography of Industrial Alloys.

This book sets forth very clearly our present knowledge of metallography, which is defined as a study of the internal structure of metals and alloys and of its relation to their composition and to their physical and mechanical properties. As it is one of the text-books of physical chemistry edited by Sir William Ramsay, it naturally takes up the study from that standpoint.

Starting with the diagram of thermal equilibrium, the basis of which is the freezing point curve, the author, by clear steps, works out the different cases and merely touches upon the phase rule and shows its application to alloys. The discussion of the ternary systems is exceedingly well written. Under the heading of practical pyrometry and thermal analysis, we have given us in 40 pages a condensation of the methods in use and the instruments for the autographic registration of curves. It would seem to us that this chapter was rather too condensed for the ordinary student; however, the references given cover the subject very thoroughly.

The chapter on the Crystallization of Metals and Alloys is exceedingly well written and the microphotographs illustrating it are very happily chosen. The chapter on the physical properties of alloys, covering density, thermal expansion, hardness, conductivity, etc., covers the ground very thoroughly and we can gather a very clear idea of what has been done and what remains to be done on this side of the subject.

The latter part of the book, dealing with the metallography of iron and steel, summarizes, in a very few words, our present knowledge of the subject and the book ends with an appendix consisting of tables of systems of which equilibrium diagrams have been published, the probable formulae of inter-metallic compounds present have been added. The table is divided into: eutectiferous series, isomorphous series, the metals only partially soluble in liquid state; series forming a single compound, complex series in which two or more compounds are formed, and systems which have not been completely investigated, and, lastly, the ternary systems.

On reading through this book, one is immediately struck by the feeling that Mr. Desch is familiar with his subject from the practical standpoint and not merely from reading the researches of others. He speaks with authority and, therefore, what he says is convincing. In addition, the style is so straightforward and simple that the book will be found both instructive and fascinating, not only to students, but also to practitioners who are interested in metallurgy in general, and alloys in particular. The book covers a very wide field, but covers it well, and Mr. Desch is to be congratulated in supplying a long-felt need, because there is no doubt that to-day metallography has become quite an important branch of study.

* * *

Fuel and Refractory Materials. By **A. Humboldt Sexton,** F.I.C., F.C.S., emeritus professor of metallurgy in the Glasgow and West of Scotland Technical College. New and revised edition. 364 pages, illus. Price, $2.50 net. New York: D. Von Nostrand Company.

There are 15 chapters, dealing with combustion, heating power of fuels; wood, peat, coal; charcoal, peat-charcoal, coke; coal washing; liquid fuels; gaseous fuel; recovery of by-products; furnaces for metallurgical purposes; supply of air to the furnace, removal of waste products, smoke, prevention of smoke; pyrometry; calorimetry; utilization of fuel; testing fuels; refractory materials, bricks, crucibles.

This is the second edition of Professor Sexton's book and while no essential change has been made in the scope and general arrangement of the book, yet many additions have been made to the chapters on coke ovens, gas producers, and pyrometry Descriptions of modern forms of plant and apparatus have been added and some descriptions of older types of plant have been deleted.

The author says: "In spite of the progress which is being made in the adoption of the metric system in this country, it is unfortunately still necessary to use the British system of weights and measures. These have, therefore, been retained." But why not adopt a compromise and give both the metric units and the British units, perhaps the latter in parenthesis? Temperatures are given by the author both in degrees Fahrenheit and Centigrade. The book is well and clearly written.

Metallurgical and Chemical Engineering

A Consolidation of Electrochemical and Metallurgical Industry and Iron and Steel Magazine

Vol. VIII. New York, August, 1910 No. 8

PUBLISHED MONTHLY BY THE

Electrochemical Publishing Company

239 West 39th Street, New York.

Pittsburgh Office, 201 Oliver Building.
Western Office, 1510 Court Place, Denver, Colo.
European Office, Hastings House, Norfolk St., Strand, London, Eng.

J. W. Richards, Ph.D., President.
Curtis E. Whittlesey, Treasurer. E. F. Roeber, Secretary.
Telephone Call: 4700 Bryant.

E. F. Roeber, Ph.D., Editor.
H. C. Parmelee, Western Editor.

Yearly subscription price for United States, Mexico and United States dependencies, $2.00; for all other countries, $2.50 (European exchange, 10 shillings, 10 marks, 12.50 francs).

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Entered as Second-Class Matter at the Post Office at New York, N. Y., under the Act of Congress, March 3, 1879

New York, August, 1910.

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History Made by the Internal Combustion Engine.

It is interesting and not uninstructive to pause for a moment and reflect how the internal-combustion engine is making history. We trust it will not offend the friends of the "steamer" to suggest that it has practically made the automobile with its important and far-reaching effects of improving the health, if not the exchequers, of hundreds of thousands of people, giving employment to tens of thousands of men, circulating millions of dollars and making an irresistible sentiment in favor of rapid improvement of the public roads. It has made the motor truck, the gasoline car for light service of steam roads, and if it has not given us the flying machine, it has at least hastened its advent. All these things make history.

* * *

That is at one end. At the other end we have the very large gas engine, using blast-furnace or by-product coke-oven gas, promising to effect a complete revolution in these branches of the iron industry. The time was that the by-product coking process was looked at askance because its general adoption might glut the market for by-products. Not so when the large gas engine is at hand, for with electricity as a means of distribution the power market can never be glutten. By a curious coincidence, the internal-combustion engine makes another demand for a retort by-product, since it is obvious that the automobiles are destroying the old-fashioned macadam roads, while we are assured that tar will save them.

Monel Metal and Nickel Steel.

The treatment of the nickel-copper ores of the Sudbury district in Canada has long been a favorite subject of metallurgical research and invention. The classical process includes blast-furnace smelting and converter treatment near the mines at the works of the Canadian Copper Company, the 80 per cent nickel-copper matte from the converters being then shipped to the works of the Orford Copper Company at Bayonne, N. J., where separation of the nickel from the copper is obtained by means of the Orford process, which is well known in its general principles (smelting with salt cake and formation of tops and bottoms), but carefully guarded by secrecy with respect to some details. Aside from this orthodox treatment, quite a number of inventors have attempted the use of different methods, essentially wet processes with electrolysis. One of these, at least, has been worked on a commercial scale with commercial success for a number of years, that of Mr. David H. Browne, chief metallurgist of the Canadian Copper Company. His process has been described in detail in former volumes of this journal and will always remain a very instructive school example of an interesting combination process, involving chloride electrolysis and almost cyclic with respect to chlorine. When the Canadian Copper Company was absorbed by the International Nickel Company, the Browne process was given

up in 1902 in favor of the Orford process, because the latter is cheaper. One of the chief advantages of the Browne process, that it yields a much purer nickel than the Orford process, was considered not to be of sufficient commercial importance, in view of the limited market for highly refined nickel. The simplicity and effectiveness of the Orford process were the decisive moments.

* * *

A further step in exactly the same direction—toward simplification and cheapening of the metallurgical treatment—has been made in recent years by the introduction of "Monel metal." This is an alloy primarily of nickel and copper in the proportion of three parts of nickel to one part of copper, this being the natural proportion in which these metals exist in the largest deposits of nickel ore in the Sudbury district. The essential simplifying principle which considerably reduces the cost of the metallurgical treatment is that the nickel and copper are reduced together from the ore and directly obtained in form of the desired alloy and that no separation of the nickel from the copper is now attempted. The rationale of this simplified process is to be found in the remarkable physical properties of Monel metal, which were very well summed up in an article by Mr. D. H. Browne in our March issue, 1909 (Vol. VII, p. 114). Monel metal probably attracted first considerable general attention by the large extent to which it has been employed in sheet form to roof the new Pennsylvania terminal station in New York City. If the incorrodibility of this alloy should prove as great in extended practical use as it has been found in tests on a smaller scale, it should find many useful applications in the chemical industries in form of wire, sheet, or castings. At any rate, it is seen that the object of metallurgical research has been shifted by this development. The metallurgical treatment from ore to metal has been so simplified that little remains to be desired in this direction. The object must now be to so thoroughly investigate all the properties of Monel metal as to determine exactly its industrial possibilities and limitations, its commercial applications, in short to create a market.

* * *

In this respect the article in this issue by Prof. C. F. Burgess and Mr. J. Aston on the strength of the alloys of electrolytic iron and Monel metal is very interesting and suggestive. One of the chief uses of nickel is for making nickel steel. Now, Professor Burgess' investigation has shown that the addition of copper as well as of nickel in certain proportions to pure electrolytic iron improves its mechanical properties; further, that the two effects are additive, and that the desired improvement can be obtained by directly using Monel metal instead of copper and nickel. The commercial importance of this discovery is due to the fact that Monel metal is cheaper than metallic nickel. Hence, this result points toward a possibility of reducing the cost of nickel steel. However, the investigation has so far been carried out only with pure electrolytic iron, free from carbon. How the presence of carbon modifies the results must be left to further investigation, which is now being made by the authors. Of course, from a practical standpoint this is all-important. Yet, while not directly applicable to steel practice, the results obtained by Messrs. Burgess and Aston with pure electrolytic iron are important and valuable, the method of beginning their researches with pure electrolytic iron and then studying the modifications of the effects due to other additions is exactly logical and in strict agreement with one of the general rules so strongly emphasized by Dr. McKenna in his admirable presidential address before the American Institute of Chemical Engineers, printed elsewhere in this issue, on the study of the properties of materials—namely, that the simple or homogeneous substances must be the basis of study before the nature of the complex or heterogeneous can be understood. It is necessary to study first the effect of copper, nickel, and Monel metal on pure iron, before the effects of carbon and other elements in commercial steel can be taken up. The results given in the present article of Messrs. Burgess and Aston are certainly among the most interesting and suggestive so far obtained in their extended and painstaking research.

The Technical Graduate in Western Metallurgy.

Occasionally one hears a solicitous inquiry as to the disposition of the rapidly increasing number of technical graduates, particularly those from schools of mining and metallurgy. It is pointed out that the consulting field is already well filled, and that many large corporations place their work in charge of an engineering department with one or two efficient heads, and rely less and less on work done for them by independent engineers. Thus the prospects of the metallurgical engineer would seem to narrow down. But the problem is not so difficult as it may sound to some, and like many other problems which arise from changed conditions, it will find its solution in other changing ideas and practices. A decade ago Western smelters, then operating independently, found that efficiency in operation could be raised by the employment of technically trained men in what might be called the minor positions. Where formerly it had been thought sufficient to train a bright boy as assistant chemist and later promote him to take charge of the laboratory, it is now a settled rule that none but technical graduates are accepted in such positions. The result has been that through this system of successive promotions the Western smelting industry to-day finds itself directed in all of its important departments by young technical men who entered the work in minor positions.

* * *

The instance is but typical of what is happening in other branches of engineering. Large companies appreciate in the technical graduate not only the possibility of performing routine work more efficiently, but the probability of being able to advance him to places of greater responsibility which could not be held by the average so-called practical man. Thus we find that the "mill man" who has grown up in the service and undoubtedly knows his business well and has filled with a large degree of success his place in the development of metallurgical art, is in a measure now being displaced by the technical graduate who has enjoyed a broader training, is able to pay more attention to the refinements of the process, is willing to consider in an unprejudiced manner any proposals of modification and improvements and can undertake investigation and experimental work in an intelligent manner. The day of the average so-called practical man in many lines of metallurgy is passed; exceptions there are and will always be, but they only confirm the rule that in general he is superseded by the tech

nical graduate. The arrangement makes for general improvement. Processes are developed and improved, routine work is handled with some degree of interest, the young graduate lays the foundation for a consulting practice later in life. Undoubtedly this system of using technical graduates will continue to spread until it will be no uncommon thing to find a corps of such men in charge of the actual work of our plants. The technically trained man cannot become a drug on the market if he understands the situation and accepts his place.

Railroad and Other Iron Consumption.

In our issue for October, 1908, under the caption "The Future of Iron and Steel Demand," we reverted to the cyclical nature of many of the occurrences in the iron industry, and also cited figures to indicate that railroad expenditures have also tended to move in a cycle. The intervening period has brought such confirmation of the views then expressed as can be furnished in so short a time. Without attempting to pass over the full ground then covered, it may be observed that we compiled a sort of "index number" to represent the increase in number of miles of road, of cars and of locomotives, weighting the items as seemed appropriate, thus deriving the following:

Annual average increase.	Index number.
1879-1887	129,921
1888-1893	107,209
1894-1898	26,656
1899-1907	180,424

Since 1907 railroad expenditures have been light, much as they were during the period following 1887. In the earlier period, or from 1889 to 1892, inclusive, the total production of iron and steel was large, far exceeding the output of previous years, and yet this large increase was attained despite a material decrease in railroad purchases.

* * *

Precisely the same thing has occurred since our former remarks were printed. Railroad buying has been very light, compared with the active period of 1905-6-7, but total demand has been heavier. In 1909 we made 25,795,471 tons of pig iron, against 25,307,191 tons in 1906 and 25,781,361 tons in 1907, but only 3,000,000 tons of rails against 4,000,000 tons in 1906 and 3,600,000 tons in 1907. Car building showed a much greater decline in 1909. In the first half of this year we made pig iron at the rate of 30,000,000 tons, a gain of more than 4,000,000 tons over 1906 and 1907, but the production of railroad material clearly showed a reduction. Were it possible to deduct the precise tonnage of material, each year, which goes ultimately into railroad consumption the increase in other lines of iron and steel consumption from 1906-7 to the present date would be shown to be tremendous. It is hard to account for the increase. The producers themselves can trace the material only occasionally. Most of the old lines of consumption have shown a growth commensurate with the increase in population and business activity, but, as a rule, this is quite insufficient to explain the total increase, for general business does not seem as good as it was three and four years ago.

Crushing and Grinding.

Radical change and progress are imminent in the matter of crushing and grinding ores. In the 55 years from 1835 to 1890 stamps increased in weight from 50 lb. to about 1000 lb., at which weight they have largely remained in general practice in the United States and Australia. In the last decade, however, the tendency in South Africa has been toward heavier stamps, and the weight of the individual stamp has been increased from about 1000 lb. to 2000 lb. The change has been accompanied by the adoption of two-stage crushing, using the heavy stamp as a coarse crusher, and a regrinding mill, usually the tube mill, as the means of reducing the ore to its final size. The light stamp, crushing the ore to the desired degree of fineness in one operation, has been relegated to a minor place in South African practice. The development of heavy gravitation stamps has been detailed by Dr. W. A. Caldecott in a communication to the Institution of Mining and Metallurgy, and his paper has caused a widespread and interesting discussion among engineers in the Institution and in the South African society. The paper by Dr. Caldecott, together with part of the discussion, will be found elsewhere in this issue.

* * *

The discussion has brought out one point very clearly, viz., while it is true that the heavy stamp will have a higher duty than the light stamp, it does not at once follow that this makes for greater efficiency, in view of the necessity of some sort of regrinding apparatus. In other words, the relative economic efficiency of single-stage and two-stage crushing is not proved by the mere fact that heavy stamps have a high duty. As to the relative efficiency of stamps and tube mills, it has been claimed that if the mesh to which the ore is to be crushed be properly adjusted to the weight of the stamp, the stamp mill will show probably greater efficiency than the tube mill, and this has led one of the opponents of two-stage crushing with stamps and tube mills to argue that if two-stage crushing is necessary in order to cope with larger tonnages of ore, it might be found advisable to use stamps for both the coarse and fine crushing, sizing the product of the heavy stamps before treatment in the light. But this argument overlooks one general metallurgical rule, namely, that when, as is so often the case, a metallurgical process is resolved into two consecutive stages or operations, essentially different machinery are employed in both stages, such are are most suitable to the characteristic features of each stage.

* * *

Dr. Caldecott is of the opinion that the present use of heavy stamps makes further investigation of light stamps largely academic, but this view cannot be shared until it is shown by sufficiently extended experience that the heavy stamp tube mill combination is more efficient from the financial point of view than single-stage crushing. Probably this will turn out so. Again, other regrinding devices have not enjoyed the popularity of the tube mill, and it is not impossible that at least under certain conditions one of them might be used in place of the tube mill if two-stage crushing is generally adopted. But this is simply hypothetical. Evidently a subject has been opened which will not be closed to discussion and experimentation for some time. The tests have brought out a great deal of useful knowledge, but they have revealed some weak points which further experiment must strengthen if the advocates of two-stage crushing expect to maintain their present strong position.

Chicago Meeting of the American Electrochemical Society.

A new section of the American Electrochemical Society was organized on July 7th in Chicago. Prof. Herbert M. McCoy, of the University of Chicago, has been elected chairman, and Mr. A. B. Marvin secretary-treasurer of the section. The first business transmitted by the Chicago section has been to extend to the national society a hearty invitation to meet in Chicago in October.

The society has accepted with great pleasure the invitation of the Chicago section, and as the Wisconsin section will cooperate, a highly interesting and profitable meeting is assured.

Among the visits which are to be arranged is one to the electric steel refining plant (Heroult furnace) of the Illinois Steel Company (U. S. Steel Corporation) at South Chicago, and another one to the electrolytic lead refinery (Betts process) of the U. S. Smelting, Refining & Mining Company at Grasselli. Possibly a trip on the way back east to Sault Ste. Marie will be arranged. The date of the meeting will be from Oct. 13 to 15.

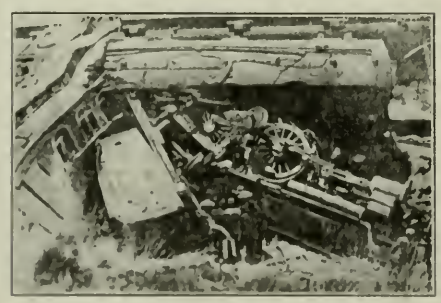
The Wreck of the Special Train of the American Chemical Society.

As noticed before in these columns, a special train had been arranged to take the Eastern members of the American Chemical Society to the San Francisco meeting. The train left Chicago on July 4 and was to arrive in San Francisco on July 12. On the latter date we received, through the courtesy of Dr. L. H. Backeland, the following telegram:

"Our chemical special train fearful smash-up near King City. Miraculous escape of all passengers. Locomotive turned turtle, sleeping car jumping over it in air, ditching right and left. Everybody calm and self-possessed. Engineer, stoker and conductor killed. Party arrived here by next train. Otherwise trip was most splendid."

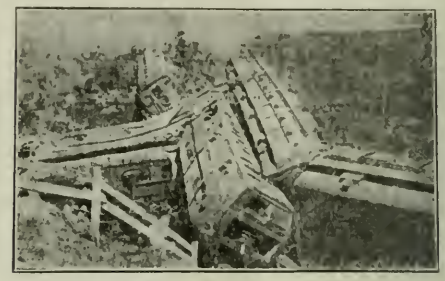
How serious the accident was may be seen from the adjoining illustrations, being reproductions from the San Francisco *Call*, the first being a photograph made by a reporter of that paper, the second a snapshot taken by Mr. J. B. Thomas, a passenger on the train.

Excessive speed in a restricted district is said to have been responsible for the wreck. Among the members of the American Chemical Society who were on the train were Mr. and Mrs.



WRECK OF TRAIN OF CHEMICAL SOCIETY.

George P. Adamson, with daughter, of Easton, Pa., Prof. and Mrs. W. D. Bancroft of Cornell, Dr and Mrs. L. H. Backeland of Yonkers, N. Y., Mr and Mrs John T. Baker of Phillipsburg, N. J., Dr and Mrs H. E. Barnard of Indianapolis, Ind., Mr W. Beardwood, Dr. Walker Bowman of New York, Mr. and Mrs. H. B. Bishop, Mr and Mrs. I. M. Bregowski of Chicago, Dr H. H. Bunzel, Dr. and Mrs. Eng. A. Byrnes of Washington, D. C., Dr Frank D. Cameron of Washington, Mr. W. E. Chamberlain, Mr. A. N. Clark of Berlin, Ontario. Miss Edna A. Clark of Washington, Mr. and Mrs. A. M. Comey of Chester, Pa., Mr. and Mrs. C. F. Crowley of Omaha, Neb., Mr. E. de Meeus, Prof. W. L. Dudley of Nashville, Tenn., Mr. L. Feldstein, Mr. Albert Finke of Cleveland, Ohio, Prof. G. B. Frankforter of University of Minnesota, Mr. L. E. Gallagher of Arlington, Mass., Prof. M. Gomberg of University of Michigan, Dr. A. P. Hallock of New York, Mr. J. F. Harbow, Mr. F. W. Hatchel, Mr. and Mrs. Elwood Haynes of



WRECK OF TRAIN OF CHEMICAL SOCIETY.

Kokomo, Ind., Mr. Morris W. Hedden of Boston, Mass., Mr. W. F. Health of Boston, Mass., Dr. C. H. Herty of Chapel Hill, N. C., Dr. W. F. Hillebrand of Washington, D. C., Prof. J. I. D. Hinds of University of Nashville, Miss Hines, Mr. W. C. Hovey of Chicago, Ill., Mr. George M. Howard of Philadelphia, Pa., Dr. and Mrs. Treat B. Johnson, of New Haven, Conn., Mr. W. M. Kelsey of Depue, Ill., Prof. and Mrs. E. F. Ladd of Agricultural College of North Dakota, Mr. and Mrs. Charles Lamont of Detroit, Mich., Miss Mary F. Leach of Western College, Oxford, Ohio, Dr. M. Le Clark of Washington, Dr. E. G. Love of New York, Prof. H. McCormack of Armour Institute, Chicago, Prof. W. A. Noyes of University of Illinois, Mr. and Mrs. M. F. Nichols, Mr. J. F. Oakleaf, Prof. and Mrs. Charles L. Parsons of New Hampshire College, Durham, N. H., Mr. A. M. Patterson of Columbus, Ohio, Mr. and Mrs. F. B. Porter of Atlanta, Ga., Miss A. P. Schaffner, Mr. R. N. Shreve of St. Louis, Mo., Prof. C. F. Sidener of University of Minnesota, Mr. and Mrs. Frank N. Smalley, Mr. and Mrs. H. R. Smith of Highland Park, Ill., Mr. M. O. Stafford of Philadelphia, Dr. T. C. Stearns of Jersey City, N. J., Mr. F. L. Steenken, of Brooklyn, N. Y., Mr. C. J. Strosacker of Midland, Mich., Mr. and Mrs. Edw. R. Taylor of Penn Yan, N. Y., Dr. Charles E. Taylor, Misses Clara E. and Alice E. Taylor, Mr. J. B. Thomas of Baltimore, Md., Dr. Willis G. Tucker of Albany, N.Y., Prof. and Mrs. George F. Weida of Kenyon College, Gambier, Ohio, Dr. W. R. Whitney of Schenectady, N. Y., and Mr. E. Wolensky of Crete, Neb.

The only members of the party of chemists injured were in the car just back of the dining car. None was hurt seriously enough to be taken to a hospital and they resumed the trip with their friends.

According to the reports received the miraculous escape of the passengers was due to all steel cars.

We reserve for our next issue a report of the San Francisco meeting itself, which turned out exceedingly successful.

The Iron and Steel Market.

The closing of steel works in July for repairs was inconsequential, a few plants closing for from 10 days to three weeks, while many did not close at all. Production, therefore suffered very little. There has only been a slight further decrease in pig iron output, which at the close of July is probably but little under 27,000,000 tons a year, representing a total reduction of about 15 per cent as compared with the maximum

rate, reached in February, and easily exceeding the average rate in 1906 and 1907.

The general tendency of buyers is to make purchases sparingly and to reduce stocks, the condition being in striking contrast to that of the fall and winter, when specifications were liberal and material was being accumulated. With this complete reversal in the attitude of buyers the flow of material is well maintained, not having decreased more than about 15 per cent at the outside. Thus the evidence is that the ultimate consumption is in excess of the current production, and is decidedly larger than it was six or nine months ago.

Indeed, the persistence of a heavy tonnage demand is remarkable, particularly when it forms such a striking contrast to the general apathy observed in market circles. Superficially, the market is intensely dull throughout; practically, there is a tremendous amount of material being made, shipped and consumed.

The trend of prices continues downward, although very moderately. Pig iron has been almost stationary in the past month, while prices of finished steel products have yielded only at a few points, and then only by a small amount.

On July 14 the American Steel & Wire Company began quoting plain wire at $1.50, base; wire nails at $1.70, base; painted barb wire at $1.70, and galvanized barb wire at $2. These prices are $3 per ton below the prices as advanced Dec. 13 last, but the actual reduction at the moment in open-market prices was only $1 a ton. In February shading of $1 a ton appeared, and this was increased to $2 about the beginning of July, this latter shading being only by the independents, the leading interest finally recognizing the condition and cutting an extra dollar so as to establish a market.

The shading in sheets has increased from $1 to $2 a ton since last report, and now amounts, in extreme cases, to $3 a ton on black and $4 a ton on galvanized products.

In plates, shapes and bars prices are unchanged, but the expectation is for lower rather than higher prices. Merchant steel pipe is strictly held, but this is due to demand being slack rather than to its being heavy, for the jobbers still have large stocks and the mills wish to protect them, realizing also that cut prices at this time would not bring out more business.

On July 19 an agreement was reached between the Republic Iron & Steel Company and the Amalgamated Association of Iron, Steel and Tin Workers relative to wage scale for the year beginning July 1. The Republic company operates a number of iron mills, which are under Amalgamated control, while a few of its steel rolling mills are similarly controlled, its large steel plant and other operations being non-union. The scales adopted make an advance of 50 cents a ton in puddling as long as bar iron is 1.25 cents per pound or higher, i.e., the base of the scale remains $5 per ton, but advances to $5.12½ on 1.05-cent bar iron instead of on 1.25-cent bar iron, subsequent advances being 12½ cents in puddling for each 1/20 cent in bar iron. The last bimonthly adjustment showed that the average realized price on base sizes of iron bars shipped from the selected Ohio mills in May and June was 1.46375 cents, which calls for $6.12½ to be paid in July and August, while the expiring scale would call for $5.62½. The settlement is a restoration of the 1907-8 puddling scale. The other iron mills scales were advanced an average of about 4 per cent, falling short of restoring the 1907-8 scales. The Western Bar Iron Association, composed chiefly of independent mills in the West, has accepted the Republic settlement.

Pig Iron.

Apparently pig iron prices have closely approached bottom in the sharp decline which has been in progress since last November, for the settling since last report has been very slight. Valley and Birmingham prices, indeed, are not quotably lower, but there have been slight declines along the Lakes and in the East. The market has been extremely quiet. Consumers are buying only for immediate requirements and are curtailing stocks in many cases. The merchant furnace production has been steadily reduced, but is still rather heavy in comparison with actual market demand. At the close of July, 11 of the 21 merchant furnaces in the Mahoning and Shenango valleys are in blast, and some iron is probably being piled. The time was that "the valleys" occupied a strategic position of great value, being able to distribute iron in all directions, but furnaces have grown up around them in a circle, greatly restricting their territory, and since the abrupt cessation of buying of Bessemer iron by the large steel works, late last year, the valley position has been poor. While $11.50, Birmingham, is quotable as the market for delivery through the second half there is some shading from this level. Valley prices remain quotable as follows, freight to Pittsburgh being 90 cents: Bessemer, $15.50; basic, $14.50; No. 2 foundry, $14.50; forge, $13.75; malleable, $15.

Steel.

There has been a sharp decline in open-hearth billets, which have become available at $26, Pittsburgh, $1.50 under last quotation, while Bessemer billets are easy at $25, Pittsburgh, having been firm at $25 to $25.50 a month ago. There has been another curious reversal in the position of raw materials for steel making. In the past heavy melting steel scrap has regularly sold at less than basic pig iron, delivered at works in the Central West. At the beginning of 1908, on account of the sudden slowing down of the whole industry leaving a large tonnage of material in transit, prices were particularly low, and predictions were then freely made that open-hearth billets would ultimately sell at less than Bessemer. Last year, on the rapid increase in operations, there was developed a corresponding scarcity of scrap, and prices passed far above pig iron. The pendulum has swung again in the opposite direction and melting steel is more than a dollar a ton below basic pig iron, delivered in the Central West, or $2 a ton below Bessemer. The industry has not yet reached the point where cost of production dictates relative prices, but the trend is distinctly in the direction of establishing a cost-of-production relation between prices of Bessemer and open-hearth steel. We quote the market, Pittsburgh, as follows: Bessemer billets, $25; sheet bars, $26 to $26.50; open-hearth billets, $26; sheet bars, $27 to $27.50; rods, $29 to $29.50.

Finished Products.

Regular prices, f.o.b. Pittsburgh, are as follows, subject to shading of $2 to $3 on black sheets and $3 to $4 on galvanized:

Standard steel rails, $28 for Bessemer, $30 for open-hearth, f.o.b. mill, except Colorado.

Plates, $1.40 to $1.45 for tank quality, ¼-in. and heavier.

Shapes, $1.40 to $1.45 for beams and channels, 15 in. and under, zees and angles 2 in. x 3 in. and larger.

Steel bars, $1.45, base; hoops, $1.50 to $1.60; iron bars, $1.50, Pittsburgh; $1.40, Philadelphia; $1.40, Chicago.

Wire nails, $1.70, base; plain wire, $1.50, base; painted barb wire, $1.70; galvanized barb wire, $2.

Black sheets, 28-gage $2.40; galvanized $3.50; blue annealed, 10-gage, $1.75; painted corrugated roofing, 28-gage, $1.70 per square; galvanized, $3; tin plates, $3.60 for 100-lb. cokes.

The Non-Ferrous Metal Market.

The market has been practically without feature during the month. Slight fluctuations, either up or down, have not shown any permanency, and the prices have returned to a level from which it seems impossible to move them.

Copper.—A rather large volume of business has been transacted owing to the fact that manufacturers have remained out of the market so long that their stocks were low. Considerable activity has been noticeable in electrolytic, especially for August and September delivery. Lake copper has been quoted at 12⅝ cents to 12¾ cents, with electrolytic at 12⅛ cents to 12⅜ cents.

Tin.—Only such business as is imperative is being transacted. Few purchasers for future delivery are being made and most manufacturers are inclined to buy for immediate needs only.

Lead.—The market is rather quiet, but prices have ruled firm at about 4.25 cents to 4.27 cents at St. Louis, and 4.35 cents to 4.40 cents at New York. Buying has been without feature, and business is nominal from day to day.

Zinc.—Buying has been sparing on the part of manufacturers in the galvanizing business and as a consequence the market has been very quiet. Prices have eased off to 5 cents and 5.05 cents, St Louis, and 5.15 cents to 5.20 cents, New York.

Other Metals.—A comparatively small volume of business is being transacted in aluminium, antimony, mercury, etc., but the prices remain unchanged. Aluminium is quoted at 23¼ cents per pound, New York; antimony, 8.15 cents to 8.20 cents per pound for Cookson's; mercury, $47 to $49 per flask of 75 lb., according to quantity taken.

The Western Metallurgical Field.

Sampling Spelter.

The inaccuracy of the drilled sample as compared with the granulated sample in spelter analysis was clearly shown at a recent meeting of the Colorado Section of the Western Association of Technical Chemists and Metallurgists, when Mr. E. M. Johnson presented some results that had come under his supervision. Both drilled and granulated samples from the same lot will show a variation in the lead and iron content, but the variation will not be so great in the latter as in the former. For instance, in drilled samples from one lot the variation in iron was 0.008 per cent, while in granulated samples it was only 0.004 per cent. In another lot the variation in iron was 0.036 per cent in the drilled samples, while it was but 0.02 per cent for the granulated.

With regard to lead the difference is even more marked. In one lot the greatest difference in the drilled samples was 0.32 per cent, while with granulated samples it was but 0.09 per cent. In many other tests the same difference in favor of the granulated sample was noted. In addition to the fact that the granulated sample is more accurate, it has other advantages pointed out by Mr. Johnson. First, the metallurgist has a much better control over his spelter, as he can get results sooner. This may save rehandling the spelter, if immediate shipments can be made. Second, the cost of drilling samples amounts to 75 cents or $1 per day, which is eliminated with granulated samples, as they are usually taken by the furnace attendant or foreman. Another point in connection with granulated samples is that when properly taken they go into solution more readily than do the drilled samples.

Bureau of Mines.

Western interests have not been satisfied with the delay in appointing a permanent director of the Bureau of Mines lately created. There is a tendency to regard the appointment of George Otis Smith, present director of the Geological Survey, as temporary director of the bureau, as a mere postponement of an issue which has been evident from the beginning, viz., whether an ardent friend or an opponent of the bureau shall be its head. Unquestionably, if the bureau is to amount to anything and perform the duties for which it was created, it must be directed from the beginning by one who is friendly to its useful existence. The consensus of western opinion is that Joseph A Holmes, former director of the technologic branch of the survey, is the logical man for the place and it is hoped that when the President finally makes his appointment Dr. Holmes will be found at the head of the new bureau.

American Mining Congress.

The official call for the thirteenth annual convention of this congress has been issued. The convention will be held at Los Angeles, Cal., from Sept. 26 to Oct. 1. The meetings will provide opportunity for the discussion of the production and treatment of useful minerals, conservation in its relation to the mining industry, uniform mining laws in the different States, and other questions that concern mining and metallurgical progress. A large attendance is expected and arrangements for a good meeting are now under way. The program will be announced later.

International Company's Tooele Smelter.

August will probably witness the blowing in of the new copper smelter at Tooele, Utah. Five reverberatory furnaces will constitute the initial smelting equipment of the plant, and these are rapidly nearing completion. The roaster equipment has been installed for some time and the balance of the plant has been tested so that everything is in readiness for operation. The principal ore supplies will come from the mines of the Utah Consolidated at Bingham and the South Utah, formerly the Newhouse Mines & Smelters Corporation, at Newhouse. The ore from the former will be delivered by a tramway recently completed by the Trenton Iron Company. The contract with the South Utah company is more recent and will provide for the shipment from Newhouse of about 200 tons of copper concentrates per day, beginning with the first of August. The Tooele plant will represent the highest state of the smelting art and its operation will be watched with considerable interest.

Colorado.

It is stated that in June, 1910, Colorado passed the billion-dollar mark in metal production. Be that as it may, it is certain that the industry in this State is not in as flourishing a condition as it should be, and steps have been taken to look into the causes of the decline. Colorado is not willing to relinquish her position as one of the leading mining States, despite the fact that her smelters and mills are either idle or run at reduced capacity. At a meeting of mining men held at Denver on July 14 to consider the mining and metallurgical situation three phases of the matter were discussed, viz., ore treatment and transportation rates, the deficiency of present metallurgical processes and the relations of mining and the forest service.

The rates on ore treatment and transportation were regarded as one of the causes of the decline of mining in the State. It was pointed out that ore in certain districts had so changed in its metal content that the smelting charges at present in force made it impossible to mine the ore at a profit. Freight rates were considered excessive, especially as compared with rates in other States where ore is hauled over mountain railroads with heavy grades. A reasonable reduction in both freight and smelting charges would provide mining expenses and permit much ore to be mined and treated that must now remain untouched.

The fact that progress in metallurgy has not kept up with the requirements of mining has been mentioned before, and is undoubtedly a valid reason for the present unsatisfactory state of mining, not only in Colorado, but in other Western States. There are many ores the gross metal of which is sufficiently high to warrant its treatment if all, or nearly all, of the values could be recovered, but in our present state of metallurgy a complex ore suffers losses in the way of by-products and treatment charges that make it unprofitable. It is firmly believed by many metallurgists that the process of the future will be chemical, and that it will recover practically all of the valuable constituents of the ore. Steps in this direction are already in evidence and the day of their success is awaited with interest.

The present relations between the Departments of the Interior and Agriculture, whereby prospecting on the public domain has been subject to unnecessary restrictions, were blamed in large measure for the failure to develop new mining districts. Formerly it was necessary only that a miner should perform his annual assessment work to secure title to a mining claim, but under the regulations of the forest service a prospector on the

forest reserves, which cover much of the most attractive ground in the State, must submit to the opinion of the ranger as to whether he has a valid and valuable mineral discovery. It is believed that these restrictions have been one of the reasons why no important new mining districts have been discovered and exploited in Colorado for many years. A change in the regulations is gradually being made, however, and it is believed that in a short time the regulations will not be heard of.

Partridge Hot Blast Furnace.

A demonstration furnace of this type has been established at the old Denver Paper Mills, near Denver, Colo. The Partridge furnace is the invention of A. R. Partridge, who has erected a number of plants in Mexico and a few in the United States. The main features of the furnace are a hearth, or crucible, which can be lowered from the shaft; an opening of about 3 in. or 4 in. between crucible and shaft when the former is in position, this opening serving in lieu of tuyers for the admission of the light blast, and finally a system of heating the blast with the waste heat from the slag by discharging the slag into an endless conveyor which travels in a steel housing through which the blast is directed. The air blast passes through this inclosure surrounding the slag conveyor and is finally delivered to a wind box surrounding the furnace between the bottom of the shaft and the crucible where it enters the former. The inventor claims for his furnace simplicity of design and operation, and adaptability to the treatment of lead or copper ores, producing either bullion or matte. The demonstration furnace at Denver is attracting some attention and is in operation almost daily on various classes of ore.

Goldfield Consolidated.

The June report of the Goldfield Consolidated Mines Company, Goldfield, Nev., shows some interesting metallurgical results. The mill treated 20,091 dry tons of ore of an average value of $45.27 per ton. The total value of the ore was $909,480; loss in tailings, $41,216; value received, $868,264; percentage extraction, 95.47. The extraction of the mill was unusually good, and in this connection it is reported that the tailings loss included more than 300 ounces of gold in concentrate residues. These residues are usually shipped to the smelters, but in June more than half the concentrate residues were too low-grade to stand the cost of shipment and were thrown on the dump. Practically all of the damage caused by the fire of last April has been repaired and the mill is nearly in condition to handle its full capacity of 860 tons daily. At present it would be impossible to increase the tonnage milled on account of the shortage of water, but this is expected to be but a temporary condition. In all other respects the plant is in good condition.

Corrosion of Non-Ferrous Metals.

The Council of the Institute of Metals in London have decided to undertake an investigation into the causes of the corrosion of non-ferrous metals by sea water, acids, etc., and by other chemical and electrolytic reactions.

A committee, known as the Corrosion Committee of the Institute of Metals, has been appointed for the purpose of carrying out the investigation.

The following are the names of the gentlemen constituting the committee:

Sir Gerard Muntz, Bart., chairman of the committee (president of the Institute of Metals).

Prof. H. C. H. Carpenter, of the University of Manchester.

Capt. G. G. Goodwin, deputy engineer-in-chief, Royal Navy.

Prof. A. K. Huntington, King's College, London.

J. T. Milton, chief engineer surveyor, Lloyd's Register of British Shipping.

A. Philip, Admiralty chemist.

Leonard Sumner, of the Broughton Copper Company, Ltd.

Prof. T. Turner, of the University of Birmingham.

Sir William H. White, past-president of the Institute of Metals.

The first meeting of the committee was held at the offices of the Institute, Caxton House, Westminster, S. W., on May 24, all the members being present with the exception of Sir William White, who had just left for a visit to the United States of America.

Realizing the magnitude of the task before them, and the vast number of ramifications to which such an investigation must lead, the committee have decided in the first instance to confine their attention to the question of the corrosion of condenser tubes in marine engines and in stationary engines using foul water or being subject to violent electrolytic action, such as often occur in electric power stations.

Before commencing experimental research the committee propose to have prepared an epitome of such information on the subject as is already available from previous research in the same direction. Mr. G. D. Bengough, of the Metallurgical Department of the University of Liverpool, has been asked to undertake this work and to become a member of the committee.

CORRESPONDENCE.

Rapid Electrodeposition.

To the Editor of the Metallurgical and Chemical Engineering:

Sir—I notice in your issue of May a communication by Allen P. Ford referring to my paper with Mr. Hughes, in which we describe some new forms of rotating electrodes for the quantative deposition of metals. Mr. Ford thinks it strange that in our paper we made no mention of the article by Dr. J. T. Stoddard in which he gives interesting information as to the merits of stationary wire gauze cathodes. The reason for not noticing the paper of Dr. Stoddard and also of other investigators along the same lines was because the paper was simply dealing with electrodes which had been gradually evolved in my laboratory.

I am myself by no means wedded to the use of rotating electrodes and am inclined to think that rather too much is made of rapid methods of deposition, although I know I have been rather a sinner in this way myself. I am not sure, but I rather believe, that I was the first to use rotating electrodes, my first one having been made in 1899, although I never published that particular form.

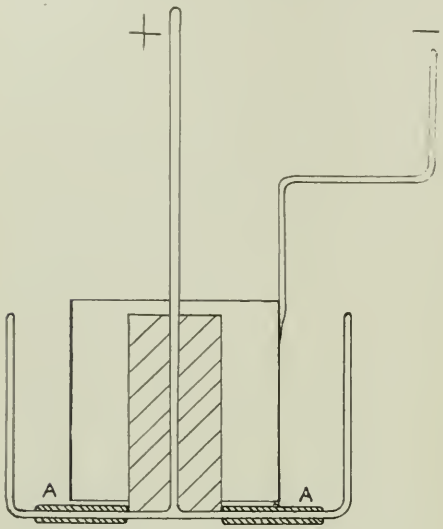
The motive power was clockwork and the rotation was four revolutions per minute. The anode was rotated and consisted of stout platinum wire (10 B. W. G.) which was bent as illustrated in the figure and having a piece of sheet platinum fastened on to the upright to cause a swirling of the electrode. The cathode consisted of the old style of cylindrical electrode. At *A A* the anode was insulated by means of pieces of glass in order to prevent uneven deposit on the edge of the cathode. In fact, it was because at that time I was using the cylindrical sheet cathode that this anode was devised so as to give even deposition inside and outside the electrode. By means of this electrode the time of deposition was more than halved and higher current densities could be employed without "burning." I believe that by using rapid rotation and substituting gauze for the plain cylinder that this apparatus would work very well. The chief difficulty is to obtain rigidity.

Very shortly after this I devised a rotating gauze cathode which was described before the first meeting of the Faraday Society. Since then I have experimented with a large number of different types, and the paper read before the Faraday Society in January was the outcome of this work.

At the same time, however, I have, for a great deal of my work, used a stationary gauze flag electrode, which I also described at the first meeting of the Faraday Society in June, 1903. I have at times used a gauze cylindrical cathode with a spiral platinum wire as anode, and I agree that very rapid deposition does take place, but I have not succeeded in deposit-

ing out the whole of the metal quite so rapidly as Mr. Ford has.

The drawback to very rapid methods is that they require so much attention. Personally, when possible, I much prefer to get a number of solutions ready and electrodes weighed, and



ROTATING ELECTRODE ARRANGEMENT

allow the electrolysis to take place over night. There are, of course, cases, notably that of bismuth, where it is by no means easy to obtain adherent and good deposits with stationary electrodes, and in such cases the rapid methods should be employed.

London, England. F. Mollwo Perkin.

* * *

Temperatures of Electric Steel Refining.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—The chief advantage of the electric furnace over the open-hearth for steel refining is generally considered to be due to two causes—the higher temperature and the neutral atmosphere. However, with respect to the higher temperature this statement should be somewhat modified. It is true that it is quite possible to produce much higher temperatures in the electric steel furnace than in the open-hearth furnace, and as a matter of fact there exist locally much higher temperatures right below the arcs. In general, however, contrary to the usual assumption, it is a fact that the average temperature of the electric steel furnace is not much higher than that of other metallurgical furnaces.

I may first refer to your article in the April issue on the 15-ton Héroult furnace at the South Chicago Works of the U. S. Steel Corporation, giving some temperature measurements of Prof. Jos. W. Richards with the Mesure-Nouel pyrometer which are so low as to show that the electric steel was in the average not hotter than the steel in the converter. The reason for this might be found in the fact that in the South Chicago works the bessemer metal is refined for only one to two hours in the electric furnace and that only 200 kw-hours are supplied to the steel per ton, so that the refining process yields only a good average quality (0.4 C, 0.4 Mn, 0.03 S and 0.03 P), but I have made myself similar temperature measurements with a Wanner pyrometer in a 5½-ton arc furnace in which steel of the highest quality was produced, and have found almost exactly the same average figures of temperature as Professor Richards. The suggestion of Dr. Amberg in your June issue (page 314) that the results were doubtful because the Mesure-Nouel pyrometer was unreliable, is, therefore, disproved.

The furnace which I tested was a Nathusius furnace of 5½ tons capacity, which has been in operation on the Friedenshutte in Upper Silesia for a year.[1]

Converter steel was charged into the electric furnace and refined to a very soft steel containing 0.5 to 0.06 C and very small amounts of sulphur and phosphorus (0.003 to 0.02 S and 0.004 to 0.017 P). The treatment lasted for 3½ to 4½ hours and the energy consumption per ton of steel was 300 to 400 kw-hours.

The converter metal when poured into the smaller ladle had still a temperature of 1450° C. to 1470° C. When poured from the ladle into the electric furnace the temperature was 1435° C. to 1450° C. The interior of the furnace (immediately after the pouring of the finished steel) had a temperature of 1535° C. to 1560° C., the points of the electrodes, when being withdrawn, being at 1300° C. to 1350° C. The temperature of the electric steel when poured was 1500° C. to 1520° C. for a soft charge and 1558° C. for a high silicon charge. In the latter case the temperature when pouring into the ingot was still 1500° C.

Since the electric furnace had always to wait a certain time for a charge of fresh bessemer metal, the interior of the furnace cooled to about 1000° C. After the metal had been charged into the furnace and the reaction was completed, the temperature of the slag layer was measured and was found to be, after an hour, 1550° C., and after an hour and a half, 1600° C. to 1650° C. It remained at 1600° C. to 1650° C. until the slag was removed, when its temperature was found to be between 1600° C. to 1560° C. The removal of this first slag and the introduction of the dephosphorizing slag caused the bath to cool off somewhat to about 1550° C. and the temperature remained in the second period of the treatment a little less than in the first period.

Concerning the Roechling-Rodenhauser induction furnace I have no numerical data of the temperature, but skilled open-hearth furnace engineers estimated the average temperature not to be much higher than in the open-hearth furnace.

What do these figures prove? They show evidently that an abnormally high temperature is not necessary for the refining process. As a matter of fact, even the heat of the arc for the local heating of the slag is not absolutely necessary, since it is generally known that the Roechling-Rodenhauser furnace permits the same far-reaching refining reactions to be carried out down to traces of sulphur and phosphorus without any arc.

The refining possibilities of the electric steel furnace are, therefore, certainly due to a larger extent to the non-oxidizing atmosphere than to a very high temperature. There is no doubt that the possibility of producing a higher temperature in the electric furnace helps the refining process considerably, since the higher basic slag can be easily kept fluid, but as mentioned above the arc furnace is not absolutely necessary for this purpose.

On the other hand, it is quite self-evident that at the places right below the electrodes the temperatures are much higher than the above average temperatures. The proof of this is the formation of calcium carbide, which has often been observed in the arc furnace and which according to the latest investigations (Metallurgical and Chemical Engineering, 1910, pages 279 and 328) begins only at about 1820° C. and which does not occur in the induction furnace. Further, the electric steel furnace offers the possibility of heating the metallic bath for special purposes to a much higher temperature than is possible in other furnaces. I have myself seen steel tapped at temperatures above 1650° C., but in the average the mean temperature required for the almost complete removal of phosphorus and sulphur is not much higher than in other metallurgical furnaces.

Bernhard Neumann

Institute of Technology, Darmstadt, Germany.

[1] For further description of this furnace see the abstract of the paper of Prof. Neumann in the report of the International Congress of Mining and Metallurgy, printed elsewhere in this issue.—Ed.

Kr. Birkeland.

Kr. Birkeland, Norwegian physicist, was born in Christiania. in the year 1867, and is now professor of natural philosophy at the University of Christiania.

He received his first scientific education at this university, and later he studied under Poincaré, Picard, and Appel, in Paris, and under Hertz in Bonn.

The first work of his to attract notice concerned a complete solution of Maxwell's equations for the electromagnetic field; he had already carried out a series of investigations on electrical vibrations and on discharges through gases.

He was the first in Norway to take Röntgen photographs, and did so only a few weeks after the incomplete report of the X-rays was received, and with apparatus made by himself, which is a proof of his keen attention in all fields and of his wide scholarship. He then threw himself with great energy into the study of cathode rays and investigated in particular the deflexion of these rays under magnetic influence.

It is an indication of his scientific imagination that he was not satisfied with a detailed investigation of these rays, but soon perceived what immense importance they possessed in the energy of the universe. He especially directed his attention to the sun and in this connection enunciated his well-known theory of sun spots.

These investigations led him to a theory of the northern lights (aurora borealis) which he explained as produced by cathode rays or related rays emanating from the sun and absorbed toward the earth's magnetic poles.

It is an interesting trait of his scientific personality that he was not content with forming a general theory of northern lights, but submitted it to a searching test, both in the physical laboratory and by exact mathematical analysis; in the latter respect his colleague, Professor Störmer, has supported his theory of the northern lights in an effective manner. These calculations showed an astonishing agreement with the laboratory experiments, in which a little magnetic model of the earth in a glass vessel, exhausted of air, was exposed to the effects of of cathode rays.

Kr. Birkeland.

Moreover, he tested his theories in Nature herself, for he undertook no less than three arduous expeditions to the northern regions where the northern lights show especial strength and characteristic appearance. One of these expeditions extended to stations in Iceland, Spitzbergen, Novazembla, and Finmarken, so that by collective observations at the various places the value of the observations might be increased.

In connection with the study of these phenomena Birkeland has been occupied with another great task—the explanation of the earth's magnetism, its variations, daily and yearly, in addition to magnetic storms. Here also Birkeland has shown himself not only a scientific genius, but also an experienced and rigorous experimenter," for without experimental confirmation even the best theory will be weak and vague.

He has, in short, proved his theory with the help of a large number of co-operative observations taken at no less than 25 stations throughout the entire world, and the mathematical utilization of these observations has required an enormous amount of work necessitating the aid of a whole staff of scientific coworkers and the expense of large sums of money. These expenses have for a large part been met by Birkeland himself. Such a sacrifice for science is rare, the more so as he was by no means a wealthy man.

In fact, he procured the money necessary for these purposes by putting aside his purely scientific studies for a period of five years and devoting himself to one of the greatest problems with which chemistry is concerned at the present day, namely, the production of nitrogenous compounds from the atmosphere by the aid of electricity.

After having succeeded with the imagination of genius and with energetic work in solving this problem even as to the details of the first practical method of manufacture ever carried out, and having obtained the money he needed for the abovementioned scientific objects, he again set to work with all his force in the service of science.

Birkeland had noticed during one of his experiments that an electric arc of high potential in a strong magnetic field assumes the form of a flat disk of fire, possessing tremendous heat and power of chemical combination. He spoke about this discovery to an engineer, Mr. Sam Eyde, who had read about the American attempts to produce nitric acid from the atmosphere with the aid of electricity, and the idea arose of using the said disk of fire for the same purpose. Experiments that were then carried out confirmed the feasibility of the scheme, and the systematical development of the process led to the formation of great companies which have factories in Norway for the production of nitric acid from the atmosphere.

The Norwegian Hydroelectric Company (capital 41,166,250 francs, or $8,233,250), which operates under the Birkeland-Eyde patents, started in 1908 the Notodden-Svolgsfos factories with 40,000 hp. This company is also building factories at Lienfos (15,000 hp) and Wamma (55,000 hp). The company has also combined with the "Badische Anilin and Sodafabrik," which has elaborated its own methods in later years. These united companies are now constructing factories at Rjukan (227,000 hp), Tyin (81,000 hp), and Matre (83,000 hp).

Professor Birkeland has been elected a member of the Scientific Society of Christiania, he is a doctor honoris causa of the technical high school in Dresden, and an honorary member of the Royal Society of Arts and other learned bodies.

He is undoubtedly the greatest living scientific investigator of Norway, possibly of Scandinavia. He possesses a deep and broad scientific education, an unusually rich imagination and forceful synthetic power; he is a scientific genius with the happy gift of thinking out and carrying through the experiments that are to prove or disprove his theories. He has, in addition, stored in him unlimited energy, which he entirely devotes to the solution of scientific problems—and he permits no obstacles to deter him from his task.

C. N. Rüber.

Smoke and Fume Problem in Bayonne.—According to the New York *World*, Attorney-General O'Malley, acting upon orders from Governor Hughes, of New York, is to bring action in the Federal Court against the Standard Oil Company, the Bergenpoint Chemical Company, the General Chemical Company, the Orford Copper Company, and other chemical companies operating in Constable Hook, Bayonne, N. J. Constable Hook is directly opposite to Staten Island, which is a part of New York City, and it is in Staten Island that the damage is alleged to have been done by the smokes and fumes from the chemical and metallurgical works of Bayonne. Since the latter are in the State of New Jersey the legal questions involved are interesting.

The Strength of the Alloys of Electrolytic Iron and Monel Metal.

By Charles F. Burgess and James Aston.

For several years past there has been conducted at the Chemical Engineering Laboratories of the University of Wisconsin, under grant of the Carnegie Institution of Washington, an extensive series of tests on the properties resulting from the addition of various alloying elements to electrolytic iron. As part of the tests, a study was carried out upon the strength of the alloys of this iron with varying proportions of nickel and of copper.

The results of the nickel series were published in Bulletin No. 346 of the University of Wisconsin,[1] and those of the copper series were presented before the American Electrochemical Society in October, 1909,[2] and reprinted in the above-mentioned university bulletin. While this work may in large measure be classed as supplementary or correlative to the work of other investigators,[3] it was considered well worth undertaking in view of the high degree of purity of the electrolytic iron, and the consequent opportunity of ascribing any observed effects solely to the alloying element, and the elimination thereby of one question of doubt arising in most investigations of the alloy steels.

The results of these earlier tests were largely confirmatory of the work of the other investigators mentioned, the chief differences noted in our tests being a shifting of the regions of certain physical properties because of variations of carbon content. The general conclusions may be stated as follows:

Iron-Nickel Alloys.

1. With increase of nickel there is an increase of strength with a slight decrease of ductility in the range of lower nickel content.

2. Beyond this range the addition of nickel causes a sudden increase of strength with a marked decrease of ductility over a zone of decided brittleness.

3. The position of the brittle zone varies with the carbon content and probably with a variation of other impurities, such as manganese, if present in appreciable amounts. For a pure alloy where the effect is due to nickel alone, the area may be set between 10 per cent and 16 per cent of nickel. The addition of carbon places the zone at lower percentages of nickel, as low as 7 per cent nickel for carbon = 0.82 per cent, or even as low as 5 per cent if the manganese rises to 1 per cent.

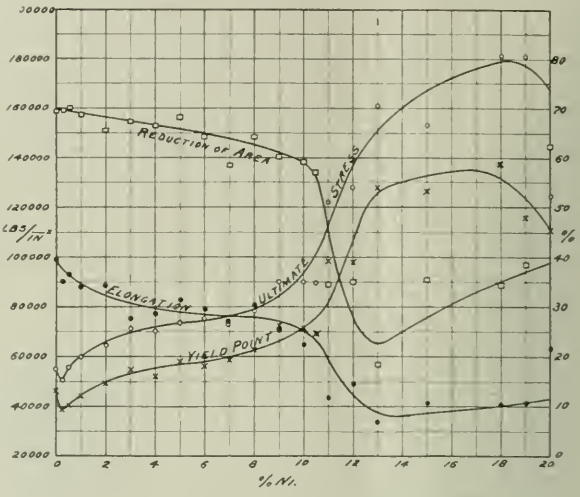


FIG. 2.—TENSILE STRENGTH IRON-NICKEL ALLOYS ANNEALED.

4. Accompanying this brittleness there is a marked hardening in the material.

5. Annealing, while not greatly affecting the region of the brittle zone or the extent of the brittleness, has a tendency to confine the range to more narrow limits.

6. For percentages of nickel above those of the zone of brittleness there is a restoration of the ductility and softness.

The data of this series are given briefly in Tables 2 and 3, and plotted in Figs 1 and 2.

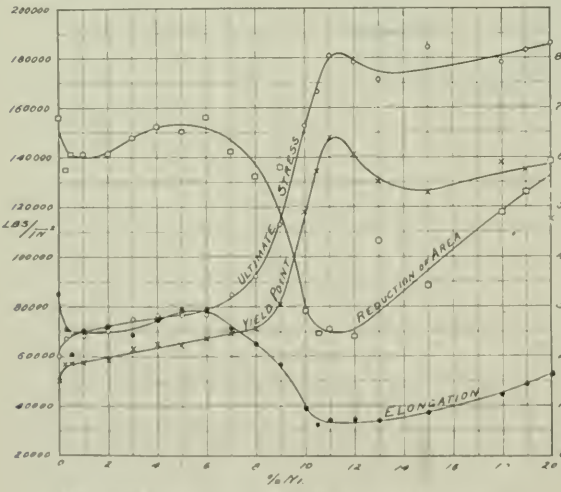


FIG. 1.—TENSILE STRENGTH IRON-NICKEL ALLOYS ANNEALED.

Iron-Copper Alloys.

The results are tabulated in Tables 2 and 3 and plotted in Figs. 3 and 4. They are in disagreement with the generally accepted opinion that copper is a very deleterious impurity in steels. It seems evident that the detrimental effect commonly noted is due rather to the combined influence of the copper and some of the elements met with in commercial steels, notably carbon and sulphur. In the absence of carbon, unannealed samples showed a successive increase of tensile strength up to the maximum copper addition of 7 per cent (the limit of forgeability). The yield point was close to the ultimate strength, and the elongation fell off with regularity. Brittleness, therefore, was the accompaniment of the copper additions, and increased with the percentage present.

Annealing had a most marked effect, lowering the ultimate strength and elastic limit throughout, but especially for copper percentages above 1. The elongation became very good throughout the entire range. The result was, therefore, an alloy of very good properties, and this fact was brought out in the paper previously referred to. We mentioned particularly the addition of 1½ per cent of copper as being advantageous, because this small amount of addition agent resulted in prop-

[1] The Strength of the Alloys of Nickel and Copper with Electrolytic Iron. C. F. Burgess and James Aston. *Bulletin* Univ. of Wis., Eng. Series Vol. 6, No. 2, March, 1910. (See Book Reviews in this issue.—Editor.)

[2] Physical Properties of Iron Copper Alloys. C. F. Burgess and J. Aston. *Trans.* Am. Elec. Chem. Soc., Vol. XVI, 1909.

[3] Nickel Steels. R. A. Hadfield, Inst. Civ. Engrs., 1898. Vol. CXXXVIII, R. Guillet, *Bul.* Soc. d'Encour., May, 1903. 7th Report Alloys Research Committee, Inst. Mech. Engrs., Vol. 2, 1905. Copper Steels. P. Breuil, *Jour. Ir. & St. Inst.*, No. 2, 1907.

erties almost as good as those obtained with larger quantities of copper, and approached the results reached with the customary nickel content.

Iron-Nickel-Copper Alloys.

The present paper is the direct outcome of the observations noted in the two binary series of alloys just discussed; its inception was due to reasons expressed in the above-mentioned university *Bulletin*, in which some preliminary work was reported and which reasons may be quoted verbatim here.

"In view of the well-known beneficial effect of the addition of nickel to iron, and because of the very great and rather unexpected increase of tensile strength observed, in our tests of the copper-iron alloys, due to increasing percentage of copper, it was thought advisable in order to round out this research to carry out tests on a series of alloys in which both nickel and copper were added to the electrolytic iron.

"The advisability of the investigation was prompted by several considerations—first, the benefits observed by the separate additions of nickel and copper might be coupled in their joint use; second, the close relationship of copper, nickel and iron in chemical and physical properties might lead to interesting results in a ternary alloy; third, (and this was the reason of greatest weight), there is on the market a nickel-copper alloy which would make a very desirable addition agent, in case the simultaneous presence of both copper and nickel gave a tensile strength no less than that observed for the two separate alloys.

"We refer to Monel metal, resulting from the reduction of certain ores from which, by smelting alone ,this alloy is formed. The main constituents are nickel and copper, in the proportions of about three to one, respectively. The material is, for the present at least, obtainable at a price about equal to that of copper, and very much below the cost of nickel. Here, then, is a means of getting nickel additions, provided the copper carried does not neutralize the effect of the nickel."

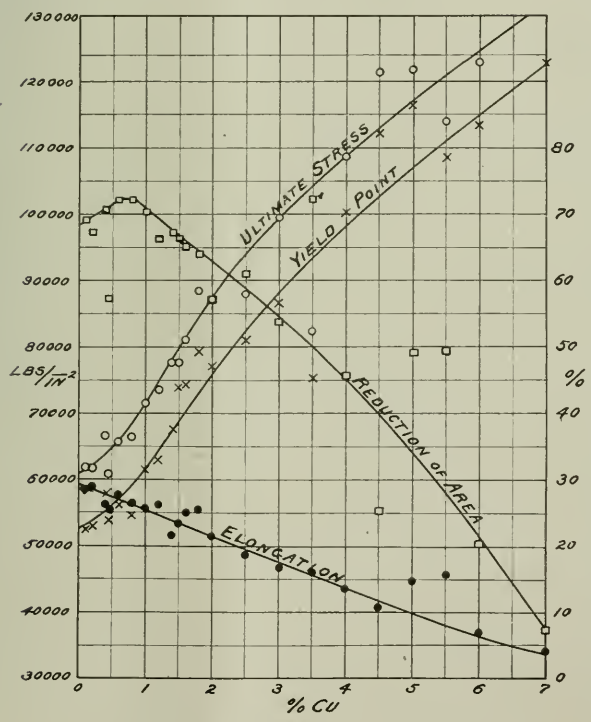


FIG. 3.—TENSILE STRENGTH IRON-COPPER ALLOYS UNANNEALED..

In the preliminary study no Monel metal was available for use. The alloys were made up by additions of nickel and copper up to a total combined content of 8 per cent. The Monel ratio of three nickel to one copper was adhered to, and, strange to say, the proportion of 1½ per cent of copper to 4½ per cent of nickel gives the values approximately the best observed in the separate alloys. The results of this work, given in Tables 2 and 3, and in the dotted curve of Plates 5 and 6, were sufficiently encouraging to warrant carrying the investigation to higher total addition content, and with Monel metal as the alloying agent.

Some particulars regarding Monel metal are given below:

Analyses.		
Nickel	66.90	67.96
Copper	24.35	26.00
Iron	5.00	2.80
Manganese	2.18	1.62

Mechanical tests gave for cast samples ultimate strengths of 30,400 lb. and 35,000 lb. per square inch. For rolled samples the figures were: elastic limit, 74,400 lb. and 79,000 lb. per square inch; ultimate strength, 100,000 lb. and 104,000 lb. per square inch.

Thus we see that the alloy has very good inherent mechanical properties, and the analyses indicate, besides the nickel and copper, only iron and manganese, and the latter is no detriment to steels.

The iron used was of a high degree of purity, obtained by a double electrolytic refining of Swedish iron. The electrolytic iron may be considered 99.97 pure. In the preparation of the alloys the brittle iron was broken into small pieces and melted in an electric furnace of the resistor type with the predetermined amounts of Monel metal. Specially prepared magnesia crucibles were used, these in turn being protected by a graphite jacket; and contamination of the charge was prevented by a magnesia lid luted on the crucible and a graphite cover for the jacket.

During the melt there was no agitation of the charge, the resulting alloy being entirely due to diffusion of the elements. This was facilitated by holding the charges above the melting temperature for a few hours, and by slow cooling in the furnace. The ingots weighted about 500 grams (1 lb.), and after forging, each furnished enough material for three test bars approximating 0.4 in. finished diameter over a free length of 2 in. Ingots were made up with the Monel metal increasing by gradations of 2 per cent to a maximum addition of 20 per cent. Samples were tested as forged and after annealing at 900° C.

The tests were made upon a Riehle 50 000-lb. power-driven machine. The load was applied very slowly, and the yield point was determined by means of multiple dividers. This was checked by the drop of the beam.

From 2 to 12 per cent of alloying addition, the ingots forged readily over a wide range of heats; from 12 to 18 per cent high temperatures were necessary and the bars did not forge well below bright red heat, while for those ingots with 18 and 20 per cent of Monel metal a white scintillating heat was necessary. Up to and inclusive of the 12 per cent addition, the test bars were machined in the lathe; above this grinding was resorted to, the material being too hard for efficient working with lathe tools.

Tensile Strength Tests.

The summary of results is included in Table 1 and in Figs.

5 and 6. There is noted the yield point and ultimate strength in pounds per square inch, the percentage of elongation in 2 in., and the percentage of reduction of area at fracture. There are included also the maximum, minimum and average values of each of the above items, together with the number of samples from which such average is calculated. In the plates the yield point, ultimate strength, elongation and reduction of area for varying percentages of alloying additions are designated by appropriate symbols, and are the values given in the tables in the columns of averages.

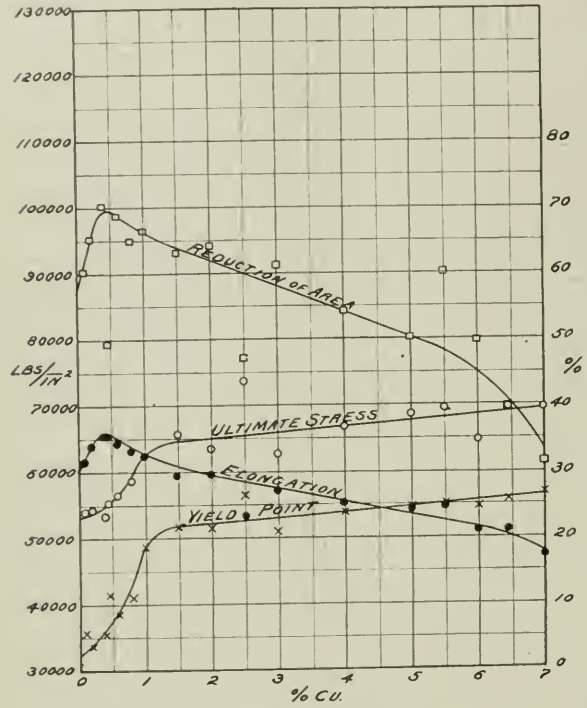


FIG. 4.—TENSILE STRENGTH IRON-COPPER ALLOYS ANNEALED.

With these points as a guide, smooth curves are drawn in each case. With the many factors affecting somewhat the strength of a metal, it is not expected that the points will fall consistently on a curve. It is intended merely that the curve will show the tendency of the effect of increasing the percentage of Monel metal in the alloy.

Unannealed.—The results of the tests in the unannealed state are enumerated in Table 1, and the general summary is best noted by referring to the curves on Fig. 5. Between 0 and 8 per cent there is a rather rapid rise in the ultimate strength and the elastic limit; the latter rises more rapidly, however, and at 8 per cent the elastic ratio is 0.85, indicating brittleness. This is confirmed by the falling off in the values of elongation and reduction of area. Up to 4 per cent of Monel addition there is an increase of ductility, and at this point we note the very fair values of ultimate stress 81,000, yield point 61,070, elongation 30 per cent, and reduction of area 64.3 per cent.

From 8 per cent upward there is a rapid increase in the maximum strength and elastic limit, but with ever-increasing brittleness, as is indicated by the steady falling off of the curves for elongation and reduction of area. The highest values reached for the series were for one of the bars with 18 per cent of alloying addition, the figures being: ultimate, 215,000; yield point, 172,000; elongation, 10 per cent, and reduction of area, 27.6 per cent.

Annealed.—These data are noted in Table 1 and Fig. 6. In the alloys of lower Monel additions annealing has had but little effect on the properties. In fact, and this was noted in the preliminary iron-nickel-copper series, the tendency has been to increase the ultimate strength and elastic limit without material falling off in the ductility. For those alloys with 4, 6 and 8 per cent of addition (the probable useful range) the strength is higher. We now note an increase of ultimate stress and yield point in almost a linear function of the amount of alloying material until at 10 per cent the values are: ultimate, 150,000; yield point, 144,400; elongation, 15.7 per cent; reduction of area, 51.9 per cent. The elongation and reduction of area are in accordance with the stress values.

Between 10 and 20 per cent of Monel metal the ultimate strength undergoes but little variation, while the elastic limit becomes less with increasing additions. Strangely enough, the elongation and reduction becomes steadily less, the latter in particular falling off very abruptly at the higher percentages.

Comparisons.

Direct comparisons of the general tendencies in the iron-Monel metal alloys is best made by means of the curves of Plates 1, 2, 5 and 6. In the unannealed condition (Figs. 1 and 5) there is a marked similarity, considering equal percentages of total additions. In each case there is a maximum of

TENSILE STRENGTH TABLE 1.
ELECTROLYTIC IRON-MONEL METAL ALLOYS.

Per cent Monel	No. Sample	Yield Point lbs. / sq. in. Max.	Min.	Av.	Max. Stress lbs. / sq. in. Max.	Min.	Av.	Elongation % 2" Max.	Min.	Av.	Reduction Area % Max	Min.	Av
UNANNEALED													
2	1			49300			71000			22.5			62.5
4	3	67400	57000	61070	88700	77000	81000	30.0	30.0	30.0	69.0	61.5	64.3
6	1			72500			84000			25.0			54.0
8	3	92500	91450	91830	118300	117200	107670	20.0	20.0	20.0	55.2	51.8	53.2
10	3	150500	132000	139830	170000	168000	163000	15.0	10.5	12.8	42.6	28.6	37.1
12	3	161000	153500	157500	189000	180500	185400	13.5	12.5	13.0	44.5	36.1	40.4
14	4	152000	130000	141370	182000	157300	166820	10.0	5.0	6.9	30.0	8.5	16.4
16	3	171000	154000	162500	191500	179000	184330	12.5	5.0	9.8	32.3	12.2	27.2
18	2	172000	168200	170100	215000	196000	205500	10.0	5.0	7.5	27.6	7.2	17.4
20	2	191[illegible]	183[illegible]	187100	20500[illegible]		202000	7.5	4.5	6.0	8.8	5.8	7.3
ANNEALED 900°C													
2	2	40100	40000	40250	64500	61500	63000	30.0	30.0	30.0	70.0	58.0	64.0
4	2	71600	71000	71300	91000	84600	87800	25.0	22.5	23.7	61.0	59.5	60.2
6	2	91000	88[illegible]	91100	109000	105600	107300	20.0	17.5	18.7	51.7	47.3	49.5
8	3	98500	75800	88100	115200	90700	102970	21.0	18.5	19.8	64.6	60.0	62.5
10	3	158[illegible]	126200	144400	165000	131000	150000	17.0	15.0	15.7	59.8	45.0	51.9
12	3	166800	120000	145930	167200	126000	151400	15.5	10.0	12.2	50.5	31.6	39.0
14	2	150000	143600	146800	156000	147000	151500	10.0	10.0	10.0	40.7	36.0	38.3
16	1	152000	127000	138600	166500	142000	153000	11.5	10.0	10.8	41.6	39.5	40.5
18	2	157000	151[illegible]	153[illegible]	166500	158000	162250	13.5	7.5	10.5	20.4	11.0	15.7
20	2	130000	123100	126[illegible]	170000	130000	151050	5.0	2.5	3.8	1.1	1.0	1.1

strength and elasticity at about 12 per cent. Up to this quantity there is a continual rise, more abrupt, however, in the case of the Monel alloys; and beyond 12 per cent there is the same dip in the curves with subsequent rise. On the other hand the nickel Monel metal than with nickel alone, but the ductility is greater in the latter case. In both cases the pronounced maximum of strength and elasticity is now not noticed. On the other hand, while annealing of the nickel alloys has lowered the ultimate

TABLE 2

COMPARISON IRON-NICKEL-COPPER ALLOYS

UNANNEALED

Per cent Addition	Yield Point				Max. Stress				Per cent Elong.				Per cent Red.			
	Ni	Monel	Ni Cu	Cu	Ni	Monel	Ni Cu	Cu	Ni	Monel	Ni Cu	Cu	Ni	Monel	Ni-Cu	Cu
1.5				73920				77300				23.5				66.9
2.0	58520	49300		76930	70350	71000		87010	26.0	22.5		21.2	60.7	62.5		57.0
2.7			62100				79200				26.0				56.7	
3.0	62920			86810	74860			99640	24.5			16.7	63.9			53.8
4.0	65000	61070	67400	100560	75670	81000	80100	108640	27.8	30.0	28.0	13.5	66.5	64.3	58.8	45.6
5.0	64030			116350	77150			121900	29.2			14.5	65.3			49.2
5.3			63300				85700				23.0				45.7	
6.0	67300	72500		113400	77300	84000		122900	29.0	25.0		6.5	68.5	54.0		20.3
6.6			66500				89400				22.0				39.3	
8.0	70770	91830	81700	135100	91740	107670	99700	148750	22.6	20.0	25.0	0	56.1	53.2	53.5	0
10.0	118960	139830			153360	163000			9.1	12.8			29.0	37.1		
12.0	141140	157500			178420	185400			7.1	13.0			23.8	40.4		
14.0		144370				166820				6.9				16.4		
15.0	125800				184730				8.2				34.1			
16.0		162500				184330				9.8				27.2		
18.0	138170	170100			178500	205500			12.3	7.5			49.3	17.4		
19.0	134730				183270				14.2				53.1			
20.0	115500	187100			186100	203500			16.0	6.0			59.5	7.3		

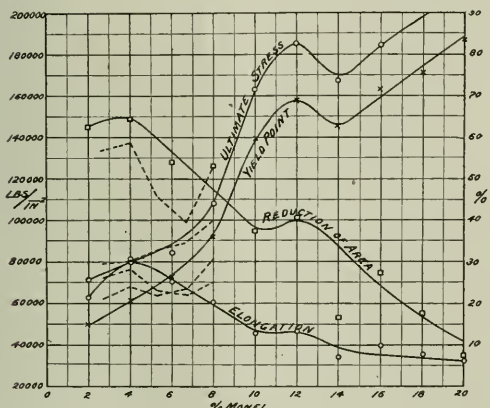


FIG. 5.—TENSILE STRENGTH IRON-MONEL METAL ALLOYS, UNANNEALED.

alloys show a brittle zone at about 12 per cent, with a recovery of ductility with further additions; with the Monel metal alloys the brittleness becomes more pronounced with increasing additions.

The relations in the annealed samples (Figs. 2 and 6) are similar to those commented upon above. The same brittle zone with subsequent recovery is noted in the nickel series, but in the Monel metal alloys the brittleness increases continuously. Up to 10 per cent of total additions the increase of ultimate strength and elastic limit is much more pronounced with strength and the elastic limit throughout the entire range, with an accompanying increase of elongation and reduction, the effect of heating of the Monel metal series has been pronounced only for percentages above 10, below this range the result is to increase the strength somewhat with little alteration of ductility.

In Tables 2 and 3 are collected the data of all of the series in order to make direct comparisons of the effects of the addition of Monel metal and of nickel and copper to iron in the ternary alloys, with those of the separate additions of the two elements in the binary series. The several items are noted in columns for each group; the compositions are given as the total percentage of alloying addition; the separate proportions of nickel and copper in the ternary series can be calculated from an assumed ratio of three nickel to one copper.

In the preliminary work the object was to see whether the properties ordinarily accompanying the addition of nickel to iron were detrimentally affected by the copper carried in Monel metal and the comparison was made on the basis of equal nickel content and regarding the copper as so much superfluous addition. However, it was found that the copper had a beneficial effect, and a more rational comparison, therefore, can be made by tabulating on the basis of total addition material.

But little discussion of the figures is necessary. It will be noted that the ternary alloys (both Monel and nickel-copper) hold their own with either of the binary alloys through the entire range. And this is especially true in the annealed samples, particularly below 10 per cent of total additions, the range of probable greatest utility. At 4 per cent, for example, we

TABLE 3

COMPARISON IRON-NICKEL-COPPER ALLOYS

ANNEALED

Per cent Addition	Yield Point				Max. Stress				Per cent Elong.				Per cent Red.			
	Ni	Monel	Ni Cu	Cu	Ni	Monel	Ni Cu	Cu	Ni	Monel	Ni Cu	Cu	Ni	Monel	Ni-Cu	Cu
1.5				51750				65720				29.2				63.1
2.0	48770	40250		51620	64400	63000		63200	34.2	30.0		29.5	65.4	64.0		64.4
2.7			48600				68700				30.0				61.5	
3.0	55400			50890	70670			62360	27.5			26.9	67.6			61.7
4.0	52100	71300	56800	53570	70070	87800	72500	66540	28.4	23.7	31.0	24.8	66.7	60.2	59.1	54.2
5.0	58320			54460	73230			68250	31.4			24.2	68.1			50.3
5.3			67600				84700				28.0				55.6	
6.0	56400	91000		54420	75300	107300		64530	29.5	18.7		20.8	64.2	49.5		49.9
6.6			74200				86400									
8.0	63320	88100	83200		78470	102970	103700		30.2	19.8	22.0		64.4	62.5	56.2	
10.0	71800	144400			89300	150000			22.3	15.7			59.1	51.9		
12.0	97730	145930			121800	151400			14.5	12.2			35.8	39.0		
14.0		146800				151500				10.0				38.3		
15.0	126300				153300				10.5				35.8			
16.0		138000				153000				10.8				40.5		
18.0	137700	130000			181550	162250			10.0	10.5			34.4	15.7		
19.0	116150				180700				10.7				38.7			
20.0	111000	126700			124900	154950			21.5	3.8			62.5	1.1		

note the following figures: ultimate strength, 87,800 and 70,070 yield point, 71,300 and 52,100; elongation, 23.7 and 28.4, and reduction 60.2 and 66.7, these figures being, respectively, for total Monel metal and nickel additions. The ductility of the

Monel samples is entirely comparable to that of the nickel series in the unannealed condition; after annealing it is somewhat lower. But this may be due to the necessity of different annealing temperatures since in the ternary nickel-copper-iron series higher values for elongation are the rule.

These results seem to warrant the conclusion that for the carbon-free alloys, at least, the combined addition of nickel and copper in the proportions found in the commercial Monel metal does not result in any deterioration of the good qualities

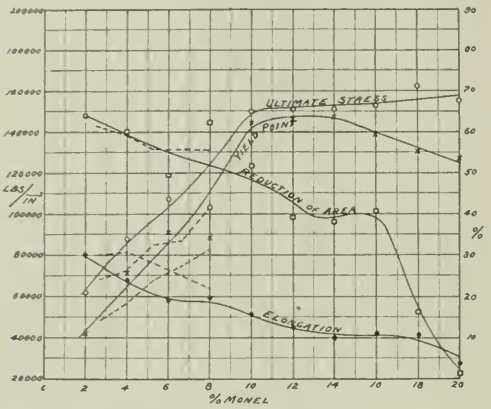


FIG. 6.—TENSILE STRENGTH IRON-MONEL METAL ALLOYS, ANNEALED.

conferred by the nickel alone; in fact, the copper has good inherent qualities of its own. The general effect is the same for both series for equal total alloying additions. Whether these same good features will result from like additions to commercial steels, with their customary amounts of impurities, is a problem now under consideration.

Chemical Engineering Laboratories,
University of Wisconsin.

Corrosion of Iron and Steel and Its Prevention.—A paper read by Mr. Gustave W. Thompson, of Brooklyn, N. Y., before the American Institute of Chemical Engineers gives a summary of the facts which have been discovered with reference to the cause and prevention of the corrosion of iron and steel. The author leaves it to the reader to interpret these facts in the terms of any theory with which he may be familiar. The general conditions conducive to the formation of rust are first given. The opinion is given that facts do not warrant the conclusion that the tendency of iron to corrode is dependent on the composition of the metal and that the use of a poorer and more expensive metal as against a more impure and cheaper metal is not justified by the facts. Practical suggestions with reference to the prevention of corrosion of iron and steel are given, among which the following may be mentioned: The action of acids, moisture and oxygen. Cement prevents corrosion because it is alkaline. Protective coatings exclude moisture and oxygen. The cleaning of iron and steel before painting is of great importance. Specific rules for the cleaning of steel and iron are given.

The principles which should guide in the selection or designing of the paint are given. Poor workmanship in the application of paint was given as responsible for most of the failures in the protection of iron and steel with paint. Linseed oil was condemned as a prime coat for iron and steel. Nineteen different paints applied to the Havre de Grace bridge of the Pennsylvania Railroad all proved equally effective because they were properly applied. Oxygen cannot be excluded from iron by means of any protective coating. In the absence of moisture, oxygen does not produce corrosion.

Study of Materials as a Subject in a Course in Chemical Engineering.

Dr. Charles F. McKenna's presidential address delivered at the Niagara Falls meeting of the American Institute of Chemical Engineers called attention to the rapid development of the studies whose aim it is to increase our knowledge of the "properties of materials." In the past, there has been a limited view of both the particular objects to be studied under the term "materials" and also of the scope of the inquest to be constituted into the properties. Elasticians began their studies upon beams and the problem of Galilee led to consideration of compression of fibers and thus to the study of columns and other forms used in construction. Naturally, the properties studied were the strength elements and the materials considered only those used by constructors.

But a broader view of both has now been brought about by two influential factors. First, the deep researches of mathematicians into the theories of elasticity have extended that study until it is seen to be founded on conceptions of cohesion molecular constitution, intermolecular attractions or other forces and that these lead far afield amongst many other new territories of physics. Secondly, modern activities have developed engineers who busy themselves in many other branches than construction or the art of war, which once monopolized their science, and who utilize as substantial agencies many other materials than wood or iron or lead. This is a return to a better practice which was in vogue in more ancient times, as is shown in some remarkable passages in literature, as in Pliny and Nitruvius, not to go back to the Book of Books.

An extraordinary number of substances are now-a-days being made the subject of inquiry by the engineer as well as by the technologist. But there are great dangers in the pathway of those who essay to study those materials in the haphazard fashion of an untrained observer, or to use them without any proper study of their nature.

As generally understood "materials" are those substances or fabrics whose applications are found in the useful arts. Raw materials form the basis of manufacture and the results are besides the chief product desired, by-products and waste materials. It is the aim of technology to so perfect materials and processes that there should be no wastes and that all products be brought into such conditions or such markets that revenues can be obtained from each adequate to its importance. In the history of some industries, it is often noted that the waste product of former days is now almost as much sought after as the former chief product, indeed, in some instances has preempted that position.

"Materiology" can be defined as the study of the properties of materials applicable in the useful arts and of the method of testing and examining them for the determination or comparison of these properties.

Properties may be real or virtual. Real properties are those which belong, in the first instance, to the substance itself, and by which we recognize it in both matter and form. Virtual properties are those which a material has by virtue of its reaction with another material. Composition, density, specific heat are real properties of gasoline, but the property of exploding in a closed cylinder is only a virtual one, for it cannot be manifested without the presence of air. Lubricating oil possesses in viscosity a real property, but its lubricating value is a virtual one, because it is effected by many other conditions.

That this distinction makes for clearness and is not a mere refinement, is illustrated by the case of coal dust and flour dust, which some might class together as explosives. But true explosives are those which contain within themselves the oxygen necessary for complete combustion, while organic bodies, such as coal dust or flour dust require air in intimate mixture before they can become explosive; it is, therefore, the mixture which is really explosive, the particular dust being only virtually ex-

plosive. Whether real or virtual, we recognize chemical properties, physico-chemical ones, physical ones, mechanical resistances, and structure.

In chemical properties we have to consider the active and the passive properties, the ultimate constitution or the proximate constitution. In some industries, as in the corn industry, the ultimate chemical constitution of the chief materials is of fundamental importance; in others, as in fertilizers, rubber gum, etc., only the proximate constitution is of importance.

Physico-chemical properties are seldom studied outside of the research laboratories, but classified knowledge of this character is of primary importance for the chemical engineer.

Under structure the materials are treated of microscopically and macroscopically, with regard to their morphology and state of aggregation. Few achievements of modern times in applied science have surpassed in importance the work done by this mode of study in alloys.

To acquire a full knowledge of the most intimate properties, direct and indirect, real and virtual, of all technical substances, we must approach the subject with a peculiar mental equipment, which can only be obtained by much experience. But there are some general principles which supply the necessary universality yet make for simplicity and order.

(1) The full and true value of a material for use in the arts can only be determined by ascertaining the nature of the substance or substances composing it.

Very often a material is applied to certain purposes after a very superficial study and a realization of the inappropriateness of the choice comes too late. The crumbling walls of limestone or of brownstone seen in New York City are examples of this. Other examples are found in the use of refractory clay materials. A fireclay can on certain rough tests be classed as refractory; a real appraisal of its value can only be made by using refined method of chemical analysis, mineralogical or rational analyses, extended fire tests, shrinkage tests, drying tests, etc., observations of conditions, structure, etc.

(2) The simple or homogeneous substances must be the basis of study before the nature of the complex or heterogeneous can be understood.

We should know the character of iron before we presume to understand the character of iron and carbon, iron and manganese, iron and nickel, etc.

Portland cement furnishes us with another example of the applicability of this principle. In this case we should first study lime and its hydrate, then the properties of the silicates and aluminates and sulphates, their interractions, and even, as well as those of the minerals or substances in nature having the same or analagous compositions, besides bringing to bear upon it a full comprehension of the conditions surrounding matter in the colloidal state.

(3) The economic adaptability of a material can only be determined by a knowledge of its behavior under all the forces inherent or applicable to it.

Tin would be an example of this, which we look upon as a firm and moderately resistant metal at all ordinary temperatures. It would be well to know, however, that if it is to be used in some very cold position it can revert to the enantiotropic form in which it has no cohesion and falls to powder. This has lately been made the basis of a detinning process.

(4) Where a series of causes operate upon a material, their effects cannot be understood until these causes are recognized and investigated, and their operation in order of time understood. Dr. McKenna made this clear by giving the case of fireproof wood as an example. Not many years ago an investigation of fireproof wood was projected in the hope of finding a satisfactory and discriminating test, and the committee assuming charge of the matter proposed to study the different treating processes and deduce some views of testing the relative fireproofing value as well as the incidental effects of poisonous gases driven off by heat from the chemicals saturating the wood. Various ways of applying the flames to shavings and to test sticks were to be assayed. Strength tests of the wood before and after treatment were to be made and various such features were to be investigated. This investigation was never consummated, but it is probable that if it had been made according to this program much futile effort would have been spent upon it.

It was early pointed out that the proper study depended upon a previous knowledge of what really goes on when wood is subjected to heat and flames. This heat can be assumed to be initially low and progressively rising. The conditions in some cases, particularly regarding the exterior surfaces, would be those normal to a free kindling of wood. The conditions as regards other portions would be those prevailing in the destructive distillation of wood without access of air. Naturally, therefore, a long series of heating and distillation tests would be necessary, and few of the other tests of the previous program would be of any value. As a matter of fact, such a study, using a small electrically heated retort adapted for quick work, did show that the yield of gas from the heating of wood in closed vessels would make an excellent indicator of the efficacy of the fire-retarding salts used for treatment, and would not give the contradictory results of the shavings or fire test. Many complex processes in technology would become research problems of the first order, and with great promise of positive additions to knowledge, if this principle were observed.

(5) Purity of substances and behavior of materials under the action of forces can be formulated in terms called Standard Materials, but cannot be determined in individual specimens with greater exactness than the limitations of observations, of instrumental precision and of the accuracy of the available standards of measure will admit.

While this simply means that errors are personal and instrumental, it is well at times to keep it in full view and to try to know well the equations and their quantitative relations. One feature of this principle was covered by Ostwald in what he called "the fundamental rule for drawing up a plan of measurement: The accuracy of the measurement must correspond to the exactness of the definition of the object to be measured."

A recognition of this principle also will save much useless work, because it is, unfortunately, too common to see enormous numbers of experiments proposed and numbers of workers engaged therein striving to use much refinement of measurement while the definition of the point involved is on a scale so much greater that no measurements will be needed other than what could be called gross measurements.

This principle also contains the germ of an idea which has often occurred to Dr. McKenna in discussing the vexed question of standard or uniform methods of analysis. Too great an insistance upon the adoption of these methods is apt to lead away from truth rather than toward it, and apt to put a premium upon too little investigation rather than to show the danger of too great effort or waste of too much time. Every substance of the same name has not the same composition, and any method which turns the chemist into a follower of system may produce economy of time in the routine of a factory and may more often than not favor comparison of results by diverse interests; but it is conceivable that the truth can thus be missed at times, and it is quite certain that in ultimate reference more independence must be permitted to the experienced analyst.

(6) In obtaining data on mechanical resistances the mode of application of a stress must be adequate, and the means of measurement of the strain adapted to the derivation of a factor commensurate to the magnitudes to which that factor is to be applied. In a great many branches of experimental science we can assert that what is true of the small is true of the large. This is more often observed in chemistry than in physics; it is most often physical factors which introduce variables into a problem when the results of a small operation are to be applied on a larger scale.

In pyrometry and the study of fuels we find that calorimeter test results of a coal do not coincide with results obtained on the large scale with the steam coal in a boiler test.

But in mechanics it is particularly necessary to consider this element of variation. For example, when calculating upon safety factors in structures whose parts are so gigantic as to make difficult the use of coefficients by the mere multiplication of cross-sectional areas. The Quebec Bridge was designed and almost completely erected upon the assumption of the correctness of certain factors for the resistance to compression obtained in laboratory tests. No data was available for computing from results obtained on large members. This was a fatal element of weakness and led to the disastrous collapse of the structure.

In the "control of materials" certain distinctions should be observed and certain authoritative positions should be assigned to the individuals concerned in these important affairs. Thus, each of the following functions are distinct: Specification, inspection, testing, judgment, sanction.

The drafting of a specification for material is the work of an engineer, an architect or technologist, making use of his experience and knowledge, or the knowledge of an authority on materials. The inspection is the routine of determining the compliance with the specifications of the material being delivered; testing preliminary samples submitted from outside sources is not inspection. Inspection involves the sampling and weighing of cargoes, determination of observable and important conditions at the time of delivery, and the transmission of samples intact to the testing laboratory. Inspection is not testing, nor is testing itself complete till the judgment is rendered as to values. Finally, this preliminary work would be useless if there could be no enforcement to the provisions of the specification. This is a *sanction* which must go by the act of acceptance, or rejection, and must be within the legal definition of the engineer's powers, as well as be susceptible of support upon referee tests.

The legal aspects bear upon the responsibility of the individuals engaged in this chain of duties. The engineer specifies; the inspector reports deliveries and samples them; the testing engineer makes return under chemical, physical, microscopical or mechanical tests, and states the degree of compliance with the specification; the engineers accepts or rejects. If a contest ensues and it is concerned with the question of accuracy of testing methods, a referee would be chosen from another laboratory, or, if the dispute was to be carried higher, we would look upon the Bureau of Standards as the final arbiter.

In conclusion, Dr. McKenna pointed out that the valuable element in this study for the young chemical engineer is the development of logical processes of thought applied in these practical fields. The great number of hours which would seem to be required for laboratory work would not be a disadvantage when it is realized that this can be co-ordinated in any good course in chemical and physical experimentation and the measurements of science. The student would know the chemical properties of substances in general, would be familiar with the fundamental concepts of the states of matter, but in this study he would consider substances, materials and fabrics in a scheme embracing co-ordinate sciences and in which the mode of attack upon the subject is always one relying upon definitions and distinctions, first principles, exactness of observation and cautious summary and judgment.

If he develops to be a skilled experimenter, manipulator and mechanician he will be especially well qualified. His powers of adapting his resources to his needs for determining the factors of materialogy in any problem before him will give him the same generous equipment of mental self-reliance that a good course in chemical analysis has given him in chemical manufacture. Any study which brings new knowledge into orderly array and reconciliation with historic knowledge, which makes insistent demands upon logical precedure, accuracy of observation and honesty of statement, and then applies results of investigation in a practical way to the most useful works of construction and production is a fit study to place high in a course of chemical engineering.

The Development of Heavy Gravitation Stamps.*

By W. A. Caldecott.

The history of ore crushing by means of gravitation stamps shows a progressive increase in their weight and in corresponding efficiency. This holds good in the development from the square, wooden non-rotating stamps in use in Germany in the Middle Ages to iron-shod wooden stamps in Cornwall, and then to rotary iron stamps in California and mild steel stamps in South Africa. The latter have greatly improved during the last few years, and probably represent the highest existing stage of development. About 1835 the first stamp mill in the United States was started in Tellurium, Virginia. The stamps weighed 50 pounds and were made of square wooden stems with iron shoes and dies, the cams operating in slots in the stems.

Two years later the Vaucluse mine installed 380-lb. stamps, of which the head weighed 125 lb. When the first batteries were erected on the Rand, over 20 years ago, Californian practice probably represented the highest state of the art, and was introduced upon these fields by J. S. Curtis, J. H. Hammond and other well-known Californian mining engineers.

To illustrate the type of battery in operation on the Rand in 1899, in which year the writer's experience in stamp milling began, the following details may be given of the Du Preez Gold Mining Company's mill, premising that this was before the introduction of the cyanide process, and that concentrators, other than blanket strakes and buddles, were not in common use.

The mill was erected in that year and was of Sandycroft make, designed in accordance with the best Californian practice to date. It consisted of 20 stamps weighing 900 lb. each when new, and was run at the rate of 90 7-in. drops per minute. The stamp duty was about 3 tons per 24 hours, using a screen of 900 meshes per square inch.

By 1899 some progress had been made, and 1250-lb. stamps were in use at the Robinson Gold Mining Company and elsewhere, though in Australia and the United States 900 lb. per stamp remained the almost universal maximum limit, and this remains true of the celebrated Homestake mills at the present time. At the Mount Morgan Gold Mining Company (Westralia), in December, 1902, 30 stamps weighing 1500 lb. each were erected. These were eclipsed, however, by the neighboring Millionaire Gold Mining Company's battery of five stamps weighing 1750 lb. each. The advance made on these small companies was not generally realized, and in 1907 few stamps dropping on the Witwatersrand weighed above 1250 lb., with the exception of certain batteries of the Consolidated Goldfields Companies.

In 1904 the author was authorized to carry out a series of milling tests at the Knights Deep battery, which was equipped with stamps weighing 1350 lb. when new. The object of the trials was to discover some means whereby stamp-milling efficiency could be increased.

The initial experiments were made to test the effect of fine breaking before stamp milling, and also whether the increased discharge area afforded by a double discharge mortar box increased the stamp duty. A pair of rolls was installed to effect the preliminary fine breaking, and also a double discharge mortar box. The tests extended over more than a year, and many variations of conditions were investigated, such as varied angles of screen frames, heights of discharge and of drop, and different methods of water introduction into the upper portion of the mortar box.

The results of the trials were almost identical within the

* Abstract of a paper read before the Inst. of Min. & Met. and published in the *Bulletin* of Aug. 12, 1909, also of discussion before the Chem. Met. & Min. Soc. of S. Africa, published in the *Journal* of April, 1910.

limits of experimental error. The average stamp duty on ordinary mill feed for six sets of trials was 5.82 tons with a double discharge mortar, and the same with a single discharge mortar. The average stamp duty for four sets of trials on the product from rolls set ½ in. apart was 5.64 tons with a double discharge mortar and 5.78 tons with single discharge mortar. There was no increased duty to compensate for the extra cost of preliminary fine breaking, and the only difference with the double discharge mortar was a considerable increase in the amount of water used.

The last conclusion is in accordance with the fact that the ordinary single-battery screen has about 40 per cent of its area available for discharge or say 1½ sq. ft. of discharge area for the small stream of pulp yielded by five stamps.

Finally it became obvious that to crush more rock it was necessary to strike a heavier blow. This was sought by fixing helical springs round the stem above the tappet so as to be compressed by the upper guide during the stamp's rise and so that when released the stamp would fall more rapidly, rendering a higher drop and harder stroke practicable. But the life of the springs did not promise to be long and it was finally concluded that a heavier stamp was the best course to adopt.

A series of runs was then made in which direct comparisons were made of the actual tonnage milled by alteration in the weight of the stamps. An increased tonnage was obtained by increased weight of stamp on ordinary mill feed, both under ordinary milling conditions and with a high discharge and back-water feed to obtain a very fine product.

Another run with 1337-lb. stamps, with a height of discharge of 2 in. and a screen of 64 meshes, gave a duty of 10.23 tons per 24 hours, of which 52.8 per cent remained on a 60-mesh screen.

Encouraged by the results of these and many other tests, including a duty of 6.5 tons through 1500-mesh screening with stamps weighing up to 1622 lb., the writer recommended that the next 360 stamps of the Consolidated Goldfields Companies should have a weight of 1550 lb. each when new. Such a departure excited much adverse criticism among experienced millmen generally, on the ground that no foundations could stand the shocks, but the reasoning that if foundations could be built to withstand the blow of the colossal steam stamps, it would be possible to build them to endure the blow of gravitation stamps proved to be correct. All of these heavy stamps are working satisfactorily to-day after a service ranging up to three years and the foundations promise to last as long as did those of their lighter predecessors.

In accordance with the present changed state of professional opinion, all Rand batteries lately erected or designed have stamps of a weight not long ago deemed extravagant, and, aided by the installation of tube mills, stamp duties of 8 tons are as common as of 4 tons a few years ago. While in the United States and Mexico the relatively light stamp is still generally upheld, there is at least one notable exception in the recently built mill of the Boston Consolidated Copper Company,[1] which is equipped with 312 single Nissen stamps of 1500 lb. weight each and a duty of 9 tons.

Somewhat later than the battery tests just given, the subject of battery foundations was investigated. A series of 23 comparative trials was carried out on two five-stamp batteries side by side in the Simmer East mill. One battery was on a concrete foundation only, while the mortar box of the other rested on a cast-iron anvil block placed on a concrete foundation. The results of the trials were practically identical. The tests included screening from 100-mesh to 140-mesh, and the cast-iron anvil block battery showed a duty of 6.84 tons per stamp as against 6.78 tons per stamp on the concrete blocks. One mortar box at the Simmer East mill was erected on a concrete block with a thin layer of insertion between, and has been in commission for three years with no visible sign of deterioration or cracking.

[1] As stated in our issue for June, 1910, p. 313, these stamps are to be discarded as a means of crushing, and will be superseded by rolls and Chilian mills.

Impact vs. Abrasion.

Crushing is usually performed either by impact or abrasion, and while hard ores are dealt with by the former method, the latter is applicable to softer material. The relative efficiency of the two methods for hard material is well illustrated by the diamond, which is hardly capable of abrasion at all except by its own dust, whereas if placed on an anvil it may be shattered into a thousand pieces by the blow of a hammer.

The crushing of larger pieces of ore in stamp milling is done as the result of reaction from the resilient die, the waves of compression transmitted through the ore from the shoe being reflected from the die until the strains set up from the oncoming and reflected waves exceed the elastic limit of the or and rupture results. The stamp is essentially an impact machine, and hence readily pulverizes hard, tough ore. Such ore if handled in any of the legion of rubbing and shearing appliances would be reduced only at the expense of serious wear by abrasion of costly steel parts, which may have to be discarded when only a fraction of their weight has been worn away. With soft material the case is different, and machines of the latter class may be employed with advantage and without undue abrasion of wearing faces.

It is much to be regretted that no satisfactory machine for testing the capacity of material to resist abrasion, or conversely of testing the abrasive capacity of materials requiring reduction, yet exists, as this property is of prime importance in all crushing operations.

When gravity stamps are used for very fine crushing they pass beyond the economic range, and operate by abrasion rather than by impact, with consequent reduction of efficiency as may be seen from the following considerations: Impact is essentially dependent on the pressure exerted by the blow, and depends on the weight of stamp and height of drop. Assume that a 1750-lb. stamp having a 9.5-in. diameter shoe, with a set height of drop of 8 in., strikes a layer of ore 2 in. thick on the die and comes to rest 0.5 in. from the die. Then, having fallen 7.5 in. and having done work for 1.5 in., it has exerted on the ore a mean total pressure of 8750 lb. The shoe having an area of 71 sq. in., the average unit pressure exerted over the whole of its area is only 123 lb. per square inch, or less than that of an average man supported on 1 sq. in. The crushing effect of such a pressure on hard quartz grains is trivial, and a certain amount of reciprocal abrasion of grains of sand rearranging themselves under pressure is probably the main effect produced. The case, however, is quite different in the first impact of the stamp on the ore in the box. Assume that only a 2-in. cube resting on the die is first struck; having an area of 4 sq. in., the average unit pressure here is 2190 lb. per square inch, or 18 times as much as under former conditions.

The finer the crushing is carried in the battery the longer sand particles remain before being reduced sufficiently to escape, and the higher the ratio of fine to coarse material in the mortar box. Hence, under these conditions the liability is for the falling stamp to have its force of impact distributed over so large an area as to produce little effect but abrasion on a bed of shifting, compressible sand. These conditions serve to explain also why fine breaking before milling with heavy stamps does little good. A large number of pieces of ore of approximately equal size afford a large area to receive the blow of relatively small pressure on each. And further, the abrasive action above referred to explains why stamp milling with a fine screen and a high discharge, or both combined, is inefficient through the waste of energy in converting fine sand into slime by abrasion of the particles rearranging themselves under the stamp.

Size of Ore for Stamp Mills.

As regards the desirable size to which ore should be broken

before entering the mortar box, the disadvantage of too fine crushing yielding a uniform bed has already been pointed out. The maximum limit varies with the class of ore, being less for hard ores, but, in general, it should be such that not more than one blow of a stamp is required to pulverize the largest piece of rock. Hence, the heavier the stamp, the coarser the preliminary breaking admissible, and vice versa. Larger pieces are more economically reduced by the rock breaker, and the lowering of stamp duty, owing to less actual height of drop is thus avoided.

Since greater attention has been directed to the use of heavy stamps, it has been customary to use a compensating weight on light stamps. Probably the earliest form of this device was placing an old head at the top of the stem or an extra tappet above the one in use. These devices are crude and do not add the weight where the writer believes it is better, viz., near the bottom of the stamp. The most convenient form of compensating weight is a split cast-iron disk about 4 in. high, weighing about 50 lb., which can be clamped on the stem by bolts. As many can be added as desired.

With increased weight of stamp, the question of increased shoe and die area arises, but the considerations already advanced, showing the small amount of pressure per square inch of total shoe area, indicate that very large shoe and die areas are not needed for hard ores, while for soft ore a lower and quicker drop can be used. An increase is desirable when the total force of impact of the falling stamp requires a thick layer of ore on the die to avoid excessive shock. Such heavy feeding lessens the duty because the thicker ore bed reduces the height of drop, and the blow is likewise cushioned so that the stamp crushes by abrasion rather than by impact. The cushioning by fine particles will increase with the height of discharge and the fineness of the screen.

The greater the force of impact per unit area, the coarser the screen required for maximum efficiency, as otherwise a thick layer of fine ore particles would be required to avoid pounding. The least thickness of ore layer necessary to take the impact of the stamp without pounding is sufficient to supply ample material for pulverization.

Consequent upon the introduction of the tube mill, coarse battery screening has generally replaced the finer mesh. The tube mill has, moreover, lessened the importance of the battery screen as the criterion of the degree of the reduction of an ore, and the grading analysis of the pulp as it leaves the overflow of the hydraulic classifier is now the guide to the metallurgist as to the degree of reduction. This change makes the degree to which a particle is reduced dependent upon both its diameter and its specific gravity, instead of on the former only, with the result that the pyritic material undergoes that finer comminution which more perfectly exposes its gold content.

In order to employ heavy gravitation stamps to full advantage it is absolutely necessary that a uniform speed should be maintained by proper governing of the engine. Only under these conditions can the maximum speed and height of drop be used without fear of camming; and the damage done in a few moments by sudden increased speed above the normal may prove a costly matter, both in loss of time and material. Further, with heavy stamps, the use of five-stamp camshafts is very desirable, in that, as compared with 10 or 15-stamp camshafts, both the power to be transmitted and the number of shocks due to lifting stamps are reduced by one-half or two-thirds for each shaft.

The future limit of stamp weight is difficult to foretell, but it will probably be determined more by the mechanical considerations involved, as in the cam system of lifting, than by any decreased relative efficiency as a device for pulverizing ore. The advantages of heavy stamps as compared with light ones may be briefly stated as follows:

1. Reduction of the initial capital expenditure in erecting, say, 200 stamps at 1750 lb. with accessories, in place of 280 stamps at 1250 lb. each.

2. Reduction of size in mill building, almost proportionate to the less number of stamps.

3. Thirty per cent less shafting, belting and other moving parts to maintain.

4. Thirty per cent less labor required for dresssing plates, lubricating moving parts, changing screens and other work incidental to milling operations.

* * *

The discussion of Dr. Caldecott's contribution has been very extended, and we append hereto a few extracts of comments by one of his critics, together with some extracts from the author's reply.

Mr. E. J. Way: While Dr. Caldecott is very emphatic in his personal opinion that the increase of weight already carried out does show very appreciable economic improvements, this opinion is not supported by any convincing arguments or array of experimentally proved data. I had been led to hope that the author's unrivalled opportunities for arriving at definite conclusions based on experimentally determined facts would have enabled him to throw some very clear light at last on that most important question: What are the limits within which primary and secondary crushing machines would operate at their best efficiency and give the best return for the money invested? I had hoped that this question would have been fully treated from every standpoint, namely, first cost, running and maintenance cost, life of mine, and all the other factors that enter into the question of economic efficiency in contradistinction to mere mechanical efficiency. The laudable and useful objective of the experiments seems to have been lost sight of, and the experiments to have been carried out merely to see how the capacity of the stamps could be increased. A careful examination of the data brought forward by the author on which he apparently based his recommendation that heavy stamps should be installed at the mines mentioned (a recommendation, by the way, hardly confirmed by the total working costs per ton milled at the mines at the present time) simply shows that, the number, height of drop and height of discharge remaining the same, the extra amount crushed is almost directly proportional to the weight of the stamps. In other words, there is no increase of efficiency, but only of capacity. A further examination shows that in order to obtain the increased tonnage it was necessary with an increase of 32 per cent in stamp weight to increase the water per ton of ore crushed 41 per cent, while the amount of +60-mesh product increased 126 per cent. This would appear to indicate that, so far from there being an increase in the economic efficiency due to crushing large tonnages through fine-mesh screening by heavy stamps, there is actually a decrease in economic efficiency.

The experiments on preliminary fine breaking with rolls before milling might with profit have been extended to trying fine breaking in conjunction with lighter stamps, lower drop and greater number of blows per minute, as it would certainly appear, theoretically, that for every size of feed there should be, in order to obtain a maximum mechanical efficiency, a corresponding pressure over the unit area resulting from the blow of the stamp and depending on the three factors, weight of stamp, area of shoe and height of drop. It would appear that for a fine feed a light stamp with a low drop and running at a high rate of speed would prove most efficient, and it is to be regretted that the author's experiments were not carried further in this direction.

The author states that "fine breaking before milling with heavy stamps does little good," and goes on to say that "the heavier the stamp the coarser the preliminary breaking admissible, and vice versa." In other words, the heavier the stamp the coarser the preliminary breaking admissible, and conversely, the lighter the stamp the finer the breaking admissible. I am of the opinion that here the author is getting at the very root

of the matter. I am inclined to believe that if we looked upon the heavy stamp as a fine breaker and were to interpolate it, in place of fine rolls, between the rock breaker proper and the light stamps, very appreciable gains in economy might result, as it must not be forgotten that the heavy stamp, even when crushing through four-mesh screening, produces as a by-product some 25 per cent of —90 grade. The fact that maximum mechanical efficiency when using heavy stamps is obtained only by crushing through correspondingly coarse screens, moves the sphere of action of heavy stamps more and more, with increase of weight, in the direction of coarse crushing. The economic limit of the weight of these stamps, leaving out mechanical difficulties, appears to be reached only when they encroach on the province of the jaw breaker or gyratory crusher with which they cannot compete. This economic limit will be reached when the mechanical advantages of the jaw breaker are balanced by the advantages accruing from the percentage of —90 grade produced by the heavy stamp, even when crushing through coarse screen.

As both heavy and light stamps, if properly designed for the size of ore fed and screen used, would be working at the maximum mechanical efficiency, and as the mechanical efficiency of the tube mill working under the best conditions is only about 25 to 33 per cent of that of a properly designed stamp mill, it would appear that both the mechanical and economical efficiencies of a plant thus designed should be considerably better than that of the present fashionable heavy stamp plus tube mill combination.

Commenting on the four advantages which Dr. Caldecott ascribes to installations of heavy stamps. I should like to make the following observations:

1. The reduction of capital expenditure on stamps and accessories is not as great as might appear probable at first sight, as the total weight of the stamps themselves is identical, while the greater vibration consequent upon the heavier weight of the stamps necessitates the use of massive concrete foundations in place of the old timber piles, the replacement of the wooden king post by steel or reinforced concrete construction, and the installation of far heavier and larger cams and camshafts, pulleys, friction clutches, etc. The total quantity of water passing through each box will be more than doubled, while the larger ore feed will necessitate bigger and more expensive feeders. If to the cost of the stamps be added the cost of the necessary secondary grinding plant, such as tube mills, with their launders, cones, pumps and other equipment, it will be obvious that the cost of crushing appliances alone will be higher with heavy stamps than with light ones.

2. As to there being any reduction in the cost of the mill building, this is more than doubtful if the mill bins are retained as an integral part of the building. The usual allowance of three days' storage capacity means that the old allowance of 12 to 15 tons per stamp will have to be increased to 60 tons per stamp, necessitating, in the case submitted by the author, a storage capacity of 12,000 tons, as against 4200 tons. This enormous increase would cause greatly increased strains throughout the bin structure, and would probably make the cost of mill buildings greater with heavy stamps than with light ones.

3. There will certainly be some 30 per cent fewer moving parts, but as these will all be larger and the wear and tear will be correspondingly greater, I very much doubt whether the maintenance and renewal costs in the mill alone will be greatly reduced, and if to these costs are to be added the tube mill maintenance costs, the total maintenance costs will almost certainly be higher.

4. I do not see that there can be as large a decrease as 30 per cent in the labor requirements. Take the question of dressing plates. It has been found that when crushing through coarse mesh screening it is necessary, owing to the scouring produced by the large particles and the increased grade of the plates required to allow the pulp to flow freely, to remove the plates from the mill building and place them below the tube mills. Here there will be no reason why their number should correspond with the number of mortar boxes in the mill, and, as a matter of fact, their number will probably be not less than in a light stamp mill of similar capacity, so that the labor of dressing plates will probably be the same in both cases. Any saving of labor in other work in the mill will probably be offset by the work in connection with the running of the tube mills. Generally throughout the plant costs will probably be found to be higher in two-stage than in single-stage crushing, due to, first, the heavier running and maintenance costs of the heavy stamps and tube mill; second, the absolute necessity of skilled supervision of the tube mill plant, if it is to run at its low maximum efficiency, and, third, the increased pumping costs in a heavy stamp and tubemill combination due to steeper grade of launders or to the increased volume of water to be handled. It must not be forgotten, too, that the smaller and more numerous the units, the less the effect of one or more units being out of action.

Dr. W. A. Caldecott in his reply to discussion noted the following points, among others:

The suggestion that crushing plants comprising both stamps and tubemills should be considered as a whole is, of course, correct, and this view has been usual on the Rand since the introduction of secondary crushing appliances in the form of tubemills. In this connection the author has found very convenient for purposes of calculation what may be termed a "nominal crushing unit" (N. C. U.) defined as a stamp with a running weight of 1250 lb. and an assumed new weight of 1350 lb. The number of drops per minute is assumed to be 100, the set height of drop 8 in., and the actual height of drop 7.5 in. A tubemill 5½ ft. x 22 ft. is assumed to be equal to 30 such N. C. U., and likewise requires about 100 hp to operate; this ratio of stamps to tubemill is not constant, but varies somewhat according to the conditions of operating. The capacity of the above nominal stamp is rated by the tons of —90-mesh product it yields per 24 working hours, and the capacity of stamps with different running weights, but operating under similar conditions is calculated in proportion to such different running weight, while the capacity of tubemills of other dimensions than those cited is taken in proportion to the cubic capacity of their shells. Thus a stamp with 1400 lb. running weight is equal to 1.12 N. C. U., and a tubemill 14 ft. long by 4 ft. diameter is equal to 10 N. C. U. This simple rule is of value in determining the crushing plant required when a final pulp containing a given percentage of —90-mesh product is desired. Thus, to estimate how many tubemills (5½ ft. x 22 ft.) must be added to a plant containing 200 stamps with a running weight of 1400 lb. in order that 1420 tons daily of pyritic banket ore should be crushed to a pulp containing 80 per cent —90-mesh product, and assuming likewise that an N. C. U. produces 3.3 tons —90-mesh product per 24 hours:

80 per cent of 1420 tons = 1136 tons —90 product.
200 stamps at 1400 lb. = 224 N. C. U.
224 N. C. U. × 3.3 tons = 739 tons —90 product.

$$\frac{1136 \text{ tons} - 739 \text{ tons}}{30 \times 3.3} = \text{four tubemills } (5\tfrac{1}{2} \text{ ft. x } 22 \text{ ft.})$$

The arrangement in the City Deep mill of multiple camshaft bearings with 2000 lb. stamps, which have an estimated 11-ton duty, constitutes a logical means of coping with the shock due to impact of cam with tappet, which naturally increase in severity with increased stamp weight. Whether some form of lifting device other than the cam will finally be adapted to the relatively small crushing unit which the present stamp constitutes it is difficult to say, or whether the encouraging results of recent coarse crushing trials, which are given later in this reply, will result in the use of large pneumatic or other stamps with coarse screening, and comparable in duty with the steam

stamps on Lake Superior. This last system, however, appears the most promising, and it is to be hoped that the pioneer work already carried out at the New Kleinfontein mill in this direction will be further developed. With the elimination of amalgamation in the battery, rolls to reduce ore to pass a screen of nine holes per square inch would produce a suitable product for tubemilling at a low capital cost, could steel be found for shells which would preserve true faces by withstanding abrasion well, and thus minimize stoppages and cost of truing the roll faces. The comparative advantage of the stamp in this respect is that even though the crushing surfaces of shoe and die wear irregularly, yet a fair crushing efficiency is maintained.

The author has recently carried out the following tests to further develop certain points referred to in the paper:

Five stamps with long heads and short stems were extemporized by joining two 24-in. heads together by means of stout pins, and were operated in a mortar box with the front cut out and replaced by a thin detachable steel plate. A wood and canvas housing prevented splash above the mortar box through which the heads projected. This design of stamp was on the lines of the early experiments carried out at the Knights Deep of increasing the weight of stamps by interposing a false shoe between the head and the true shoe, although the advantages of such a design were not then realized. The open front mortar box facilitated access to the head when required, and the long head combined with short stem lowered the center of gravity of the stamp to an appreciable and desirable extent as compared with the usual type. While the stamp in question was of a heavy type, obviously the design is applicable to stamps of any weight and has now been adopted for the new City Deep mill.

A still more important investigation conducted by Mr. W. R. Dowling, at the Simmer East mill, concerned the maximum size of particle which could be efficiently pulverized in the usual tubemill, 5½ ft. x 22 ft., or, in other words, the coarsest battery screen practicable under these conditions. Briefly, the maximum efficient aperture was found to be 0.27 in., equivalent to battery screen of nine holes per square inch, and besides obtaining very high stamp duties, the actual tonnage per day of the desired —90-mesh product of stamps and tubemills was further found to be much greater by using such coarse screening than with the finer battery screens in common use. The fact that pulp from a coarse battery screen should contain more tons of fine product per stamp than that from a fine screen would hardly be expected, but though the percentage of fine product in the former pulp is low, yet this is more than offset by the high stamp duty. The general improvement in crushing efficiency is really due to applying the principle of promptly removing from the mortar box all particles too small for the stamp to readily reduce further, and by feeding the coarser of these to the tubemill, providing the latter with material which is within the limits of its most efficient operation.

In the Simmer East investigations some trials of 50 stamps with coarse battery screening (16 and 36 holes per square inch) and the remaining 80 stamps with fine screening (1000 to 1600 holes per square inch) were first carried out with encouraging results, though the scouring of the amalgamated plates with this coarse product was found to be excessive, and in subsequent trials the plates were covered with linoleum. The cyaniding results of the trials were normal and in proportion to the ultimate fineness of the pulp.

In all trials the usual tubemill circuit was in operation whereby the coarse portion of the pulp after elevation was separated by cone classifiers for tubemilling. The tubemill outflow, after passing over shaking amalgamated plates, rejoined the pulp from the batteries for re-elevation and separation of any coarse particles, which again passed through the tubemills, while the fine material overflowing the classifiers gravitated to the cyanide plant

In the subsequent trials, only stamps with coarse screening were used, and the tonnage of ordinary mill feed crushed was calculated from numerous measurements of ore supplied to the batteries from a bin, which was partitioned off for the purpose, and the surface of the ore levelled before and after each run, 20 cu. ft. of ore being taken to the ton. The duties thus found were checked where practicable and with substantially the same results, by the tonnage of sand and slime collected during each run. Fifty stamps and three tubemills were available for the trials and were run in carrying ratios to each other.

Trials were made with battery screen of four holes per square inch, but it was found that with this product the tubemills discharged very coarse material, which accumulated in the tubemill circuit, many particles in the pulp being apparently too large to be shattered by the falling pebbles, and being nearly rounded by abrasion during their passage through the tubemill.

Regular and frequent sampling of battery pulp during each run was carried out to obtain samples for grading analyses and to determine the ratio of water to solids. The grading of the final pulp was calculated back from the grading of the sand charges, in accordance with the percentage by weight they constituted of the ore milled. The production of —90-mesh product per stamp was determined from the percentage of —90 mesh in the battery pulp and the stamp duty, while the product of —90-mesh product per tubemill was calculated from the tonnage of ore milled during each run, divided by the number of tubemills in operation, and multiplied by the difference between the percentage of —90 product in the battery pulp and the final pulp.

The average running weight of the stamps in all trials was 1400 lb., with 8 in. set height of drop at 98 drops per minute, and 2 in. height of discharge, while all mortar boxes were fitted with back-water feed as well as the usual supply from above. The utility of the back-water feed in coarse crushing may be attributed to its washing toward the screen particles which are fine enough to pass through, though too coarse to readily become suspended in the splashed pulp.

The increased efficiency of tubemilling, as measured in tons of —90-mesh product, with coarse battery screening and a consequent coarse tubemill feed, may be explained as due to the fact that such coarse particles provide work to be done by the falling pebbles throughout the full length of the mill, whereas with a finer feed many particles are early reduced to the required size and hence the pebbles near the outlet of the mill are not supplied with sufficient coarse particles to secure their full crushing efficiency.

Oildag and Graphite.—The Acheson Oildag Company, of Niagara Falls, N. Y., has purchased a factory site of 30 acres at Port Huron, Mich., and will erect there their prospective plant. Port Huron was selected as the site because of its comparatively pure water and excellent shipping facilities. Oildag has made phenomenal progress in the matter of establishing itself as a superlative lubricant and conserver of petroleum lubricating oils, but owing to the past limited manufacturing capacity of the company, the price was of necessity high and retarded its rapid introduction into general lubrication. On the first of July, the Acheson Oildag Company reduced the selling price of Oildag to approximately 60 per cent of that previously quoted, thereby bringing it down to a price, when bought in quantity, of considerably less per gallon than that of the oil with which it is mixed. The presence of the deflocculated graphite in the oil makes each gallon equal to 2 gal. to 4 gal. of plain oil. The International Acheson Graphite Company has arranged to erect a very substantial and large addition to its works at Niagara Falls. This will afford capacity for 10 or 12 new electric furnaces. The buildings will be of brick and steel construction and erected to the west of its present works, where there is ample room for still further enlargement. The additional furnaces will make possible a very large increase in the output of graphite rendered necessary by the very rapidly increasing demand.

Analysis of Manganese Bronze.

By H. B. Swan.

The analysis of this well-known and important alloy necessitates the determination of tin, copper, iron, aluminium, manganese and zinc. Tin, iron, aluminium and manganese are present in relatively small amounts, and inasmuch as a difference of a few tenths of 1 per cent of these ingredients may materially alter the physical properties of this bronze, it is extremely important at times to have accurate and reliable results. The method outlined below has been worked out in this laboratory; it has given excellent results and is comparatively rapid, a period of about six to seven hours being required for the complete analysis.

A 5-gram sample of drillings is weighed out into a No. 2 beaker, 20 c.c. of water added, and then carefully 20 c.c. of HNO_3 (specific gravity 1.42). Cover, and after the reaction has subsided, heat to complete solution, and boil for about five minutes. This strength of acid has been found most satisfactory, as in using a more dilute solvent the precipitate of metastannic acid runs through even a double filter.

Filter off the precipitate of metastannic acid through a double filter of "S & S" 589 white ribbon paper, using a platinum cone and employing suction. Wash the precipitate well with half-and-half nitric and allow the suction to act sufficiently long after the paper is drained to dry it somewhat. Ignite the precipitate and paper together, charring carefully, and afterwards raising the heat to the full heat of a Bunsen or Tyrell burner. Calculate the tin from the precipitate of SnO_2. It is not necessary to subject this precipitate to any further treatment to purify it, as it is so small that only traces of iron or manganese can be precipitated along with the tin.

To the filtrate is added 5 c.c. of concentrated H_2SO_4 (sp. g. 1.84) and the whole evaporated on the hot plate or bath to almost gentle dryness. That is to free it from the nitric acid, and occasionally, if a low-grade spelter is used in the manufacture of the alloy, to precipitate any lead present.

If there is an appreciable amount of lead present and it is desired to determine it, after cooling the crust of sulphates in the bottom of the casserole, about 50 c.c. of water are added and the sulphates will readily dissolve with a little heating. The lead sulphate is filtered off on to an asbestos pad in a platinum gooch crucible, washed with a 5 per cent solution of H_2SO_4, then, after removing the filtrate, washed with alcohol into a clean flask. The crucible is then ignited at a low heat, but sufficient to remove any H_2SO_4 that may still be present.

The filtrate is then transferred to a 500 c.c. graduated flask and diluted to 500 c.c.

For copper either pipette off 100 c.c. or measure out, by means of a 100-c.c. graduated flask, 100 c.c. of the solution, and put into a tall non-lipped 150-c.c. beaker. Just neutralize with NH_4OH and then add about 3 c.c. of a mixture of three parts nitric (sp. g. 1.42) to one of sulphuric acid (sp. g. 1.84) and submit the solution to electrolysis. This laboratory uses a gauze cathode and a rotating anode, and employs a current of $N.D._{100} = \frac{3}{5}$ amp. After testing with H_2S on a spot plate to insure the complete removal of the copper, the cathode is washed, as it is removed from the solution and after removing, with H_2O, and then dipped in alcohol and dried. A quick and very satisfactory method of drying consists in holding the cathode above a piece of asbestos, covering the top of a Tyrell burner, and at the same time allowing a current of compressed air (not too strong) to play over the gauze.

Two 100-c.c. portions of the solution are run simultaneously, and, after the removal of the copper, iron is determined in one portion and aluminium in the other. To one portion add 5 grams of ammonium chloride and make just slightly alkaline with NH_4OH and heat to the boiling point. Cool, allow to settle and filter. Iron and aluminium are precipitated. Wash with dilute ammonium chloride solution. The precipitate is redissolved in 5 c.c. of HCl (1.12) and the solution diluted up to 400 c.c. in volume. Add 10 c.c. of a 10 per cent solution of sodium ammonium hydrogen phosphate and then NH_4OH till just slight but permanent precipitate forms. Then add 1½ c.c. of strong HCl and then 10-15 c.c. of a 20 per cent solution of sodium thio sulphate. Heat to boiling, and after the boiling point is reached add a mixture of 15 c.c. of a 20 per cent solution of ammonium and 8 c.c. of strong acetic acid. Boil for about ten minutes, all to settle, and filter. The precipitate filters readily, but is hastened by using suction. Wash several times with hot water. The paper and precipitate are burned together and then ignited over the blast lamp, either in a porcelain or platinum crucible. Weigh as $Al\ PO_4$, the conversion factor being 0.2219 for aluminium.

The iron, aluminium and manganese are precipitated together in the other 100-c.c. portion with NH_4OH. The precipitate redissolved in about 5 c.c. of strong HCl. The iron reduced with stannous chloride and then titrated with a solution of $\frac{N}{80}$ $K\ Mn\ O_4$, using a guard solution made by mixing phosphoric acid, manganese sulphate, and sulphuric acid together. For details of this method see Blair's methods for the analysis of iron and steel.

Now two other 100-c.c. portions of the solution are used for the determination of manganese and zinc. The copper may be removed either with metallic aluminium or by electrolysis. In the latter case the copper is removed from the sulphuric acid solution without the addition of ammonia or nitric acid, but adding about 1 c.c. of H_2SO_4 for a total volume of 125 c.c. of solution.

After removing copper, for manganese add a few c.c. of strong bromine water and make ammoniacal and then add a slight excess. Heat till all the iron, aluminium and manganese are precipitated, then filter, wash and dissolve in a few c.c. of sulphurous acid. Now add to the solution 10 c.c. of HNO_3 acid and 30 c.c. of H_2O and boil a minute, then add about ¼ gram of sodium bismuthate and heat till the pink color completely disappears. Add a few drops of sulphurous acid to clear the solution and boil again to expel all nitrous fumes. Cool the solution till quite cold and add a slight excess of sodium bismuthate again. Agitate a minute or two, then add 50 c.c. of 3 per cent HNO_3 and filter through asbestos on a platinum cone, using suction. Wash with 50 c.c. more of 3 per cent HNO_3 and then reduce with ferrous ammonium sulphate and titrate with a solution of $K\ Mn\ O_4$, which has been standardized against the ferrous ammonium sulphate solution.

This is merely an application of the well-known bismuthate method for very small amounts of manganese and the details may be found in any standard manual for volumetric work. It is said to be the most accurate method for manganese determinations in which the manganese does not exceed 2 per cent. In manganese bronze the manganese does not usually exceed 0.25 per cent.

Zinc is determined in the other c.c. portion of the solution from which the copper has been removed, without the addition of nitric acid. Zinc may be estimated either as the pyrophosphate, which is the more accurate method, or by electrolysis, which is the more rapid, but not quite so accurate. When selecting the pyrophosphate method, separate the zinc by hydrogen sulphide from a slightly acid solution of formic acid. This is done by neutralizing the solution with Na_2CO_3 and then adding ½ c.c. of 50 per cent formic acid. Now run in H_2S till all the zinc is precipitated.

Zinc sulphide is precipitated free from iron, manganese, and aluminium, and can be filtered quickly and completely in a short time by using asbestos on a platinum cone and employing suction. The ZnS is then dissolved in HCl and precipitated as the phosphate with $NaNH_4HPO_4$.

If the electrolytic method is selected, the zinc is plated out from the solution made sufficiently alkaline to redissolve the precipitated zinc and aluminium. The iron which is precipitated along with the zinc and aluminium by the sodium hydrate

is filtered off before the electrolysis is proceeded with. It is best to boil the alkaline solution to free the iron hydrate from any zinc. The filtrate from the iron is then cooled and the zinc plated out on a copper-plated cathode, with the use of a rotating anode.

By the use of the method outlined above we repeatedly obtain results which check closely and total from 99.85 per cent to 99.99 per cent.

Lumen Bearing Company,
Buffalo, N. Y.

The Chemical Export Trade of Germany.

By W. Pollard Digby.

Of all national manufacturing industries the chemical trade of Germany would probably be selected by an impartial observer as the most characteristic of Teutonic application and adaptiveness. Her racial attributes of co-ordination, disciplined organization, and painstaking research, no less than her splendid educational systems, have given Germany her just pride of place in this branch of the industrial arts.

Students of German conditions are apt to urge that her military system has dwarfed her initiative, and that for the original pioneer work in engineering, in metallurgy and in chemistry, we must look to nations like England, France or Sweden, whose natural initiative and whose preliminary discoveries have been the inspiration upon the lines of which the less brilliant but more patient Teuton has plodded along and has achieved pre-eminence.

Even though the charge of lack of initiative has, in the writer's opinion, still some basis, there has to be set against this the advantage of an imagination which is educated, logical and scientific, which seizing on any vantage ground advances from it in an ordered manner to a definite end. One cannot, indeed, speak or even think of German chemical industry without thinking of the aniline dye industry, founded on British neglect of

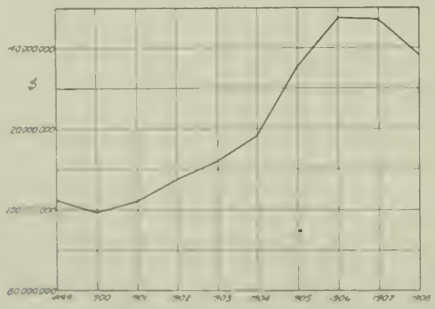


FIG. 1.—TOTAL EXPORT TRADE (CHEMICAL) 1899-1908.

Perkin's discoveries, and of the success of the synthetic indigo industry with its dire effect on Indian indigo planters.

Co-ordination may be said to be the keynote of German success. It requires the best type of specialist to take the fruits of one branch of scientific research, and, securing them in another field, to establish yet fresh industries. The present position of the incandescent lamp may be said to be due to a harking back along the road traversed, and attacking with new methods and new material obstacles formerly regarded as inseparable. At first a matter of metallic filaments in the shape of platinum wire, then a sphere occupied wholly by the carbon-filament lamp. But where other countries were concerned in developing the latter, the Teuton scientist, using the rarer metals, discovered or cheapened by the discoveries in metallurgical chemistry the metallic-filament lamps.

Co-ordination has, moreover, not been wholly educational and scientific. Systematized transport has done much for the development both of internal industries and of external trade. Sociological conditions, to which it is beyond the scope of this article to allude, have also contributed.

As in England and in the United States, local industries have been intimately associated with the chemical trade, and Germany's advances in the textile, porcelain and paper industries have profited largely by the advances made in chemistry.

Taking the 10 years, 1899 to 1908, as in previous articles, we find that the export chemical trade reached a maximum of $147,551,000 in 1906 and a minimum of $99,404,000 in 1900, the average being $121,077,000.

Fig. 1 shows the annual fluctuations in the total trade, while the following table shows the percentage ratio between the chemical and total exports, as well as the estimated population for each year and the value of the chemical exports per head.

	1899.	1900.	1901.	1902.	1903.
Chemical exports	$102,467,000	$99,404,000	$102,092,000	$107,603,000	$112,001,000
Total exports	1,051,750,000	1,152,850,000	1,107,850,000	1,169,450,000	1,253,650,000
Percentage ratio	9.75	8.62	9.23	9.28	8.94
	1904.	1905.	1906.	1907.	1908.
Chemical exports	$118,375,000	$135,585,000	$147,551,000	$147,113,000	$138,514,000
Total exports	1,305,650,000	1,432,900,000	1,589,750,000	1,711,300,000	1,599,625,000
Percentage ratio	9.07	9.45	9.28	8.60	8.66

The general tendency of the first set of figures is to decrease, while the latter shows a more upward tendency.

	1899.	1900.	1901.	1905.
Population	55,248,000	56,367,178	56,874,000	57,767,000
Value of chemical exports per head	1.86	1.77	1.80	1.87
	1903.	1904.	1905.	1906.
Population	58,629,000	59,475,000	60,641,278	61,177,000
Value of chemical exports per head	1.91	1.99	2.24	2.41
			1907.	1908.
Population			62,083,000	62,982,000
Value of exports per head			2.37	2.20

In the first of these articles it was pointed out how the prosperity or otherwise of the chemical industry depended to a very large extent on the activity or stagnation of the other industries which are directly or indirectly connected with it, and hence it is that the rise and fall of the figures for the total chemical output affords, like the market prices of copper, a very close guide to the general trend of commercial activity, and the marked fall in the export figures for the years 1907 and 1908 correspond with the general depression in the industrial world which has characterized the latter part of last decade.

The total figures given above include, as before, drugs and dyes, as well as chemicals in the stricter sense of the word.

Turning now to a more detailed examination of the most important chemicals, the exports have as before, apportioned to the six zones: Great Britain, rest of Europe United States, British North America, other British Colonies, South America, and the rest of the world, as affording a convenient and instructive method of subdivision.

Figures for **Sulphuric Acid** are low—Great Britain and the United States, the chief consumers, making their own in large and increasing quantities, require none from Germany.

TABLE I.—GERMAN CHEMICAL EXPORTS.

Sulphuric Acid.	1904.	1905.	1906.	1907	1908.
Great Britain					
Rest of Europe	$500,000	$450,000	$561,250	$566,500	$708,000
United States					
Brit. North America					
Other Brit. Colonies					
South America					
Rest of World	100,000	100,000	0 250	57,250	40,250
Total	$600,000	$550,000	$661,500	$623,750	$748,250

Strassfurt (Saline) Deposit.—This, an export trade in raw material rather than in a finished product, is a large and growing industry, the United States being the best customers. Separate enumeration was not made previous to the year 1906.

TABLE II.

Strassfurt (Soline) Deposit.	1906.	1907.	1908.
Great Britain	$388,750	$441,000	$495,000
Rest of Europe	1,347,250	971,500	1,891,000
United States	1,994,500	2,112,000	1,869,250
British North America			
Other colonies			13,000
South America			
Rest of World	37,250	780,000	22,000
Total	$3,767,750	$4,304,500	$4,200,250

Caustic Soda, Soda Ash and Caustic Potash.—Great Britain is the chief producer of caustic soda, and as far as soda ash is concerned "Rest of Europe" accounts for the major portion of the German export trade.

For potassium salts Germany is preëminent, this being the one point in which Germany has greater natural advantages than England in the shape of mineral deposits necessary for this industry. The detailed tables of figures for these chemicals are set out below:

TABLE III.

Caustic Soda.	1904.	1905.	1906.	1907.	1908.
Great Britain					
Rest of Europe	$175,000	$175,000	$247,750	$340.250	$353,000
United States					
Br. Nth. America					
Other B. Colonies					
South America					
Rest of World	100,000	125,000	68,000	44,500	60,250
Total	$275,000	$300,000	$315,750	$384,750	$413,250

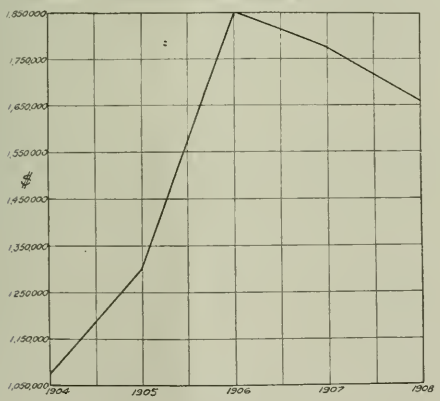


FIG. 2.—EXPORT TRADE IN POTASSIUM AND SODIUM CYANIDE (1904-1908).

TABLE IV.

Soda Ash.					
Great Britain	$25 000	$25,000			
Rest of Europe	900,000	900,000	$830,000	$757,000	$1,268,250
United States					
Br. North America.					
Other B. Colonies.					
South America					
Rest of World	50,000	75,000	56,250	27,500	12,500
Total	$975 000	$1,000,000	$886,250	$784,500	$1,280,750

TABLE V.

Caustic Potash.					
Great Britain	$450,000	$300,000	$334,000	$382,500	$323,250
Rest of Europe	1,525,000	1,475,000	1,394,750	1,201,500	1,698 750
United States	125,000	100,000	132,500	154,500	143,000
Br. North America.					
Other B. Colonies.					
South America					
Rest of World	75,000	75,000	43,250	34,500	27,750
Total	$2.175,000	$1,950,000	$1,905,000	$1,773,000	$2,192,750

Bleaching Powder.—These figures show a down-grade tendency; it is quite possible that the success of the Castner-Kellner process in England may partly account for this, as the demand for this product is still as great, if not greater, than ever.

TABLE VI.

Bleaching Powder.	1904.	1905.	1906.	1907.	1908.
Great Britain	$250,000	$275,000	$253,000	$164,250	$148.750
Rest of Europe	100,000	125,000	167,000	163,250	188 250
United States	175,000	225,000	183,750	175,750	142,500
Br. North America.					
Other B. Colonies.					
South America					
Rest of World	150.000	75,000	62,000	34,250	58,750
Total	$675 000	$700.000	$665,750	$537,500	$538,250

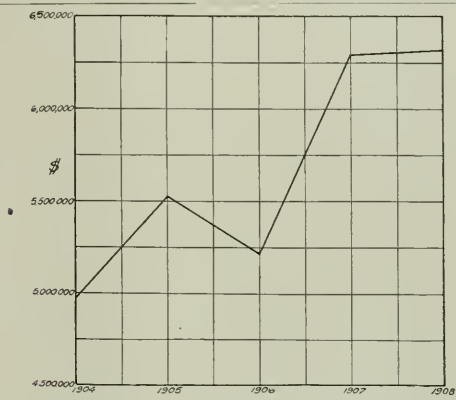


FIG. 3.—EXPORT TRADE IN POTASSIUM CHLORIDE (1904-1908).

Sulphate of Copper.—Here again the export trade is not large, that to Great Britain being too small to merit enumeration. The value for the five years under notice is as follows:

TABLE VII.

Sulphate of Copper.	1904.	1905.	1906.	1907.	1908.
Great Britain					
Rest of Europe	$175,000	$125,000	$333,500	$251,000	$271,500
United States					
Br. North America.					
Other B. Colonies.					
South America					
Rest of World	50,000	75,000	60,750	44,250	44.000
Total	$225,000	$200,000	$394,250	$295,250	$315 500

Sodium and Potassium Nitrate.—In the case of these two chemicals we have an example of a large and increasingly important industry. In the case of the sodium salt, the export to Great Britain is not large, but the trade to the United Kingdom in potassium nitrate is considerable, while the total trade, as the following table shows, has more than doubled since 1904.

TABLE VIII.

Sodium Nitrate.	1904.	1905.	1906.	1907.	1908.
Great Britain			$18,000		$39.750
Rest of Europe	$925,000	$1,000,000	1,121,250	$1,481,000	1,301.250
United States					
Br. North America.					
Other B. Colonies.					
South America					
Rest of World	125,000	75,000	40,750	69,000	74,250
Total	$1,050 000	$1,075,000	$1,184,000	$1.550,000	$1,415,250

TABLE IX.

Potassium Nitrate.	1904.	1905.	1906.	1907.	1908.
Great Britain	$350,000	$400,000	$354,500	$578,500	$488,250
Rest of Europe	450,000	525,000	461,250	511,250	365,000
United States					
Br. North America.			14,000	23,250	42.250
Other B. Colonies.			21,000		
South America	50,000	50,000	72,500	77,250	78.000
Rest of World	250,000	325,000	238,250	240,000	201,250
Total	$1,100 000	$1,300,000	$1,161,500	$1,480,250	$1,174,750

Carbonate of Potash is an industry of increasing importance. The United States trade accounts for about one-half of the total exports.

TABLE X

Carbonate of Pota	1904.	1905.	1906.	1907.	1908.
Great Britain ..	$200,000	$200,000	$200,250	$228,500	$222,500
Rest of Europe.	175,000	200,000	236,250	331,000	356,500
United States...	400,000	525,000	543,000	611,500	522,750
Br. North America.					
Other B. Colonies.					
South America..					
Rest of World...	175,000	125,000	90,750	70,750	64,500
Total	$950,000	$1,050,000	$1,076,250	$1,241,750	$1,162,250

Sodium Sulphate and Potassium Sulphate.—Separate enumeration of the latter was not given previous to 1906. The large trade in the latter salt with the United States is worth noticing

TABLE XI.

Sodium Sulphate.	1904.	1905.	1906.	1907.	1908.
Great Britain ..					$32,000
Rest of Europe ..	$250,000	$325,000	$436,500	$453,500	518,500
United States ..					
Br. North America.					
Other B. Colonies.					
South America....					
Rest of World....	50,000	50,000	45,250	66,000	38,250
Total	$300,000	$375,000	$481,750	$530,500	$588,750

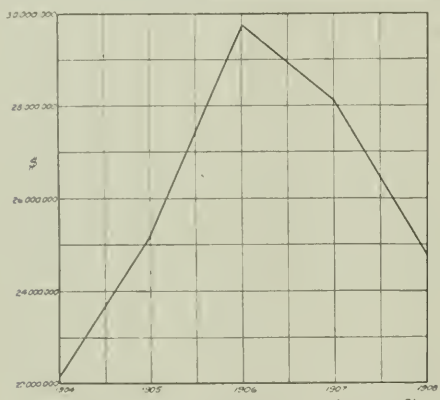


FIG. 4.—EXPORT TRADE IN ANILINE DYES (1904-1908).

TABLE XII

Potassium Sulphate			
Great Britain.	$148,500	$238,500	$588,750
Rest of Europe	312,500	462,500	495,000
United States	1,045,500	988,750	1,073,000
British North America			
Other British Colonies	31,000	92,250	45,750
South America		34,750	13,750
Rest of World....	126,000	114,250	198,000
Total	$1,663,500	$1,929,000	$1,914,250

Potassium and Sodium Cyanide.—There is, of course, a large trade in these products to Africa and other mining regions, the total export value amounting in 1906 to nearly $2,000,000. A curve is given in Fig 2 of the total trade for the years 1904-1908.

TABLE XIII.

Potassium and Sodium Cyanide.	1904.	1905.	1906.	1907	1908.
Great Britain	$50,000	$25,000	$47,500	$51,250	$52,250
Rest of Europe..	50,000	50,000	99,000	46,500	76,250
United States....	225,000	300,000	458,000	488,000	491,000
Br. North America.				20,250	
Other B. Colonies	525,000	775,000	1,067,000	1,019,750	924,750
South America					
Rest of World	225,000	150,000	180,250	107,750	115,750
Total	$1,075,000	$1,300,000	$1,851,750	$1,773,500	$1,660,000

Potassium Chloride.—This, as a necessary re-agent in the manufacture of potassium chlorate, is a very important and growing export product, as the following and Curve No. 3 shows the value of the total trade in 1908 annuity to as much as $6,320,750

TABLE XIV.

Potassium Chloride.	1903.	1905.	1906.	1907.	1908.
Great Britain	$450,000	$550,000	$459,250	$491,500	$419,250
Rest of Europe...	1,550,000	1,750,000	819,000	1,956,750	2,183,750
United States.....	2,975,000	3,100,000	3,871,250	3,792,500	3,646,250
Br. North America.					
Other B. Colonies.			24,250	30,250	29,250
South America....					
Rest of World....		125,000	44,750	23,500	42,250
Total	$4,975,000	$5,525,000	$5,218,500	$6,294,500	$6,320,750

Aniline Dyes and Indigo.—The trade in aniline dyes is, of course, the crowning triumph of the German chemical industry. Thirty-eight millions of dollars is indeed a figure to be justly proud of as the total value of one year's export. Figures for the last two years have fallen slightly, and there is the prob-

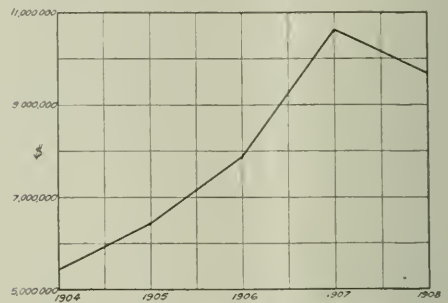


FIG. 5.—EXPORT TRADE IN INDIGO (1904-1908).

ability that the new English patent laws will not be without effect in lowering the value of the German exports to the United Kingdom. The distribution is shown in the following tables and a curve shows the fluctuation in the totals.

TABLE XV.

Aniline Dyes.	1904.	1905.	1906.	1907.	1908.
Great Britain..	$4,125,000	$4,525,000	$4,980,000	$4,978,250	$4,321,250
United States..	4,575,000	5,050,000	6,655,250	6,537,750	5,244,250
Rest of Europe.	6,850,000	10,175,000	11,547,500	9,778,250	9,917,250
Br. N. America.	75,000	200,000	138,000	139,000	111,500
Other Br. Col.	1,175,000	1,325,000	1,342,500	1,354,000	1,459,250
South America.	625,000	700,000	510,000	637,000	579,250
Rest of World.	4,725,000	3,200,000	5,162,000	4,688,750	3,149,750
Total	$22,150,000	$25,175,000	$29,761,250	$28,113,000	$24,773,500

TABLE XVI.

Indigo.					
United Kingdom.	$475,000	$625,000	$645,000	$513,250	$668,250
Rest of Europe.	2,350,000	2,225,000	2,549,000	2,890,500	2,506,500
United States..	875,000	1,025,000	1,126,500	1,363,000	1,182,750
Br. N. America.			17,750	28,000	14,250
Other Br. Col.	25,000	100,000	100,250	140,250	97,000
South America.			23,000		
Rest of Europe.	1,700,000	2,450,000	3,432,250	5,701,500	5,105,000
Total	$5,425,000	$6,425,000	$7,893,750	$10,645,500	$9,663,750

Finally, a table is added showing the total distribution of chemical exports, from which it will be seen to what an extent Germany depends on Europe for its trade, Canada and South America consuming almost a negligible amount in proportion.

Total Distribution.

	1904	1905.	1906.	1907	1908
Great Britain.....	$6,375,000	$6,925,000	$7,814,750	$8,067,500	$7,700,250
Rest of Europe...	15,975,000	18,500,000	22,463,750	22,170,750	22,633,000
United States	9,350,000	10,325,000	14,283,750	14,860,750	13,133,000
Br North America			14,000	43,500	42,250
Other B. Colonies.	725,000	2,200,000	2,586,000	2,636,500	2,569,000
South America ...	675,000	750,000	631,500	747,000	667,000
Rest of World. ..	1,350,000	1,175,000	1,185,750	1,797,500	1,008,750

Silver Shipments.—The output of the Rio Plata Mine, in the Arteaga district of Chihuahua, Mexico, last month, was 55,779 oz. of silver. Nine hundred and forty-six tons of ore were treated. Sixty-eight bars, containing 70,000 oz. of silver, have been shipped to the city of Chihuahua since the first of July

The Outcome of the Calcium Carbide Suits.

In our September issue, 1909 (Vol. VII, page 379), we noticed the decision of the United States Circuit Court, Northern District of New York, in two suits of the Union Carbide Company versus the American Carbide Company. One was based on the calcium carbide process patent of Thomas L. Wilson, the other on his product patent. Both decisions were in favor of the American Carbide Company, but the Union Carbide Company appealed from the decree of the lower court to the United States Circuit Court of Appeals. The suits have been tried before Judges Lacombe, Coxe and Noyes, and in the suit which is by far the most important one (the product suit) the decree of the lower court has been reversed and the Union Carbide Company has come out victorious.

The decision is written by Judge Noyes. The claim of the patent in question reads: "As a new product, crystalline calcium carbide, existing as masses of aggregated crystals, substantially as described."

The first defence of the defendant was that even if the patent broadly covers crystalline calcium carbide and even if all the calcium carbide of the prior art were amorphous, still there would be no patentable novelty in the crystalline form of calcium carbide; that the patent is not for a new product, but for a new form of an old product, having the same composition, properties and uses as the old.

In viewing the alleged anticipation by Woehler the decision of the court states that the Woehler product was the highest development of the prior art, but it was a black pulverulent mass, worthless commercially and never commercially used. "It would be unfit for use in gas generators and, we are satisfied, would rapidly deteriorate when exposed to the air. The product of the patent is more durable. It is hard, compact and so unlike the powdery mass as almost to amount to a new body. Moreover, we think the complainant correct in its contention that the iridescent surfaces of the crystalline carbide would protect it to some—although, perhaps, slight—extent from atmospheric action.

"It is also quite clear that Woehler published a mere result of a laboratory experiment which was put to no practical use. Crystalline carbide, on the other hand, has been a great commercial success and has furnished the foundation for important industries. . . .

"For these reasons it is held that the product of the patent possesses the requisite patentable novelty. And we must regard this conclusion as not only well founded in law, but as most just. To hold an important discovery which has given to the world a commercially new product—a product the high utility of which must be conceded—not entitled to protection for want of novelty would, as it seems to us, be applying the patent statute to defeat its fundamental purpose."

The defendant's second defence was that the patent, if broadly for crystalline calcium carbide, is invalid, because it is anticipated by the Woehler carbide. The question is whether the defendant has established the crystallinity of the article made by the Woehler process. "Upon this question the testimony of the expert witnesses called by the defendant and complainant is wholly irreconcilable. . . . It would serve no useful purpose to review this conflicting testimony or to discuss the reasons pro and con given by the experts. It is sufficient to say that upon careful consideration of the testimony we are not satisfied that the carbide of the experiment was crystalline. The burden is upon the defendant to establish crystallinity, not upon the complainant to disprove it. . . . Consequently, the Woehler product, whether correctly represented by the earlier compound or by the result of the recent experiments, must be held not to anticipate."

The defendant's third defence was that the patent was invalid because the claimed product was in public use for more than two years prior to the filing of the application. The court decides that while there were experimental uses of the product before that time, yet the proof fails to establish that there was any public use of the invention.

The fourth and last defence was that the defendant does not infringe. The file wrapper of the patent shows that Willson's original claim was as follows: "As a new product, crystalline calcium carbide having a bluish iridescence, substantially as described." This claim was rejected and the claim finally allowed by the Patent Office reads: "As a new product, crystalline calcium carbide existing as masses of aggregate crystals."

The decision of the court as to the meaning of the change of the claims states: "It is said that the applicant in his original application having broadly claimed crystalline calcium carbide and having acquiesced in that rejection of the claim cannot now contend that 'crystalline calcium carbide existing as masses of aggregated crystals' means nothing more than crystalline carbide. And we think this contention correct. The claim is undoubtedly limited to the form of crystalline carbide which exists as masses of aggregated crystals. There must be an aggregation. The claim does not cover an individual crystal nor any number of separate crystals not aggregated or joined together. To illustrate: had there been a question whether the Woehler powdery substance was amorphous or composed of minute, separate and distinct crystals, the claim as amended would have excluded it in either case, while the original claim would have been anticipated by it had it been composed of crystals."

As to the argument of the defence that the limitations of the word "crystalline" is not by the word "aggregated," but by the word "crystals" and that the term "masses of aggregated crystals" is used in a specific and limited sense, the court says: "We are by no means satisfied that any such fine distinction as this—based upon academic definitions in the Williams book—is so generally recognized." . . . "To confine the claim of the patent to masses of perfectly developed crystals would be, we think, by a strained construction to read out of the claim the only product producible by the process of the patent and to deny protection to a meritorious invention.

"In our opinion the patent covers crystalline carbide when the crystals are aggregated in masses, whether such crystals be perfect or imperfect, and as it is admitted that there are crystals in the defendant's carbide, and as those crystals are so aggregated, we think that the product of the defendant infringes.

"The decree of the Circuit Court is reversed with costs and the cause remanded with instructions to enter a decree for the complainant for an injunction, an accounting and costs."

The patent thus upheld by the Court of Appeals is No. 541,138, granted on June 18, 1895, to Thomas L. Willson and assigned to the Union Carbide Company.

* * *

In the second suit, the "process suit," the decision of Judges Lacombe, Coxe and Noyes reads as follows: "The question of infringement in this case depends upon the result of the inquiry whether the defendant employs the arc, as distinguished from the incandescent, principle in its electric furnace in the manufacture of calcium carbide. The burden is upon the complainant to establish by a fair preponderance of testimony—and not by mere scientific speculation—that the defendant does use the arc. We are, however, not entirely satisfied that the complainant has sustained this burden and accordingly feel constrained to affirm, with costs, the decree of the Circuit Court upon the opinion of the judge holding it, and it is so ordered."

Heating of cyanide solutions may be a distinct advantage in increasing the extraction of gold during the agitation of the pulp. A temperature of 90° F. is about the maximum limit. This may be accomplished by steam coils in the tanks, or by the introduction of live steam. At the Montana-Tonopah mill, Tonopah, Nevada, the extraction has been increased 5 per cent by this means, without any additional consumption of cyanide. The method seems occasionally worth considering and trying, especially where waste steam is available.

Hardness Tests on Alloys of Nickel and Copper with Electrolytic Iron—I.

By G. A. Roush.

While hardness as a *prime* factor is limited mainly to tool steels, it is still of sufficiently great importance in so many other materials that any investigation of this property is of interest. The relative hardness of any material varies with the kind of stress to which it is subjected. Hence different methods of measuring hardness, employing different methods of applying the testing stress will give results which are not strictly comparable. A series of samples arranged according to increasing hardness as measured by machining in a lathe will not necessarily remain in the same order when tested by a compression method. For this reason it is desirable that in any particular test the stress applied may be as nearly as possible similar to the stress to which the material will be subjected in use In this investigation, however, no one particular use of the materials in hand was under consideration, so any method by which a series of comparative results could be obtained was sufficient.

The method used for determining the hardness was the scleroscope method devised by Albert F. Shore in 1906. The scleroscope was first put on the market in 1907, and is now used quite extensively in the commercial field as a means of determining hardness. The instrument consists of a vertical glass tube provided with a graduated scale suspended over a steel anvil, on which the sample to be tested is placed; a small

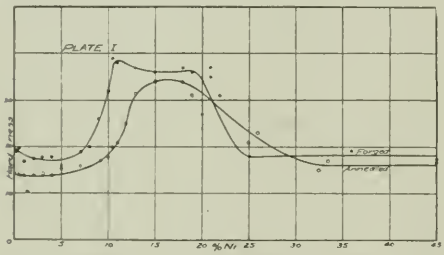


FIG. 1—HARDNESS OF FORGED AND ANNEALED NICKEL ALLOYS.

steel plunger with a diamond point, suspended in the top of the tube, is allowed to drop on the sample clamped on the anvil, the rebound of the plunger being read off on the scale at the back of the tube. When the plunger reaches the surface of the metal under test it carries a certain amount of potential energy, and the weight of the plunger, the height of the fall and the area of striking surface are so proportioned that in all cases this energy is sufficient to exceed the elastic limit of any material which may be under test. This being the case, the material on receiving the impact of the plunger is stressed beyond its elastic limit and given a permanent deformation. If the elastic limit of the material under test is high, the excess energy of the plunger is small, and only a small dent is made on its surface That portion of the material immediately adjoining the deformed portion has then been stressed only up to its elastic limit, and in regaining its original form causes the plunger to rebound a certain distance. If the metal under test has a low elastic limit a smaller portion of the energy of the falling plunger is used in overcoming the elasticity, and a greater portion is used in displacing metal. The stress of the adjoining metal is then only sufficient to give the plunger a small rebound This rebound is then taken to represent the hardness of the material under test.

Basing his conclusions on this line of reasoning, Shore makes the statement[1] that the curve for hardness, as determined by the scleroscope, is very nearly parallel to the curve of elasticity, but since the value obtained in any test for hardness depends to a certain extent on the method used to obtain the result, there is liability for considerable variation.

[1] Amer. Mach. Nov. 14, 1907

Preparation of the Materials.

This investigation covers a series of alloys of nickel and iron up to a nickel content of 50 per cent, and of copper and iron up to a copper content of 8 per cent, being a part of the extended study of electrolytic iron and iron alloys that is being carried on in the Chemical Engineering Laboratories of the University of Wisconsin, under grant from the Carnegie Institute. The results are of particular value in that they are obtained from materials the history of which is completely known, and which have been subjected to a number of other tests.[2]

The preparation of the electrolytic iron is sufficiently familiar that it does not require particular notice here. The iron used in the preparation of the alloys, according to several analyses, showed a purity of about 99.97 per cent, the latest analysis being as follows:

Carbon	0.000
Sulphur	0.003
Phosphorus	0.003
Silicon	0.002 or less.
Manganese	0.020
Iron, by difference	99.972 per cent.

The alloys were prepared by melting the iron with the calculated mounts of the added substance in magnesia crucibles in an electric resistance furnace, as described in the papers mentioned above. The amount of carbon absorbed in the melting has been shown by several different analyses to be less than 0.10 per cent, so that the hardening observed can be accounted as due to the addition elements.

The number of bars tested of each composition varies from 1 or 2 up to 12 or 15, and in each case 10 or more readings were taken on each bar, and averaged for the final result. Considerable variation was noticed in the various bars and in different parts of the same bar, due to irregularities in forging and annealing, but the general averages show a fair degree of uniformity.

Not all of the finished alloys were subjected to analysis, but a sufficiently large number of both the nickel and the copper series was analyzed to make one safe in making the assertion that all were sufficiently close to the calculated composition for the requirements of an investigation of this kind. A few analyses typical of the series are given below.

Nickel Alloys.

Mark.	Calculated.	Found.
144C	0.25	0.27
144F	2.0	1.93
144J	7.0	7.05
157H	12.0	12.07
166G	22.0	22.11
166L	35.0	35.00

Copper Alloys.

Mark.	Calculated.	Found.
158A	0.2	0.202
147H	1.0	1.006
158I	4.0	3.900
147U	7.0	7.050

All samples were tested in the forged condition and after annealing at 600° C.

Nickel-Iron Alloys.

There are no investigations described in the literature which are strictly comparable to the results given here, since in other

[2] See a, Physical Properties of Iron-Copper Alloys, Burgess and Aston, Trans. Amer. Electrochem. Soc., 16, 241-256 1909.
b, The Magnetic and Electrical Properties of Iron-Nickel Alloys, Burgess and Aston, Met. and Chem. Eng., 8, 21-26, 1910.
c, The Magnetic and Electrical Properties of Iron-Copper Alloys, Burgess and Aston, Met. and Chem. Eng., 8, 79-81, 1910.
d, The Strength of the Alloys of Nickel and Copper with Electrolytic Iron, Burgess and Aston, Bull. of Univ. of Wis., Eng. Series, Vol. 6, No. 2.

investigations there have been present considerable quantities of impurities which had to be taken into account while in this case these impurities are kept down to a point where the variations observed may be ascribed to the nickel alone.

The hardness tests, as shown, bear out the general qualitative observations made by Burgess and Aston in their work on the same alloys.[2] In the forged condition up to 7 per cent Ni the curve is concave and lies slightly below the value found for pure iron. From 7 per cent to 11 per cent there is a rapid rise to a maximum, dropping off slightly to 19 per cent, and then a rapid drop up to 25 per cent to about the value for pure iron, where it remains practically constant. In the annealed condition the curve starts at about the value for pure iron, rising gradually to 10 per cent, then abruptly to a maximum at 16 per cent, followed by a sharp decline up to 32 per cent, and then

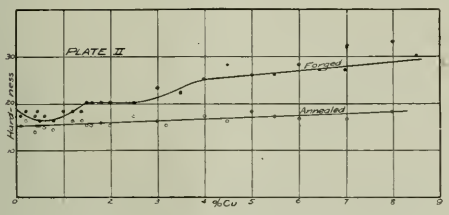


FIG. 2.—HARDNESS OF FORGED AND ANNEALED COPPER ALLOYS.

remains practically constant at a value slightly higher than that of pure iron. As a rule, the variations noted are not large. From 21 per cent to 30 per cent the curve for the annealed samples lies above that for the forged samples. Aside from this, the values found for the annealed samples were about what would naturally be expected as deduced from the curve for the forged samples.

Results on the forged nickel are given in Table I, and on the annealed nickel in Table II, both being plotted together in Plate I.

TABLE I.

Hardness of forged nickel samples.

Per cent Ni.	Number.	Maximum.	Minimum.	Average.
0.25	3	20	15	18
0.5	3	21	17	20
1.0	3	19	15	17
2.0	4	18	17	17½
3.0	5	20	16	18
4.0	5	21	16	18
5.0	5	17	15	16
7.0	5	21	18	19
8.0	5	21	19	20
9.0	6	28	21	26
10.0	5	34	26	32
10.5	3	44	34	39
11.0	8	41	32	38
12.0	5	36	34	35
13.0	6	39	35	37
15.0	3	37	35	36
18.0	3	45	38	37
19.0	2	36	35	35½
20.0	1	..	27	27
21.0	3	40	33	37
25.0	2	20	16	18
36.0	3	20	19	19
45.0	3	20	15	17

Copper-Iron Alloys.

As in the nickel-iron series, quantitative measurements confirm the qualitative observations made on these materials.[4] In the forged condition up to 1½ per cent the hardness curve is concave and lower than that of pure iron. From 1½ to 2½ per cent it proceeds in a straight line, a trifle above that for pure iron, and from 2½ per cent on there is a continued rise. In the annealed condition the hardness increases in a straight line with only a very slow rise throughout the series. In the copper series the variations of the individual readings from the mean were much larger than in the nickel series.

The results from the forged samples are given in Table III and from the annealed samples in Table IV. Both series of results are plotted together in Plate II.

TABLE II.

Hardness of annealed nickel samples.

Per cent Ni.	Number.	Maximum.	Minimum.	Average.
0.25	5	15	14	14½
0.5	5	15	13	14
1.0	4	16	12	14
2.0	4	15	13	14
3.0	5	15	13	14½
4.0	6	15	12	14
5.0	7	17	14	15½
7.0	6	17	15	16
8.0	6	18	15	16
9.0	8	19	15	17
10.0	5	19	17	18
10.5	7	22	18	20
11.0	6	24	18	21
12.0	5	26	23	25
13.0	10	36	26	31½
15.0	3	35	32	34
18.0	6	36	31	34
19.0	5	35	28	31
21.0	2	35	34	34½
22.0	1	..	31	31
25.0	1	..	21	21
26.0	1	..	23	23
35.0	1	..	15	15
36.0	1	..	17	17
45.0	2	19	16	17

TABLE III.

Hardness of forged copper samples.

Per cent Cu.	Number.	Maximum.	Minimum.	Average.
0.1	6	20	17	18
0.2	6	20	16	18
0.4	3	18	17	17
0.45	4	20	17	18
0.5	1	..	16	16
0.6	3	18	16	17
0.8	4	18	15	16
1.0	10	24	16	18
1.2	2	20	16	18
1.4	3	21	17	18
1.5	3	21	19	20
1.6	4	22	18	20
1.8	6	21	19	20
2.0	6	21	17	20
2.5	3	21	19	20
3.0	7	25	21	23
3.5	1	..	22	22
4.0	7	28	22	25
4.5	2	30	26	28
5.0	4	30	23	26
5.5	2	28	25	26
6.0	3	30	25	28
6.45	4	36	25	27
7.0	4	29	24	27
7.5	4	27	24	26
8.0	2	37	29	33
8.5	1	..	30	30

[2] See reference above, 2d, p. 10.
[4] See reference above, 2d, p. 20.

TABLE IV.

Hardness of annealed copper samples.

Per cent Cu.	Number.	Maximum.	Minimum.	Average.
0.1	8	17	14	15
0.2	8	17	13	16
0.4	3	15	12	13½
0.45	7	16	13	15
0.5	2	17	13	15
0.6	3	15	14	14½
0.8	5	15	11	14
1.0	15	22	12	15
1.2	1	..	16	16
1.4	1	..	16	16
1.5	10	16	13	15
1.6	2	15	15	15
1.8	2	16	15	15½
2.0	13	16	13	15
2.5	4	19	16	17
3.0	14	18	14	16
3.2	1	..	15	15
4.0	11	18	15	17
4.5	3	18	15	16
5.0	8	20	15	18
5.5	3	20	16	17
6.0	6	23	14	16½
6.45	6	21	16	18½
7.0	7	19	14	16½
7.5	9	18	11	15½
8.0	2	19	17	18

General Discussion.

Kuranow and Zemezuzny[5] have attempted to correlate the hardness of alloys with the metallographic condition of the material. Their conclusions are that when two metals form a solid solution the hardness increases and the curve passes through a maximum which generally coincides with the maximum electrical resistance; when the two metals form a mechanical mixture of two components the hardness curve will be a straight line; a definite compound may have a hardness either higher or lower than either of the constituent metals. These conditions are fulfilled, however, in neither the copper nor the nickel series of alloys. In Plates III and IV are plotted the hardness and the electrical resistance for comparison.[6] Curve I shows the

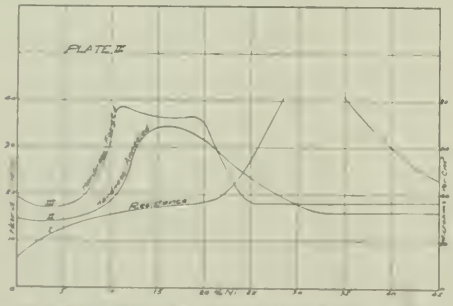


FIG. 3.—HARDNESS AND RESISTANCE OF NICKEL ALLOYS.

resistance in microhms per cubic centimeter, Curve II the hardness of the annealed samples, and Curve III the hardness of the forged samples. In the iron-nickel series a solid solution is formed throughout the series, and a rise in hardness is to be expected. So far the rule holds good, but the maximum hardness is between 10 per cent and 20 per cent, while the maximum in the resistance curve is above 30 per cent.

In the iron-copper series a solid solution is formed up to 3.5 per cent Cu; above that point the two metals are present as a mechanical mixture of two components. According to this a rise of hardness is to be expected up to 3.5 per cent, the curve bearing a maximum at 1.5 per cent, followed by a straight line beyond 3.5 per cent. The hardness, however, in the forged sample lies below that of pure iron, rather than above, the curve bears no definite maximum, and is not a straight line beyond the range of the solid solution. In the annealed samples the straight line curve persists throughout the series.

Portevin[7] refers to the above conditions in his investigation of the electrical resistance of various ternary steels, and after comparing his results with the hardness obtained by Guillet[8] on similar steels, reports a greater variation from this law for the nickel steels than for any of the other materials that he examined.

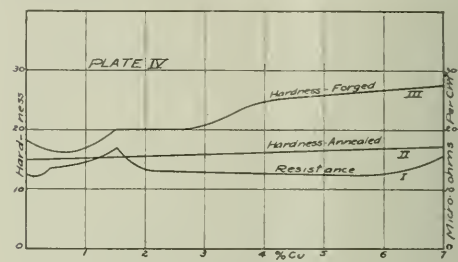


FIG. 4.—HARDNESS AND RESISTANCE OF COPPER ALLOYS.

Conclusions.

From the facts here presented the following conclusions may then be drawn:

1. The addition of nickel to iron gives a maximum range of hardness between 10 per cent and 20 per cent.

2. The addition of copper to iron, up to 8 per cent, gives a gradual, but irregular, increase in hardness.

3. The nickel and copper alloys in the range studied are exceptions to the law of hardness formulated by Kuranow and Zemezuzny.

The results of a similar investigation on the effect of the addition of nickel and copper together to iron are in the course of preparation and will be ready for presentation shortly.

Chemical Engineering Laboratories,
University of Wisconsin.
Madison, Wis.

[5] Rev. de Metall., 6, 1304, 1909.

[6] Les Aciers Speciaux. Dunod, Paris, 1904.

The Northern Chemical Engineering Laboratories is the designation of an enterprise recently started at Madison, Wis., the purpose of which is to carry on experimental and investigation work along lines of chemical engineering and applied electrochemistry. This is under the direction of C. F. Burgess, professor of chemical engineering, University of Wisconsin. Associated with him are Carl Hambuechen, formerly chemical engineer for the Aluminium Company of America; James Aston, formerly owner and manager of a metal foundry business in Milwaukee and later instructor in chemical engineering, University of Wisconsin, and O. E. Ruhoff, recently research chemist for the Mineral Point Zinc Company. It is proposed to continue some of the more promising lines of investigation which have been begun in some of the college laboratories, and the lines of work to which especial attention is being given include the corrosion of metals, metallography, and the investigation of the production and properties of alloys of iron, dry cells, chemical products and various types of machinery and apparatus used in chemical engineering work.

[7] Rev. de Metall., 6, 1304, 1909. Zeit. Anorg. Chem., [illegible] Jour. Soc. Chem. Ind., 93, 1909.

[8] Values for electrical resistance are those measured by Burgess and [illegible] for these same samples see reference 5b.

Elementary Principles of the Designing and Proportioning of Electrical Furnaces.

BY CARL HERING.

The fundamental principle of an electric furnace is so extremely simple, namely, the generation of heat by a current, that one is apt to think that the design of a furnace is an equally simple matter. It is true that heat is a low form of energy to which all other forms naturally degenerate, hence the conversion of the higher forms of energy, like chemical energy (of coal, for instance) or the highly serviceable form called electrical energy, into the degenerated form called heat, is decidedly simpler and generally far more efficient than the reverse. Hence an electrical furnace would naturally be a simpler device than one which performs the reverse operation. But simple as it may be in its fundamental principle, there are many details of design and proportions which make the difference between a good and a bad furnace, and there are systematic and direct methods of design as distinguished from the haphazard, cut-and-try, or roundabout methods which are often resorted to when the fundamental principle is such a simple one. When the necessary constants are known there is no reason why furnaces should not be proportioned rationally and with care, so as to get the best results possible, although the accuracy will never reach that of dynamo design, because there exist no heat insulators like those used to insulate electricity.

The purpose of the present article is to summarize briefly the principles involved in furnace design as far as the proportions are concerned, and to describe the proper sequence in making the calculations, so as to get the best results in the most direct and rational way. Also to point out the ideal conditions which we should strive for even if we can only approach but never reach them.

While a very definite class distinction is often made between arc and resistance furnaces, which distinction is important on account of details, the difference is in fact not so great as might at first appear. The arc is really a gaseous resistor, and the heat is generated by the current passing through these gaseous resistances, which appear to have a very high specific resistance, thereby enabling a very large amount of heat to be generated in a very small space, hence high temperatures; and it is this that gives it its desirable qualities for furnaces. In this feature, therefore, it is merely a resistance furnace with a very small resistor.

There is a characteristic difference, however, in the fact that an arc produces a true counter electromotive force besides the drop of voltage due to resistance; in this it is somewhat like self-induction coils, motors, electrolytic baths, etc. The energy represented by the product of the current and this fractional part of the total voltage is not primarily set free as heat and must not necessarily be included in determining the heat evolved, although often it may finally degenerate into heat even in the furnace itself, in which case it may become available.

This counter-e.m.f. in the arc always represents some other form of energy into which this part of the electrical energy is transformed. In most, if not all, simple arcs it represents that energy which is necessary to volatilize the carbon or other electrode material—that is, to change its physical state. If this vapor passes off as such without condensation or combustion (as, for instance, in a zinc or arsenic furnace) it is lost as far as its heat equivalent is concerned. If it combusts in the furnace its combustion heat is added and may become available. This counter-e.m.f. factor should, therefore, be taken into consideration if large enough; in other respects most of the thermal calculations for arc and resistance furnaces are alike.

The first step in the design of a furnace is to decide upon the output and, of course, the rate per hour, for neither the size of a furnace nor the power input will depend on the former alone; the latter must be specified. A small furnace with a relatively high power input may produce as much product as a large one with a relatively low power input. The choice must be based on the particular process; some processes may be analogous to the hatching of an egg; if hurried, the result will not be the one desired. In general, however, having decided on the desired product per day, the advantages are decidedly in favor of the smallest possible furnace that will yield this product. In other words, the furnace should be relatively small so that it will be forced as much as other considerations permit. The chief reasons are that a small furnace is not only cheaper, but it enables the heat losses through the walls to be reduced.[1]

This should not be confused with another fact, namely, that per cubic inch of internal space the heat loss through the walls is much less for a large than for a small furnace.[2] This means that (other conditions being the same) it is more economical, per pound of material, to produce a large output than a small one, and that it is much better to produce the whole output in one large furnace than in several smaller ones.

The power plant is, of course, smaller the more steadily the furnace is run during the 24 hours. The energy required to heat up a furnace to the stable temperature will probably be found to be relatively very great, and as this is all non-productive it is, of course, much better to use a continuous rather than an intermittent process, or with the latter, to lose as little time as possible between the charges. With correctly proportioned electrodes the loss through the electrodes when the current is stopped, is almost exactly half of that when normal current flows; when not correctly proportioned this stand-by loss may be much greater than one-half.

The next step is to calculate the theoretical amount of energy required for the chemical and physical changes of the specified amount of the product, as that will represent the least possible energy. This, of course, is independent of the furnace design and includes the heat necessary to raise the raw material as it enters to the temperature at which the final products leave, therefore involving the specific heat, generally the latent heat of liquefaction, and sometimes also that of vaporization; to this must be added or subtracted (more generally the former) the energy of chemical combination, if any; that is, the energy represented by the chemical changes which may take place in the furnace product. If this change absorbs energy, the process is an endothermic one and the energy must then be added to the other quantities to give the total which the furnace must supply. The resulting product then is of a higher order, on a thermochemical scale—that is, the product stores up a part of the furnace energy in itself while in the furnace, and is then generally in a condition in which it can again give off this energy later. The production of the metals, iron, aluminium, sodium, etc., from their ores, the production of calcium carbide, the production of steel from cast iron, etc., are all processes of this nature. In fact, it seems that most electric furnace products are of this nature, and sometimes this chemical energy is relatively very large and therefore very important in the calculation.

If, on the other hand, the chemical change in the furnace gives off energy, then that much energy may be subtracted; it is then called an exothermic process. This is less frequently the case, as such processes generally do not require an electrical furnace, because many of them will go on by themselves after starting. If the process is a composite one in which several chemical changes take place, some of which may be thermally positive and some negative, they should, of course, all be included. In simple smelting there is no such chemical change, except that between the metal and the slag, which may be very small.

Besides the energy for these primary chemical changes there may also be some secondary or subsequent thermochemical reactions, such as the heat of combustion of carbon vapor, which may be worth considering.

[1] "Heat Conductances Through Walls of Furnaces." Hering. Trans. Amer. Electrochem. Soc. Vol. 13, fig. 5, p. 225.
[2] Ibid. fig. 6, p. 225.

Having thus determined the least possible quantity of energy required for both the physical and the chemical changes, estimate the first approximation to the electrical input in watts by allowing for the loss of heat by conduction through the walls, and that necessarily lost in the electrodes. The former depends, of course, greatly on the construction of the furnace, its size, etc. Efficiencies of 80 per cent have been claimed, but 50 to 60 would probably be nearer the truth, unless care has been taken in the design.

The next step is to decide what the current and voltage are to be, as so far only their product is known. This also depends greatly on the construction of the furnace, and must be governed by it. Thermically, of course, it makes no difference. The loss in the electrodes increases with the current and is independent of the voltage, hence, if this loss, which may be made very small by proper design, is of any importance, it is best to make the voltage as high as possible, thereby reducing the current. But as this difference is small in percentage, there will probably be other practical considerations which will outweigh it, and, therefore, govern the choice.

In an arc furnace the arc should be made as long as other considerations permit, and two arcs in series would be better than one, as this halves the current. In a resistance furnace the reduction of the current at the expense of the voltage makes long and slender resistors, which are as a rule very objectionable and impracticable. The calculation of the length and cross-section of a resistor of a given material, to set free a certain amount of heat, is too simple to need description here.

The next step is to determine the size of the interior of the furnace. The only rule is to make it as small as practicable to hold the material. Waste space in the interior of a furnace diminishes the efficiency greatly.[3] By properly proportioning the electrodes and the walls so that they will not chill the furnace product, the whole of the interior of a furnace becomes available, hence it may be made much smaller than when the material is being chilled, and it seems likely that in most existing furnaces there is such a chilling action, due to the electrodes, because our former rules for proportioning electrodes tend to make them much larger than necessary, or too short, both of which cause not only a useless loss of heat by conducion, but reduce the available space in the furnace on account of the space taken up by the chilled material.

The correct proportioning of electrodes assumes that the hot end of the electrode can withstand the temperature of the furnace. In some very high temperature furnaces this may not be possible either for the electrodes or for the walls, and in such cases the only way seems to be to allow some of the product next to the walls to remain inactive by conducting off the heat from it as fast as it reaches the material, as is the case, for instance, with the water-cooled tuyers of blast furnaces around which some slag chills and remains solid although in contact with molten slag. In such a case this material must be considered as a part of the walls, like in the carborundum furnace, and not a true, available part of the interior of the furnace proper. Or when such material is at the end of the electrode and is conducting, it must in the same way be considered as part of the electrode which ends (when correctly proportioned) at that place where there is no flow of heat from the useful product out through the electrode—that is, where the temperature equilibrium between the electrode and the furnace is reached.

Having thus established the actual interior size of the furnace, its shape and wall thickness must then be decided upon. As to the shape, the only general rule is that a sphere is theoretically the best, as far as the heat loss through the walls is concerned. Hence, if that loss is of importance the nearer it approaches to a sphere the better; hence, a cubical shape is better than a long rectangle or long cylinder. But in this, as in many other features, practical considerations will often out weigh the gains due to the theoretically best conditions; but it is well to know what the latter are so that they can at least be approached as closely as practical conditions permit.

The thickness of the walls and the loss of heat through them is then determined. The paper referred to above gives further information about this as well as about the question of the best shape. In these calculations it should be remembered that the loss of heat through the walls depends on the *ratio* of the cross-section (perpendicular to the flow of heat) and the thickness, as distinguished from the actual dimensions—that is, the latter may be greatly different in two cases and yet the loss be the same if that ratio is the same.

Having determined the wall thickness, it generally establishes the necessary length of the essential part of the electrodes—that is, the part which must exist to get the current into and out of the furnace. It is this part in which the current may be made to heat the electrode tip, to just the furnace temperature so as to check the outflow or loss of any furnace heat through it, under which conditions the total loss in or through electrodes will be either exactly a minimum or very nearly so. This establishes the cross-section and the loss in the two electrodes. This part of the calculation has been described so fully in other papers by the writer[4] that it is not necessary to repeat it here.

Whatever additional length the electrodes may have must be determined from other considerations, remembering that the electrical resistance heat in the external parts is always a dead loss. Hence, if feeding is required it is best, if possible, to place the terminal close to the wall and feed the electrode throught it, or to make the feeding length as short as is consistent with the labor cost of moving the electrode through its terminal. In the latter case it is preferable to increase the cross-section and therefore also the essential length, as much as practicable, remembering, however, that the essential part must be well embedded in the walls or else the losses will increase and perhaps very greatly.

This completes the first trial calculation. The losses which were assumed at the start have now been calculated more accurately (provided the necessary constants exist) and if the assumed ones are found to be materially different, the proportions may be corrected accordingly, if necessary.

If, now, in the operation of the completed furnace it is found that the necessary temperature is not reached (after giving ample time for the stable state to be reached, especially at first when the brick work may be damp), it means that either the feeding of the material is too rapid or that the losses are too great, or both; in either case it shows that the constants were poor or that an error has been made in the calculations. The temperature will always rise until the heat absorbed or lost is equal to that produced. If, on the other hand, the temperature rises too high, then simply feed in more material, as the furnace then has a greater capacity than that which was assumed.

If the furnace product is found to be chilled around the ends of the electrodes, they are too large in section, or too short; increasing the current would remedy this. If they burn out the walls, because their temperature is higher than that of the furnace, they are too small in section or too long; diminishing the current will correct this.

It is self-evident that the losses in the walls and in the electrodes become greater for higher temperatures; hence, it is a disadvantage to develop a higher temperature than is necessary for the desired chemical reaction or physical change to take place. Therefore, as far as this is concerned, a resistance furnace would generally be better than an arc. The temperature of an arc is believed to be that of the vaporization point of the material of the electrode, hence generally very high and far higher than necessary. On the other hand, the rapidity of the

[3] See reference cited above, p. 314, near bottom, referring to fig. 5.

[4] In the most condensed form in an article entitled "Simple Rules for Proportioning the Electrodes for Minimum Loss," *Electrical World*, June 16, 1906, p. 1308, in which references to other articles are also given.

transference of heat to the product increases when the temperature of the source of heat is much higher. The ideal method would be to generate only that temperature which is required for the chemical reaction, or only slightly more, and then to transfer the heat by rapid circulation; hence, heating from the bottom would be far preferably than from the top, as gravity then assists in this transference of the heat. It seems to be a very slow and expensive method to rely entirely on the differences of temperature for the necessary transference of heat.

This rapidity of transference of the heat may often be a very important factor in determining the rapidity of the treatment of the material, on which in turn the size of the furnace depends; hence, it deserves careful attention. The inavoidable heat losses through the walls and in the electrodes are, of course, continuous, hence, if a pound of the product goes through the furnace twice as fast, for instance, these losses will be only half as great per pound of product.

It is also better, and perhaps far better, wherever possible, to transfer heat from a good conductor (metal) to a bad one (slag) instead of the reverse, as the extraction of the heat from a thin layer of a good conductor in contact with a poor one tends to leave it a still better conductor; while the reverse tends to make most poor conductors still worse approaching an insulator; hence, in the latter case the rapidity of flow will tend to be checked, while in the former it will tend to be augmented.

FIG. 1.—TRAIN LOADED WITH GAS-ENGINE BLOWING UNIT.

The limit of the temperature in a resistor which remains solid, like carbon, seems to be its vaporization temperature, like in the arc, although in practice the rapid oxidation, and deterioration due to other causes, its fragility at high temperature, etc., may determine a far lower limit. The limit of temperature in a liquid resistor is for most materials determined by the "pinch phenomenon" discovered by the writer some years ago,[5] which causes the column to part when a certain current is reached, after which the product freezes on open circuit.

In conclusion the writer desires to repeat that, as in most engineering designs, practical considerations will often determine some of the questions for the designer, or will limit him in his efforts to approach the theoretical ideals; that is no reason, however, why we should not try to find out what these ideal conditions are, and to keep them in mind so as to approach them as closely as practical consideration will permit, even though at times we may not be able to get very close to them. And unless we know what the theoretically best or limiting conditions are we may be attempting something which is impossible or commercially impracticable.

[5] "A Practical Limitation of Resistance Furnaces; The 'Pinch' Phenomenon." Trans. Am. Electrochem. Soc. Vol. 11, 1907, p. 329. "The Working Limit in Electrical Furnaces Due to the Pinch Phenomenon." ibid. Vol. 15, 1909, p. 255.

Two-stage crushing is being adopted in South Africa, using heavy stamps of 1800 to 2000 pounds for the first stage and tube mills for the second. The stamps crush to 4 or 8-mesh, and have a capacity of about 20 tons per day. The tube mill grinds this product to the degree of fineness considered necessary to free the gold.

Gas Power for Canadian Steel Plant.

During the past few months Allis-Chalmers Company has shipped four gas-engine generating sets and two gas-engine blowing units to the Lake Superior Iron & Steel Corporation at Sault Ste. Marie, Canada, and has two more of the blowing units nearing completion. Each one of these units required twelve cars for its transportation and its weight approximated 1,000,000 lb. Fig. 1 shows a train loaded with a single gas-engine blowing unit.

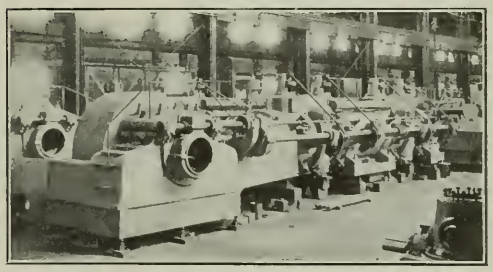


FIG. 2.—34-48 GAS ENGINE.

These gas-engine units are being installed in a large extension of the Lake Superior Iron & Steel Corporation's plant at Sault Ste. Marie, Ontario. The original rail mill will have a largely increased capacity and new plate and merchant mills will be built as well as coke ovens. For furnishing power for the mills and blowing the furnaces the company purchased the above units from Allis-Chalmers Company. They will be supplied with gas from the blast furnaces.

The gas engines (Fig. 2) are all alike, being of the twin-tandem, four-cycle type with cylinders 34 inches x 48 inches. They conform to the company's standard in all respects. Four of these engines are direct-connected to 1765-kva, 25-cycle, three-phase, 2300-volt alternators running at 107 r.p.m. These sets will supply power for driving the motors about the mill.

The other four units are to be connected to the new Slick blowing tubs (Fig. 3) manufactured by Allis-Chalmers Company. These tubs are 64 inches x 48 inches and are arranged to operate duplex on the opposite side of the main shaft from the engine. Each unit has a capacity of 25,000 cu. ft. per minute at 72 r.p.m., but can be speeded up to 85 r.p.m.

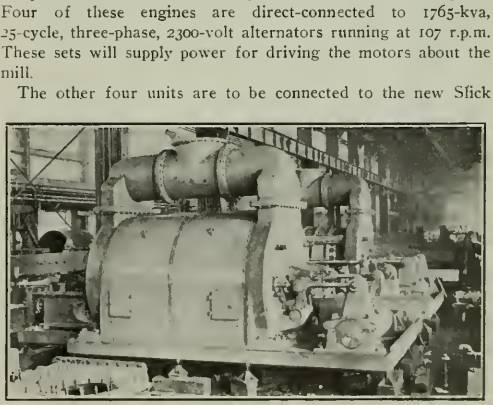


FIG. 3—SLICK BLOWING TUBS.

When the new works are completed this will be the largest steel-producing plant in Canada, and naturally the most modern. Mr. Alfred Ernst has been consulting engineer on the work and much of its success will be due to his efforts.

The Bureau of Mines.

The act establishing a Bureau of Mines in the Department of the Interior, approved May 16, 1910, became effective July 1. As originally approved, the law contemplated the transfer of the entire Technologic Branch of the United States Geological Survey, the mine accident investigations, fuel investigations, structural materials investigations, the entire personnel, property and equipment to the Bureau of Mines, but the sundry civil appropriation act, approved June 25, amended the law to such an extent that the structural materials investigations, including the personnel and equipment for these investigations, went to the Bureau of Standards, Department of Commerce and Labor.

Carrying out the spirit and intent of the law so amended, the Secretary of the Interior has transferred to the Bureau of Mines the investigation of mine accidents and fuels, together with the personnel and equipment of these investigations, and has transferred to the Bureau of Standards the structural materials investigations and the employees of the Technologic Branch of the Survey engaged in these investigations. The fully equipped testing station at Pittsburgh also goes to the Bureau of Mines.

The Bureau of Mines, therefore, includes the mine accidents and fuel investigations, for which an appropriation of $410,000 was made by Congress. The total appropriations for the bureau, including salaries, rent and expenses of removal amount to $502,200.

The work of the Bureau of Mines for the first year will be a continuation and expansion of the work carried on by the Technologic Branch of the Geological Survey. The law in itself provides for a variety of other problems that properly belong to the Bureau of Mines and which should eventually be undertaken, such as methods of mining and metallurgical processes, but these activities will be deferred for the most part until Congress gives additional authorization in the shape of adequate appropriations. The spirit of the debates in Congress, both on the Bureau of Mines legislation and on the appropriation items, emphasized the desire to regard the mine accidents investigations as urgent and this will be the feature of the work.

In all, $310,000 was appropriated for mine accident investigations. Of this sum under the general plans approved by the Secretary of the Interior, $120,000 is to be spent on the rescue stations, $36,000 for housing nine stations, $34,000 for equipping eight new stations, and $10,000 for additional equipment for five existing stations. The allotment for the investigation of explosives is $40,000, for electricity in mining, $14,000; appliances for preventing mine accidents, $8,000; examination and codification of mining laws, $5,000, and other technologic investigations, ore treatment, etc., $10,000.

For the analyzing and testing of the coals, lignites, ores and other mineral fuel substances belonging to or for the use of the United States, $100,000 was appropriated. Of this amount, $35,000 will be spent in the chemical and physical investigation of fuels, $25,000 in the inspection of Government fuel purchase, $22,000 in fuel efficiency investigations; $5,000 in lignite and peat investigations, and $4,000 in briquetting investigations.

The mine accident investigations, which have been transferred from the Geological Survey to the Bureau of Mines, were first authorized in the legislative appropriation act of May 22, 1908, carrying for this purpose an appropriation of $150,000. This was followed by a similar appropriation carried in the act for the sundry civil expenses of the Government for 1910. A mine experiment station was established in Pittsburgh during 1908, at which, since that time, investigations of explosives, coal gas, dust, electricity and other possible causes of mine explosions have been continually under way. The mining engineering field forces of the Geological Survey has already made decided progress in the study of underground mining conditions and methods. Practically all of the coal mines in which mine explosions have occurred during the last two years have been carefully examined, the gases, coke and dust have been analyzed at the laboratory at Pittsburgh and every effort has been made to determine the explosibility of various mixtures of gas and air in the presence of shots of different types of explosives.

Considerable progress has also been made in the investigation of explosives used in coal mining, and the conditions under which they may be used with least risk. Manufacturers have submitted many explosives for test at the station and a considerable portion of them passed and have been classified among the permissible explosives. The investigations and educational work in connection with the use of artificial breathing and other types of mine rescue equipment, the so-called oxygen helmets, have not only been useful in developing a more satisfactory use of such equipment in the examination of mine explosions, but also better methods for using this equipment in mine rescue work.

The fuel investigations under the Geological Survey, and which are transferred to the Bureau of Mines, have already resulted in a better realization throughout the country as to the value of fuels. One result of this work is that nearly all of the fuel now purchased by the Federal Government is bought on specifications and subject to test by the fuel division, or purchased after examination made of the coal supplied by the mines from which coal is delivered to the Government.

The publications of the Survey relating to mine and fuel investigations, those prepared by the Technologic Branch, will in the future be distributed by the Bureau of Mines. The publications relating to structural materials will continue to be distributed by the Geological Survey. The last of the bulletins of the Technologic Branch to be published by the Survey will be issued from the Government Printing Office about Aug. 1. This bulletin relates to the explosibility of coal dust and was prepared by Mr. G. S. Rice, with chapters by Messrs. J. C. Frazer, Axel Larsen, Frank Haas and Carl Scholz.

The first of the Bureau of Mines bulletins, "The Voltatile Matter of Coal," by Messrs. H. C. Porter and F. K. Ovitz, will be published in the next few months. Then will follow "Coal Analyses," by Messrs. N. W. Lord and J. S. Burrows; "Final Data Regarding Steam Tests," by Mr. L. P. Breckenridge; "North Dakota Lignite as a Boiler Fuel," by Messrs. D. T. Randall and Henry Kreisinger; "Producer-Gas Tests in 1905-1907," by Mr. R. H. Fernald; "The Coke Industry as Related to the Foundry," by Mr. Richard Moldenke; "Coals for Illuminating Gas," by Messrs. A. H. White and Perry Barker; and "Petroleum for Combustion Under Steam Boilers," by Mr. I. C. Allen.

These publications, when issued, can be obtained by addressing the Director of the Bureau of Mines, Washington, D. C.

Effects of Ultraviolet Rays on Gases.

M. M. Berthelot and Gaudechon have investigated the effect of ultraviolet rays from a quartz mercury vapor lamp on acetylene, cyanogen, ethylene and oxygen, and in every case found a decrease in volume consequent on polymerization, but did not observe any other chemical change. Acetylene is transformed into a yellow solid, and the change takes place more readily under reduced pressure or when an inert gas is admixed; ethylene yields a liquid polymer having an odor resembling that of rancid fat; cyanogen becomes paracyanogen, and oxygen is to a limited extent converted into ozone.

The Fifth International Congress for Mining, Metallurgy, Applied Mechanics and Practical Geology.

By Jos. W. Richards

Düsseldorf, the vigorous young city which "watches on the Rhine" as the representative city of the great Westphalia coal and iron industry, was selected by the fourth congress, at Liège, for the congress of 1910. How the Westphalians, technically the most progressive people in Germany, rose to the occasion and surpassed all expectations, we will try to tell.

The congress was a model of careful forethought, system and minute attention to detail, to a degree which astonished even the Germans themselves from other provinces. And besides time and thought, money was provided in abundance for every purpose; it was said that the city of Düsseldorf itself contributed $50,000 to entertain the visitors. Fifteen hundred members were in attendance, and yet the extraordinary ability of the organizers and conductors of the congress assured everyone a comfortable and profitable time.

HISTORY OF THESE CONGRESSES.

The first International Congress of Mining and Metallurgy was organized by the Société de l'Industrie Minerale of St. Etienne, France, in connection with the International Exposition of Paris in 1878. Only about 150 persons took part in this congress, most of them from France.

The second congress was invited by the French Government to meet in Paris, in connection with the exposition of 1889. Over 600 accepted the invitation, among them many from outside France.

The third congress, called together in connection with the Paris exposition of 1900, was attended by representatives from almost all civilized countries.

The fourth congress was held at Liège, in 1905, in connection with the International Exhibition. Here it was determined to unite in one congress the fourth Congress for Mining and Metallurgy with the third Congress for Applied Mechanics (the two previous ones had been held in Paris in 1889 and 1900) and the first Congress for Practical Geology. It was thus that the congress attained its present scope.

The fifth (and present) congress was held in Düsseldorf on June 20 to 25, under the auspices of the *Verein für die bergbaulichen Interessen im Oberbergamtsbezirk Dortmund* and the *Verein deutscher Eisenhüttenleute*. At their invitation 1910 members enrolled, of whom 654 were from outside Germany, and about 1500 were in actual attendance. The membership of the four sections were: Mining, 1178; Metallurgy, 1177; Applied Mechanics, 972; Practical Geology, 810. This congress was the first to meet unconnected with an international exhibition, but at the close many of those present participated in a three-day excursion to the Brussels exposition.

The sixth congress will be held in London in 1915.

SOCIAL FEATURES OF THE DUSSELDORF CONGRESS.

On the evening of June 19 an informal gathering of members and friends was held in the Tonhalle; the ladies were given a separate reception in the Art Gallery.

On the evening of June 20 the city of Düsseldorf gave a reception to the congress in the Tonhalle. This was accompanied by music and refreshments served in the fine gardens, and was a very enjoyable function. The afternoon of this day the ladies were shown the sights of the city.

On the 21st the ladies were taken on an excursion to Zons in the morning and in the evening a banquet was given by the city in the two great halls of the Tonhalle, at which over 1000 were present. At the close a number of fine after-dinner speeches were made in German, French and English, and the festivities closed with a wonderfully clever bit of theatrical staging wherein was shown the interior of the International Steel Works, with Herr Gärtner, of the Düsseldorf Stadttheater, as Obermeister, and Frau Paschke as the Genius of Technique. The pouring of the steel and casting of the ingots was finely done, and the conversations of the workmen and declamations of the chief actors were very well performed.

On the 22d the ladies were taken on an excursion into the hill country around Solingen in the morning, while the evening was given over to a steamer trip on the Rhine, followed by illumination of the banks and a splendid exhibition of fireworks. On this day Count Zeppelin traveled from Switzerland to Düsseldorf in his great airship, making the 450 miles in nine hours. Opportunity was given to see the splendid balloon, and in the evening Count Zeppelin accompanied the members on the steamer, where he was most enthusiastically received and conversed with many of the "congressistes."

FIG. 1.—OFFICE BUILDING OF THE SOCIETY OF GERMAN IRONMASTERS.

On the 23d the ladies were shown the sights of Essen in the morning, and in the evening, after the closing sitting of the congress, there was given a splendid "Gartenfest" in the City Hall of Essen, with dancing and illumination of the park.

The 24th, 25th and 26th were devoted to the excursion to Brussels, made from Düsseldorf in two special trains, and which included a reception in the German Exhibition Building by the German commissioner, on the 24th; a banquet, followed by fireworks, given by the exposition commissioners, on the 25th, and a reception by the city of Brussels, in the magnificent Town Hall, on the 26th.

Fig. 1 shows the new office building of the Society of German Ironmasters (Verein Deutscher Eisenhüttenleute); Fig. 2 the office building of the German Steel Trust in Düsseldorf; Fig. 3 the crucible steel department of the Krupp Steel Works in Essen.

VISITS AND EXCURSIONS.

A finely printed book of 100 pages, printed in both German, French and English, was provided for describing the 40 different excursions and visits planned. Of these, 35 were visits to industrial plants, and five were geological excursions. Of the former, there were open to visiting 1 river improvement

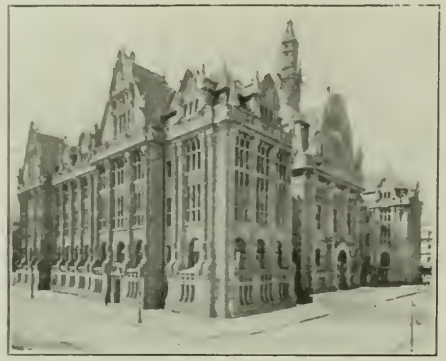


FIG. 2.—OFFICE BUILDING OF GERMAN STEEL TRUST.

plan, 2 harbor and dock-yards, 1 electric power station, 1 technical high school (Aachen), 8 machine works, 20 collieries, 12 coking plants (all by-product), and 14 iron and steel works. Each excursion was accompanied by a guide and was run on a time schedule. The members taking the excursions were not even asked to pay for the railway tickets, all being provided for.

SCIENTIFIC WORK OF THE CONGRESS.

In the four sections there were altogether 120 communications, 35 on mining, 44 on metallurgy, 23 practical metallurgy, 21 theoretical, 17 on applied mechanics and 24 on practical geology. In the writer's opinion, the congress would gain in interest and usefulness if its field were not so broad; if, for instance, the section "Practical Geology" were turned over to the International Geological Congress, which meets every four years, this year in Stockholm; the section on "Applied Mechanics" should form the nucleus of an international mechanical congress; and the sections which remain—Mining and Metallurgy—representing the original Congresses I, II and III, would be quite sufficient to constitute an international congress of themselves. The dragging in of the other sections divides interest in the congress and makes it unwieldy in size.

Geological Papers.

Of these papers, the only ones which would deeply interest metallurgists and chemists are

The Iron Supplies of the World, by Dr Beyschlag, of Berlin. This was not in print. Metallurgists interested are referred to the great compilation on this subject which is now being printed in two volumes in Stockholm by the International Geological Congress.

The Utilization of Water Powers, by Holz of Aachen. This was also not in print in time for distribution at the congress.

The Mineral Resources of Canada, by H. Mortimer-Lamb, of Montreal. A condensation of the reports on mineral resources, such as are issued by the Canadian Government, showing an increase in mineral production from $10,000,000 in 1886 to $90,000,000 in 1909; the increase *per capita* product being from $2.23 in 1886 to $12 in 1909. The water-powers of Canada are given as follows:

	Possible horsepower.	Developed horsepower.
Nova Scotia	54,300	13,300
New Brunswick	150,000	....
Quebec	10,000,000	700,000
Ontario	6,190,000	350,000
Manitoba	495,000	18,000
Northwest Territories	600,000	....
Saskatchewan	500,000	....
Alberta	1,144,000	1,330
British Columbia	2,065,500	73,100
Total	21,198,800	1,155,760 (= 5.5%)

The Canadian Government pays the following bonuses on pig iron and steel manufactured electrically in Canada:

	Pig iron.	Steel.
During 1910	$2.10 per ton	$1.65 per ton
During 1911	1.70 per ton	1.05 per ton
During 1912	0.90 per ton	0.65 per ton

It is interesting to note that if the steel is made electrically from electrically produced Canadian pig iron, both bounties can be collected.

Mining Papers.

Of the 35 papers on mining the most important were probably those of Engineer Zaeringer on the latest developments in sinking shafts by the freezing method, and Bergassessor Viebig's excellent treatise—one might almost call it—on the use of reinforced concrete in mine construction. For us, the following only was of metallurgical interest:

The Use of Low-Grade Fuels, by Bergassessor Dobbelstein, of Essen, considered the question of burning coal of the following kinds:

1. Coarse coal, with up to 50 per cent ash and up to 20 per cent water,
   (a) with high volatile matter,
   (b) with small volatile matter.
2. Fine coal, with up to about 40 per cent ash and up to 30 per cent water.

FIG. 3.—CRUCIBLE STEEL DEPARTMENT OF KRUPP STEEL WORKS, IN ESSEN.

(a) with high volatile matter,
(b) with small volatile matter.

3. Coke breeze (fine coke), with up to 30 per cent ash and up to 20 per cent water.

The Essen Boiler Inspection Society with the Dortmund Mining Society contributed a fund to investigate the possible use of these materials. The projected gas-producer tests have not yet been made, but the boiler tests have been completed with valuable results. A simple rule for comparing the values of different fuels was arrived at, viz., the values in pfennigs (1 pfennig = 0.25 cent) per ton equals the calorific power divided by the per cent of ash plus half the per cent of water in the coal. The quotients express with satisfactory accuracy the relative industrial values of the respective fuels.

If briquetted, they should be pressed into small cylinders 2 in. in diameter by 2 in. long, which quickly become ignited and stay together, while larger briquettes warm up so slowly that the pitch escapes and they fall to pieces. Tar-pitch residues should be used, which can be bought in Germany for about 9 marks ($2.25) per ton. Coke breeze was likewise burned on a perforated grate with blast and by the Praesto firing.

The Praesto firing showed up cheapest and most effective, giving 1 ton of steam for 0.98 mark (24.5 cents), and evaporating nearly 23 kg of water per hour per square meter (4½ lb. per square foot) of heating surface. Experiments with the poor, coarse fuels on ordinary grates showed that in many cases it was more advantageous, regarding cost per ton of steam produced, to use the poor fuel than ordinary good fuel.

Metallurgical Papers.

Electricity in Metallurgical Work was discussed by K. MALEYKA, of Berlin, in a very general manner. He gave examples of the different systems of electrical energy generation and distribution and the different types of motors suitable for various metallurgical operations, and gave particulars of the energy requirements for separate purposes in blast-furnace, steel and rolling-mill service. He concludes that it is usually necessary, in the designing of electric driving machinery, to consider each case by itself.

The Present Status of Ore Dressing, by Bergassessor P. BODENSTEIN, of Kalk, near Cologne, starts out by saying that hand-sorting will probably remain always the first necessary treatment, so as to remove coarse ore and gangue. High-speed rolls, wet ball-mills and wet pendulum grinders are all commended, while wet edge-mills and stamp mills, although releasing the ore well, are expensive to install and to work because of their small output.

Electrostatic separation is only effective on completely dry, fine-grained and at the same time dustless ore; its application is limited, and it is used only where other methods are ineffective. Spathic iron ore is being successfully roasted in revolving tube furnaces, and more recently it is being treated in "roasting converters" (Bessemer roasting, or blast roasting, or blast sintering, however it may be called), constructed by the Humboldt Machinery Company.

Wire-Rope and Electric Suspended Railways are described by Oberingenieur G. VON HANFFSTENGEL, of Leipzig. He particularly recommends for use in metallurgical works the "electric rope-way system," which is a combination of the wire-rope railway with the electric suspended railway on the Bleichert system. This combination possesses the advantages of the rope railway in surmounting steep grades with the adaptability of the electric service for turning corners, and is recommended for charging cupolas and blast furnaces. Since elevations of any height can be surmounted, the system can be used for piling material into storage heaps, which it performs at a cost of about 2 cents per ton.

Wire-Rope Railways for Heavy Service were discussed by Direktor ELLINGEN, of Cologne. Lines up to 37 km (22 miles) length are in operation; the four-wheel bogie has mostly displaced the two-wheel for heavy transport. The firm of Pohlig has built lines which carry 250 loads per hour, of 2 tons each, or a total of 500 tons per hour, at a cost of 1.5 to 2 pfennigs (0.4 to 0.5 cent) per ton-kilometer (0.6 to 0.8 cent per ton-mile). The author believes that by using four-wheel supports the cost may be brought down to half the above-given maximum, i.e., to 1 pfennig per kilometer.

Electrical Lifting Appliances in Metal Works were extensively described by Engineer F. HEYM, of Wetter (Ruhr). He particularly dwelt on the increased speed and increased output made possible by the use of fast-moving electrical machinery.

The Changes in Fire Bricks During Use, by Dr. BLASBERG, of Dahlhausen (Ruhr), deals with the changes due to heat, the action of slags, the action of gases, deposition of carbon, ash of the coal, etc. The author made numerous analyses of the slagged or glazed part of the bricks, and compared them with analyses of the unaltered kernel of the brick.

The work covered a large range and gives much valuable information. Of the 86 complete analyses recorded in the paper, we cite the following representative ones, and repeat some of the author's conclusions:

	SiO^2.	Al^2O^3.	Fe^2O^3.	CaO.	MgO.	$(K_1Na)^2O$.	
Cupola brick:							
Kernel..	89.66	7.79	0.69	0.33	0.17	1.20	(MnO.)
Slagged.	72.59	7.24	11.24	0.28	1.47	4.61	2.57
Blast-furnace brick:							
Kernel..	59.79	28.50	3.45	0.56	0.36	10.34	...
Slagged.	53.50	27.17	5.34	0.88	0.52	12.59	...
Copper-furnace brick:							
Kernel..	64.13	29.61	1.89	0.80	0.72	2.85	(CuO.)
Outside.	63.95	29.67	1.93	1.10	0.75	2.50	0.10
Slagged.	52.10	19.81	1.39	0.52	0.47	3.90	21.66
Blast-furnace brick:							
Kernel..	69.54	25.34	1.20	0.50	0.54	2.68	...
Crust...	54.18	25.45	1.25	0.90	0.57	17.65	...
Coke-oven brick:							
Kernel..	84.99	11.28	1.02	0.34	0.31	2.06	...
Slagged.	82.14	12.37	0.99	0.39	0.37	3.74	...
Siemens-Martin silica brick:							
Kernel..	94.67	1.84	1.00	1.67	0.46	0.36	...
Slagged.	90.66	2.94	2.50	2.87	0.60	0.43	...

An inspection of these typical analyses shows in almost every case an increase in the alkalies, coming mostly from the ash of the fuel used; in coke ovens, however, common salt in the wash water used for washing the coal may be an active agent in slagging the bricks, the chlorine going into the gases as ammonium chloride. Lime and magnesia increase somewhat in blast-furnace bricks, but more in open-hearth bricks, where they are subjected to the dust of the lime and burned dolomite shoveled in. The iron oxide increases somewhat in open-hearth furnace bricks, more in blast-furnace bricks, nearly as much in some bricks at the bottom of coke-oven retorts, but most in the cupola lining, where the oxidizing influences are stronger.

The silica decreases in every case, which phenomenon Dr. Blasberg surmises may be due to volatilization; the writer quite agrees with this explanation, and would add that the removal of silica by volatilization decreases the ratio of silica remaining to the other materials present, and, therefore, increases the fusibility. This change takes place even at moderate temperatures far below those prevailing in the Siemens furnace. Dinas bricks sometimes become dark gray; an analysis of such a slagged and discolored brick showed 0.65 per cent of carbon, probably deposited from hydrocarbon gases.

The Latest Development of Coke Ovens and Coking Practice was handled very extensively by Prof. FR. HERBST, of Aachen, in a communication covering 35 large quarto pages. In general we note that, in Germany, the coke-oven gas engine is winning its way against the steam turbine for power purposes; further that the producing of lighting and heating gases for cities is becoming more and more of a question of putting up coking plants which are adapted to giving the largest possible quantity of coking gas, thus giving great impetus to the regenerative oven and to the heating of the same by producer gas, usually from a bank of centrally placed producers.

The latest ovens are mostly of the type with vertically placed flues between the ovens, usually worked in four groups with central chimney or regenerator flue. The inclined ovens are also greatly in favor, with the retort at about 45°, since they allow of easy automatic charging and discharge themselves when the lower door is opened. Oven doors with gas-tight planed and water-cooled surfaces are being preferred to those luted with clay, being tighter, saving the clay and the labor of sealing up the retorts.

The newest Otto retort has gas supplied by 10 large bunsen burners, each supplying two vertical flues. The new Collin retort has a waste-gas flue between each pair of vertical heating flues. Brunck lays stress on distributing the gas to various places along the length of the retort, and has devised a new recuperator with shelf-formed heating surfaces and horizontal flues.

The oven of Kros, built by Salau & Birkholz, in Essen, has each vertical heating flue supplied with its own gas nozzle, with a horizontal flue along the whole length of the floor of the retort and thence communicating with the regenerators still further down beneath the retorts. The reversing of the draft is done entirely beneath the retorts. Each side flue is connected with two regenerative chambers, so that, as the retorts are charged alternately, the flues are always properly heated. Five of these ovens were run very successfully at Waltenscheid, reducing the coking period from 44 to 28 hours.

The Rheinischen Chamotte & Dinas-Werke, of Cologne, has likewise attempted the more uniform heating of the retorts by using burners of a particular construction entering both from beneath and from above, and arranged so as to give the most intimate possible mixture of gas and air. The variations in temperature on reversal are also reduced to a minimum. With these improvements, retorts up to 2.5 m (8 ft.) in height are in successful use.

The firm of E. Coppée, of Brussels, has also improved its regenerative oven, dividing the vertical heating flues into 10 groups of three flues each, and reversing the flow for regenerative purposes in each alternate group.

For city gas supply purposes the inclined retort with independent producer-gas supply for heating is making great advances. The gas supplied must be of at least 5000 calories per cubic meter, and such plants are, therefore, run so as to furnish the maximum of oven gas. If the coal gives gas averaging over 5000 (at Innsbruck 1 ton of coal gives 328 cu. m of gas of 5887 calories heating value), it is possible to increase the quantity of gas by sending in steam just before emptying the retort, in which manner as much as 400 cu. m of illuminating gas of over 5000 calories value has been obtained from 1 ton of coal. In such cases, with a production of 70 per cent of coke, about 25 per cent of the coke is used in gas producers to furnish the heating gas. Fig. 4 shows the newer ovens of this type as erected for the city of Munich by the Stettiner Chamottefabrik, on the Ries pattern. Each retort holds 6 tons of coal; the whole plant supplies 300,000 cu. m (10,000,000 cu. ft.) of gas daily. No trouble is met from retort carbon deposition in these furnaces. The lateral flues are also inclined.

The Vienna gas plant, now being built, is operated likewise by producer gas for heating the retorts. The producers use 13 per cent to 14 per cent of all the coke produced, and the evaporator under the producers, warmed by the hot gas, sends 0.8 kg of water into the producer for every kilogram of coke burned therein. The producer gas contains 21 per cent CO, 10.6 per cent H^2 and 10 per cent CO. The plant is to [illegible] horizontal ovens and partly inclined, its total capacity, in 1911, will be 1,000,000 cu. m (30,000,000 cu. ft.) daily.

In the new horizontal ovens, built after the Kopper's system, regenerators are provided for both heating gas and air, since these are of nearly equal volume. The arrangement of these beneath the retorts is very instructive (Figs. 5 and 6), two gas regenerators and two air regenerators being built together as shown, with considerable advantage.

The automatic machinery (Fig. 7) for bringing up the coal, stamping it into forms which have the shape of the oven, and charging it at once without manual labor have been greatly improved and are being rapidly adopted everywhere. These traveling loaders are highly developed, and described in detail in the article, as are also various minor details, such as the improved gas-tight doors, reversing valves, charge levelers, quenching devices, etc.

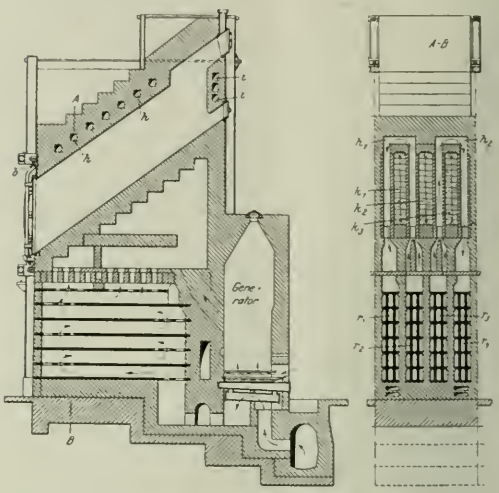


FIG. 4.—SECTIONS OF RIES COKE OVENS OF STETTINER CHAMOTTE FABRIK (*hi*, HEATING CHANNELS; k_1, k_2, k_3, RETORTS; r_1, r_2, r_3, r_4, REGENERATORS)

Progress in Recovery of Coke-Oven By-Products was treated by Prof. Dr. O. Rau, of Aachen, in a 28-page memoir containing a wealth of important information. Of the coal raised in the respective countries, the following percentages are coked:

	Germany.	Belgium.	England.	U. S. A.
In coking plants without saving of by-products	4	5	10	10
In coking plants with saving of by-products	16	11	2	2
In gas works with saving of by-products	4	4	6	1
Total percentage coked	24	20	18	13
Coked with saving of by-products	20	15	8	3
Percentage of coke made with saving of by-products	83	75	44	23

The world's production of by-product ammonium sulphate has risen to 500,000 tons per year, and of benzol to 90,000 tons. Investigations have shown that tar is formed between 200° C and 500° C., and ammonia between 500° C. and 700° C.; by very slow distillation between the latter temperatures the recovery of ammonia may be increased by one-half. The tar is finding increased use in street paving and benzol in motor cars.

The waste gases are being used on a large scale for light, heat and power, particularly in large cities; coke-oven gas is piped 63 km (38 miles) to Barmen and sold there at 2.5 pfennig per cubic meter (18 cents per 1000 cu. ft.). The paper contains a lengthy discussion of the various reactions in the ovens at different temperatures, and closes with a complete bibliography of recent information on the subject.

Gas-Producers, by Ober-Ingenieur J. Hofmann, of Witkowitz, treated of the oldest and the latest forms of generators. In 1841 Ebelman proposed his shaft generator, with tuyeres, addition of iron slags as flux, and tapping out of the slag. Since 1850 there have been in constant use in Witkowitz producers without a grate, using lime as flux for the ash of the fuel, and tapping out liquid slag every one to two hours. Meanwhile, all the various types of grate and revolving-bottom

producers have been developed. Numerous analyses of coals and gas used and made in various producers in Europe are given.

Quite lately, however, a return has been made to the slagging shaft-furnace producer. (Fig. 8.) The firm of Sepulchre, Fichet & Heurty, of Paris, build and operate the S. F. H. generator, which is built like a small charcoal blast furnace, burns coke with addition of lime, sand and granulated blast-furnace slag for flux, and uses three to six water-cooled tuyeres, with blast at 0.5 m to 1.5 m water-column pressure. No steam is blown in.

When the coke ash contains iron or iron is in the fluxes, pig iron runs out with the slag tapped every one to two hours. Tests in Girancourt, near Nancy, with poor fuel, gave the following results:

Brown Coal.		Gas (Dry).		Slag.		Pig Iron.	
C	44.41	CO	28.3	SiO^2	42.17	C	2.30
N	0.45	H^2	7.7	AlO^3	18.83	Si	1.73
H	1.36	CH^4	7.5	CaO	20.57	P	0.23
S	6.30	CO^2	1.4	MgO	12.89	S	1.97
Ash	37.66	N^2	55.1	MnO	0.17		
H^2O	10.00			S	1.72		
				P^2O^5	0.04		
				FeO	1.26		

The brickwork of the crucible needs renewing every three months. The gas-producing power is high. A furnace only 1.2 m in diameter at the tuyeres and 1.13 sq. m area gasified 24.4 tons of fuel per 24 hours, or 900 kg per hour per square meter of section at the tuyeres (175 lb. per square foot). In actual amount gasified this is equal to the output of the ordinary 3-m (10-ft.) diameter producer.

Either in the slagging or the non-slagging producer it is recommended with caking coals to charge unburned limestone as a flux; this keeps the coal from caking too hard, the CO^2 driven off is reduced to CO and increases the value of the gas, while the lime forms a better slag or ash.

The author dwells on the injurious influence of water vapor in producer gas, saying it is impossible to produce best quality steel in the open-hearth furnace if the producer gas contains over 30 grams of moisture per cubic meter. The best way to achieve this is to superheat the steam used before sending it into the producer; this is *very strongly* recommended. It can be easily done by passing it through a coil heated by the producer gas itself. By doing this the producer gasifies more coal and gives better gas.

The making of producer gas has come to a turning point. It is now possible to utilize very poor fuel, such as is often thrown on the waste heap at the mines, to produce from it good gas and to completely burn it to ash or slag. In June, 1910, the Compagnie des Mines Blancy, at Montceau, will begin to gasify culm-heap residue containing only 20 per cent carbon, 15 per cent volatile matter and 65 per cent ash. Coke breeze is mixed with this material, and thus utilized. The complete utilization of our coal supplies is only possible by the manufacture and use of producer gas, which is, therefore, of highest economic importance.

The Production of Tar-Free Producer Gas was discussed by Dr. H. BRAUNE and EDV. HUBENDICK, of Stockholm. The

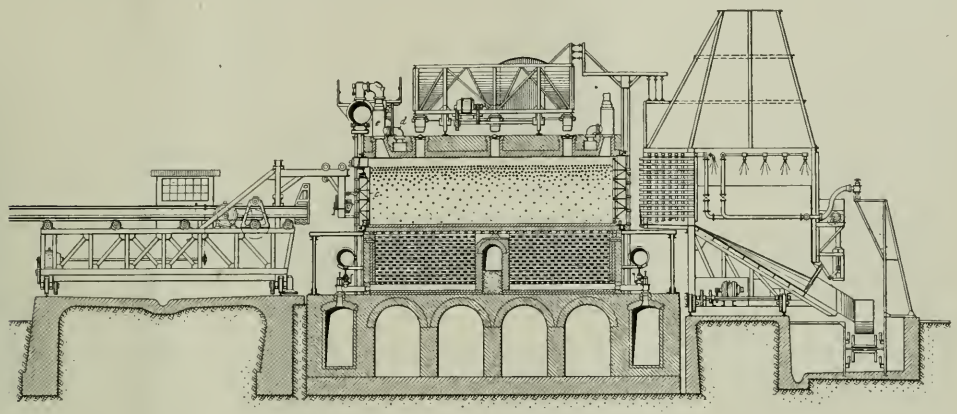


FIG. 5.—LONGITUDINAL SECTION OF KOPPER'S COKE OVENS.

authors start from the classification of hydrocarbons into aliphatic and aromatic compounds, the former further into paraffins (C_nH_{2n+2}), olefins (C_nH_{2n}) and acetylenes (C_nH_{2n-2}) all built upon the compound CH_4, and the latter built upon the closed benzol ring (C_6H_6). The paraffins are most resistant to heat, being able to withstand high temperature, such as 400° C. to 500° C. for hours.

Wood is a mixture of aliphatic and aromatic substances, the most important being the alipathic material cellulose. On dry distillation, on slow heating many condensable products result because of the oxidizing action of the water of constitution upon the hydrocarbons; on slower heating more gaseous products result.

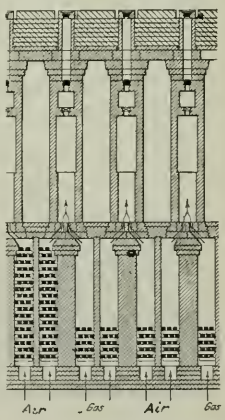


FIG. 6.—VERTICAL SECTION OF KOPPER'S COKE OVEN.

Peat contains very little aromatic compounds, which is established by the nature of the dry distillation products.

Lignite, or *brown coal*, consists principally of alipathic compounds, yet there are found with the peat deposits aromatic compounds coming from the decomposition of resin. Such deposits are seldom found, but when found are of considerable value.

Bituminous coal consisted at one time principally of alipathic compounds, but by dry distillation gives the whole aromatic series; older coals contain more of the aromatic compounds.

In the gas producer aromatic compounds are weak and are easily decomposed at 750° C., the aliphatic compounds are not so easily broken up. It is, therefore, important, in making gas from coal which gives aromatic compounds, to keep the temperature of the issuing gases between 750° and 780°, and to

work with a large excess of hydrogen in the producer gas made by using steam. This prevents the decomposition into sooty carbon and hydrogen, and yet breaks up easily condensable aromatic compounds.

The aliphatic compounds from brown coal, wood, etc., are not to be thus decomposed, but require a temperature of over 1000° and oxidizing conditions. Aromatic distillation products should, therefore, pass through the fuel in the upper part of the producer, in presence of much hydrogen in the gas, at temperatures not over 750° to decompose them in a reducing atmosphere and so free the gas from tar; aliphatic distillation products are best drawn off and passed through the bottom of the producer with the air for combustion, so as to decompose them at a high temperature in an oxidizing atmosphere.

The Status of Iron-Ore Briquetting and Agglomeration in Germany was considered by Geheinier Bergrat G. Franke, of Berlin. Under iron ore were grouped also flue dust, pyrites, cinders, converter dust, roll scale and similar ferruginous wastes. There are at present nine briquetting plants, daily output 2200 tons and one agglomerating plant of 130 tons capacity. Together, they treat 500,000 tons of flue dust per year, averaging 35 per cent iron, which is about one-third of all which is produced by the German blast furnaces. Six new plants are under construction, and the time is not far off when all the flue dust will be briquetted.

The magnesium chloride process uses 1½ per cent of a waste 33 per cent $MgCl^2$ solution from the potassium chloride works, costing 26 pfennig (6½ cents) per ton of ore; the plant is cheap, 60,000 marks ($15,000) will build a plant treating 60 tons in 10 hours; the solution is mixed with the ore, pressed at 400 atmospheres, and let stand 24 hours. The total cost is 1.60 marks ($0.40) per ton of briquettes.

The lime process uses 6 per cent to 8 per cent of slaked lime, ground up with 1 per cent of blast-furnace slag; the moist mixture is pressed under 200 to 400 atmospheres, but the briquettes must stand several weeks in the air in order, by slow absorption of CO^2, to become sufficiently hard and strong, making the cost of this method high.

The Ronay process uses no binder, but presses the moist material at 1000 atmospheres and then bakes the briquettes at

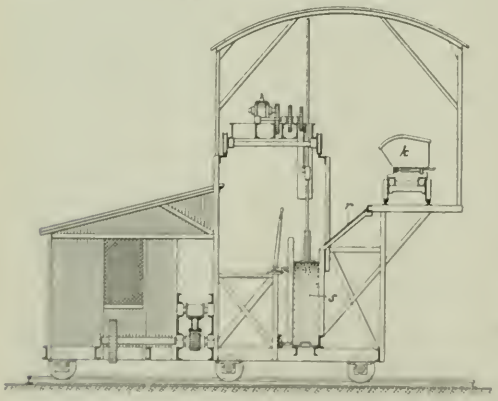


Fig. 7.—Automatic machine (*k*, tilting car, *r*, inclined platform; *s*, stamping former).

low redness in a furnace heated by blast-furnace waste gas. It requires a comparatively expensive plant and costs much for power, still some new plants on this line are projected.

Only one plant in Germany sinters in revolving cylinders, firing with coal dust. The Giessener Braunsteinwerken vorm. Fernie, at Giessen, sinters Fernie manganese ore containing 36.5 per cent Mn and 22.5 per cent Fe, mixed with spathic ore containing some pyrites and zinc blende. The furnace plant is very dear, and the costs of treatment run up to 3 marks to 4 marks ($0.75 to $1) per ton of product.

The briquetting of turnings, filings and planings from iron and steel works by the Ronay method is very successful. Up to 2000 atmospheres pressure is gradually applied, and the briquettes thus made melt down with only 4 per cent to 10 per cent loss. They are particularly suitable for open-hearth and

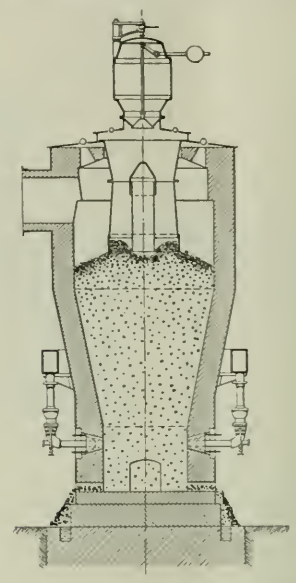


Fig. 8.—Slagging shaft-furnace gas-producer.

electric steel manufacture, and are coming rapidly into general use.

Turbo-Compressors for Blast Furnaces were discussed by Prof. Dr. Ing. Rateau, of Paris. The first multiple-wheel, high-pressure turbo-compressor was started at Bethune in 1905. Turbo-blowers have been furnished to the blast furnaces of Vizcaya in Spain and at Chiers; very high-pressure blowers were constructed for the Rand Mines in the Transvaal, which have attained 65 per cent efficiency on isothermic compression. Two centrifugal compressors are being built for the Bessemer converters of Denain and Auzin. Over 80 of these high-pressure compressors are now in operation.

Hydraulic Air Compressors were described by Oberingenieur P. Bernstein, of Cologne. They produce a high degree of air compression with a small height of fall. Comparison with mechanical compressors shows much greater economy for the hydraulic. Besides using only water-power, the hydraulic compressor is very simple, reliable and always ready to work. The air supplied is free from dust and oil, and because of its coolness is low in moisture.

Reduction by Carbon and Carbon Monoxide was the subject of a short communication by G. Charpy, of Montluçon. During several hours heating at 1000°, in a vacuum, neither sugar carbon, graphite nor diamond carburized iron; the least trace of gas caused immediate cementation. CO gas carburizes iron from 600° up to 800° C.; it oxidizes chromium and manganese at 1000° and over, forming a deposit of carbon. Solid carbon does not reduce iron oxide, in a vacuum, until 950° is reached. CO gas commences to reduce Fe^2O^3 to metallic iron at 280°, but very slowly. NiO is similarly reduced.

Gruner's Ideal Working of the Blast Furnace, by Prof. Dr. Jos. W. Richards, of Lehigh University. This paper has

been printed in full on page 403 of the July issue of this journal.

The Advantages of Hot Blast and Dried Blast in Blast Furnaces was discussed by Geh. Prof. Dr. F. Wüst, of Aachen. An interesting table is given concerning the changes produced by heating the blast, gathered from the literature of the blast furnace. The figures are quite curious, varying from 0 per cent fuel saving and 40 per cent faster running to 0 per cent faster running and 20 per cent fuel saving. If we arbitrarily add the two percentages together, there is obtained a minimum of 17 per cent advantage and a maximum of 98 per cent. Dr. Wüst concludes this part of his review by the statement that "although 80 years have gone by since hot blast was introduced, yet a satisfactory explanation of its action, free from all objections, has not yet been found."

The use of dried blast by Gayley led to the same extraordinary results; the saving was several times as great as could be calculated from the heat saved by removing the moisture. The results show from 11 per cent fuel saving and 21 per cent faster running, to 2 per cent faster running and 19 per cent fuel saving. Arbitrarily adding these percentages together, we have a minimum of 17 per cent advantage in September and November, and a maximum of 65 per cent in April. After reviewing all the proposed explanations which have been advanced (except that of J. W. Richards in *Trans.* Am. Inst. Mining Engineers, February, 1905, which was overlooked and not mentioned) Dr. Wüst again concludes that "all the explanations advanced have been insufficient and unconvincing."

He then offers his own explanation, as follows: When the blast meets carbon, there is first formed CO^2 and next CO. The velocity of these reactions depends on the temperature, the higher the temperature, caused by heating or drying the blast, the quicker these reactions occur, and the smaller the zone or area in which free oxygen or CO^2 exists at the tuyere level. As the reduced iron falls through this tuyere section, the part of it which drops through the space where CO^2 or O^2 exists is oxidized or refined, losing some of its silicon, manganese, fosforus and carbon, while some of the FeO dissolves in the slag.

These reactions require heat (according to Dr. Wüst's argument) and therefore the smaller this oxidizing zone the less heat is thus lost and the faster the furnace works, with less fuel.

The concluding part of Professor Wüst's explanation is neither very clear nor convincing. He concludes by predicting that blast enriched in oxygen (Lindeluft) would be another means for contracting the "oxidizing zone" and producing advantageous running in a way similar to hot blast and dried blast.

The Rationale of Dried Blast was further discussed by Prof. Dr. Jos. W. Richards, of Lehigh University. This paper was as follows:

The higher maximum temperature of the furnace gases in the smelting zone (region of the tuyeres) is the sole and the sufficient reason for the advantages produced by drying the blast.

Qualitative Considerations.—Removing the moisture of the blast results in saving the heat formerly required for its decomposition and enables the furnace gases to attain a higher maximum temperature of combustion. The higher maximum temperature of the furnace gases quickens the rate of transfer of the necessary smelting heat to the slag and pig iron, and thus increases the rate of smelting.

The smelting down of slag and pig iron to the liquid and superheated state requires to have added to those materials, at a temperature at and above their melting points, their latent heats of fusion and the sensible heat of superheated liquid material. This is the principal thermal work done in the smelting zone of the furnace. It can only be accomplished by the agency of furnace gases possessing maximum temperatures far above the melting points of these materials, and is accomplished only by the sensible heat which these gases are able to give out in cooling down from their maximum temperature to the melting points of the slag and iron.

The sensible heat which the furnace gases possess above the melting points of the slag and iron is therefore the only effective smelting heat which they possess, and is a direct measure of their smelting capacity. Any rise in the maximum temperature of the furnace gases increases in a large proportion the part of their sensible heat available above this necessary smelting temperature, and thus increases largely their smelting power.

When the smelting power of the carbon burned in the tuyere region is increased, more iron and slag can be smelted per unit of carbon there oxidized, and therefore the fuel consumption in the smelting zone is greatly decreased per unit of pig iron produced. This decrease of carbon consumption in the smelting zone does not necessitate the use of more carbon for reducing purposes in the upper part of the furnace, because there is ordinarily produced in the smelting region by the oxygen of the blast such a large excess of CO gas that only a small fraction of it can be possibly used up in reducing the ore. The poor ratio of CO^2 to CO in the gases of the ordinary furnace is not due to imperfect reduction of ore by CO, but to the fact that such a great excess of CO gas is formed in the tuyere region that the oxygen of the ore can only convert a small fraction of this CO into CO^2.

The smaller amount of carbon consumed in the tuyere zone when using dried blast is therefore more perfectly changed into CO^2 in the upper part of the furnace, during the reduction of the ore, and therefore gives greater thermal service in the furnace per unit of carbon consumed. This improvement in the ratio of CO^2 to CO in the throat gases is a direct result of the smaller carbon requirement in the smelting region.

The smaller amount of carbon consumed per unit of pig iron or of burden charged, results in a smaller volume of CO and N^2 gases, rising through the furnace per unit of burden. They therefore perform reduction more thoroughly, and become cooler before leaving the furnace. While, therefore, the smelting zone is working hotter, the upper part of the furnace works cooler.

Since the furnace works faster, the radiation and the conduction losses per day being no higher or even less than before, these losses expressed per unit of output become lessened.

Since less blast is required per unit of product, the stoves can heat it hotter and the blowing engines have less work to do, unless the blast is kept constant and the output increased to its maximum. In any case, the blowing engine power per unit of output is decreased.

Besides these direct results of the higher temperature in the tuyere region there are secondary advantages of removing or rendering uniformly low the moisture in the blast, which are of great value to the management of the furnace. These are the regularity of running, of output, of fuel consumption and of grade of pig iron produced. The manager does not have to fear sudden changes in the quality of iron produced, nor to carry an excess of fuel in the furnace charge in order to provide against such unforeseen changes. This regularity gives such great relief to the manager that the practical man may well count it as the chief advantage, as far as he is concerned, of using the dried blast.

But, uniformity and regularity are only one side of the advantages of dried blast; the other side is the economy resulting directly from the absence of high moisture and the consequent high temperature in the tuyere region. *Regular and uniform running at high efficiency* is the combination which in its entirety has made such a great advance in practical blast furnace running with dried blast.

Quantitative Considerations.—With dry air blown upon incandescent carbon, the maximum temperature of the gaseous products of combustion, CO and N^2, will be 1678° for cold

blast, and 84° hotter for every 100° that the blast is heated. With moisture present in the blast, these temperatures may be reduced as much as 235° when the air is warm and saturated with moisture, giving a theoretical maximum temperature of only 1443° with unheated blast. Details of these calculations, and tabulation of temperatures for various atmospheric conditions and temperatures of blast may be found in the writer's *Metallurgical Calculations*, Part II, pp. 282-286.

As an illustration, if the outside air carries 40 grams of moisture per cubic meter and the blast is heated to 500°, the maximum temperature of the gases of combustion in the tuyere region is 1848°, while if the moisture is reduced to 5 grams, and the blast temperature is unchanged, the maximum gas temperature rises to 2061°.

Assuming a smelting temperature of 1250° as required while melting is being performed, the first case gives an excess, or temperature head, of 598°, available to transfer heat to the charge as it is being smelted; the second case gives an excess of 811°, or an increase of temperature head of 213°, or 36 per cent. Under these conditions, the furnace would smelt down charge approximately 36 per cent faster with the dried blast than with the moist. Admittedly, these atmospheric conditions are extremes. The average increase on the Atlantic Coast of the United States has been 15 to 20 per cent for the year's running.

The three blast furnaces producing the largest number of tons of pig iron in 1909, in the United States, *were all three running on dried blast.*

The increased smelting power per unit of carbon burned in the tuyere region, when using dried blast, is also calculable. When a kilogram of carbon is burned to CO, it evolves 2430 calories; if the blast necessary is preheated, there is added to this the sensible heat in the blast. The total available smelting heat per kilogram of carbon, using dry-air blast, can be thus calculated, e. g.:

Temperature of Blast.	Available Heat.
0°	2430 Cal.
300°	2847 Cal.
600°	3286 Cal.
900°	3746 Cal.

If, however, moisture is present, it rapidly absorbs heat, and reduces the available heat in the smelting region. Its effect is therefore to leave less heat available at a lower initial maximum temperature. Taking again the example of blast at 500° and air with 40 grams of moisture to the cubic meter, the total heat available is 3137 calories from combustion and the blast, diminished by 574 required to decompose the water vapor, leaving only 2563 calories available. The calories available for smelting purposes in the tuyere region is thus decreased 18 per cent by the presence of the moisture. Remove this moisture, and the weight of carbon necessary to be burned in the tuyere region to do a given amount of smelting work will be reduced 18 per cent, i.e., the fuel requirement of the furnace will be reduced at least 18 per cent, calculating per unit of output. Other incidental advantages will increase the saving even above the calculated 18 per cent.

If less carbon is burned in the tuyere region per unit of output, the oxygen from the solid charges given off by reduction, and which is fixed in amount, will produce a relatively greater proportion of CO^2 in the issuing gases. This means that a greater proportion of the CO formed will be converted into CO^2, and therefore a greater utilization in the furnace of the calorific power of the carbon of the fuel. For example, if in a given furnace 75.3 kg of carbon is burned at the tuyeres to CO, producing there 175 kg of CO, and the solid charges give up in being reduced 41.8 kg of oxygen, this can convert 73.15 kg of CO into CO^2, and the total heat of oxidation of the carbon to the proportion of CO and CO^2 can be calculated as 6119 calories per kilogram of fixed carbon consumed. If by using dried blast 18 per cent less carbon is thus consumed (giving, however, just as much available heat in the tuyere region) and 61.75 kg is burned to CO, producing 144.1 kg of CO, 73.15 kg of this will be likewise converted into CO^2 by the 41.8 kg of oxygen from reduction (producing just the same amount of heat in the upper part of the furnace) and the total heat of oxidation of the carbon to this proportion of CO and CO^2 can be calculated as 6928 calories. The carbon burned in the furnace is therefore utilized with some 13 per cent greater thermal efficiency when using the smaller quantity in the tuyere region required by dried blast.

Similar calculations could be made as to the smaller velocity with which the smaller amount of cooler gas passes through the upper part of the furnace, and the consequent better utilization of its reducing power and of its sensible heat in heating the descending charges. Analogous calculations could be made showing quantitatively the smaller radiation and conduction loss per unit of product, when using dried blast. The data for such calculations are, however, not as abundant or as trustworthy as those already used. A longer experience with dried blast in practice will enable these data to be also used quantitatively, with convincing results.

Resumé.—The drying of the blast increases the available heat, in calories, in the smelting region, per unit of carbon there consumed. This increase of available heat quantity increases correspondingly the maximum temperature of the gases in the smelting region. The increased quantity of heat, available at a higher initial temperature, increases in a multiplied proportion the smelting power of the furnace, i.e., increases the rate of smelting down in unit time and increases the quantity of smelting down done per unit of fuel. As secondary or incidental results of these favorable conditions, the furnace works cooler above, the proportion of CO^2 to CO in the gases is increased, the radiation and conduction losses are decreased per unit of output, and the furnace works with great regularity and uniformity. The given analysis explains satisfactorily the observed facts of practice, and calculations based on the above principles explain quantitatively, with satisfactory concordance, the observed economies and advantages.

Material and Heat Balance Sheet of the Blast Furnace, by W. G. Gillhausen, of Aachen. This was a very praiseworthy attempt to get accurate balance sheets of four furnaces, making respectively (I) 10 per cent spiegeleisen, (II) open-hearth iron, (III) Hematite iron, and (IV) basic Bessemer iron. The 40 large quarto pages give all the details, which may serve as the basis for a great many arguments and conclusions; they contain the results of an enormous amount of work, for instance, besides all the analyses of materials charged and pig iron produced, 40 to 60 gas analyses on each furnace, equally many determinations of heat in pig iron and in the slag. Test I lasted one month, test II three days, test III and IV each five days.

The best of the tests was undoubtedly the first, and we therefore reproduce the conclusions of this investigation.

Charged.		Flue Dust.	Slag.	Pig Iron.		Gases.	
Fe^2O^3	1113.99	5.25		Fe	843.23	O	363.28
FeO	137.78		2.64				
Mn^3O^4	184.77	6.00					
Mn^2O^3	9.32			Mn	105.70	O	50.63
MnO^2	4.65						
MnO	7.70		65.54				
CaO	328.25	14.84	284.25			O	8.66
MgO	81.55	2.83	78.72				
SiO	342.20	18.00	290.08	Si	24.03	O	12.76
$AlPO^4$	7.04	5.59	85.11				
Fe^2O^3	1.67	0.20		P	0.85	O	0.06
SO^3	0.80	0.17				O	0.03
SiO^2	9.74					O	4.86
S	1.10			Si	9.19		
CaO	7.75			Cu	2.00	O	0.35
CO	108.42					{ C	111.39
C	991.20	4.80		C	46.00	{ O	295.13
H	8.05					C	919.44
N	6.00					H	8.05
Ash	0.01	165.01				N	6.00
H^2O (chem.)	17.95						
H^2O (mech.)	218.71					H^2O	214.66
		50.23	891.53		1000.00		

The gases contained (dried) CO^2 11.40, CO 27.90, CH^4 2.59, H^2 2.30 and N^2 55.81 per cent. Temperature 350° C. The blast was at 740° mean barometer 748 mm, temperature of air 3.3°, moisture 9 grams per cubic meter. From these data, the blast per 1000 of pig iron was calculated as containing H^2O 30.66 kg, O^2 994 kg, N^2 3283 kg.

From these data, the following heat balance sheet was calculated:

RECEIVED (per 100 kg. of pig iron).			
Combustion of carbon	328,676 Cal.	=	80.0%
Heat in blast	81,567 "	=	19.9%
Heat in charges	235 "	=	0.1%
	410,458 "	=	100.0%
DISTRIBUTION.			
Reduction of iron oxides	148,551 Cal.	=	36.2%
" " silica	8,850 "	=	2.2%
" " manganese oxides	23,854 "	=	5.8%
" " phosphoric oxide	532 "	=	0.1%
" " copper oxide	45 "	=	0.0%
" " water (decomposed)	9,793 "	=	2.4%
(Total used in reductions)	(191,625) "	=	(46.7%)
Decomposition of hydrates	121 "	=	0.0%
" " carbonates	33,376 "	=	8.1%
Evaporation of water	16,638 "	=	4.1%
Sensible heat in dust	352 "	=	0.1%
" " " gases	53,718 "	=	13.1%
" " " slag	37,146 "	=	9.1%
" " " pig iron	26,500 "	=	6.5%
Radiation and conduction (difference)	50,982 "	=	12.3%

In connection with this balance sheet, which is the best of the four, the following criticisms are almost self-evident, and should be considered by anyone making use of the figures:

1. The latent heat of vaporization of the moisture in the blast should *not* be reckoned in, because it comes to the furnace already as vapor. If it is counted as available heat, then the heat necessary to decompose the moisture of the blast considered as *liquid* water should be used. It is a mistake to reckon it on one side and not on the other. This makes a difference of 1858 calories.

2. The volume of gases was incorrectly calculated, since the carbon in the flue dust was not subtracted. This made too much carbon calculated as consumed. This mistake makes a difference of 1643 calories.

3. The heat of formation of the slag from its constituent oxides is omitted altogether. This can be approximately calculated, but not accurately. It is better to calculate it than to leave it out altogether. It amounts approximately to 5300 calories.

4. The heat of formation of the pig iron from its constituents is omitted. This is not accurately known, but can be estimated in this case at 3306 calories.

5. The heat evolution is therefore probably about 5000 calories too low.

6. The moisture in the blast, outside air, at 3.3° C. and 748 mm. barometer, is given as 9 grams per cubic meter. This is at least double what it could possibly have been, and the quantity of moisture in the blast and heat absorbed in decomposing it is probably at least double what it should be given.

For various reasons, not easy to discern, tests II, III and IV were unreliable in that the heat balance sheets did not check. They left over for radiation and conduction only 5.08 per cent, 0.58 per cent and even —6.20 per cent, and it is very difficult to fix the unreliable factors. Test I has been given in detail because it seemed the most reliable of the four. We congratulate the author, Herr Gillhausen, on having done a splendid piece of work, entailing an enormous amount of time and effort, such as has probably not been accomplished since the classic researches of Sir Lowthian Bell, and we hope he will accept our comments on their small imperfections in the same spirit as they are made.

The Theoretical and Practical Importance of the Electric Pig Iron Furnace was discussed by Privatdozent Ingenieur Carl Brisker, of Leoben. He considers the electric furnace as even now able to hold its own in places where power is cheap, and that works manufacturing special steels will soon be able to utilize them to great advantage for producing the specially pure pig iron which they need.

He studies in detail the reactions in the Domnarfvet furnace in Sweden, making the assumption that the iron oxides are reduced according to the following reactions:

$$Fe^2O^3 + 2C = 2Fe + CO + CO^2$$
$$Fe^3O^4 + 3C = 3Fe + 2CO + CO^2$$

This assumption is supported by analyses of the gases just above the smelting zone, where there was found 33.6 per cent CO^2 and 56.3 per cent CO in the gases, and there was calculated from the above reactions 37.4 per cent CO^2 and 54.5 per cent CO.

Further up, at the throat, the gases were found to be 41 per cent CO^2 and 45.6 per cent CO, which change the author ascribes to the splitting up of CO into C and CO^2 in the upper part of the furnace shaft. This would require the deposition of 1.73 kg C from the CO gas, and set free an amount of heat sufficient to warm the gases 337°, at that part of the furnace shaft where the reaction takes place. The author does not prove this by temperature measurements taken in the furnace.

He regards this carbon deposition as of great importance to the reduction in the electric shaft furnace, and as resulting in a much better ratio of CO^2 to CO in the issuing gases. The gases, to take advantage of this reaction, must be cooled to the necessary degree by passing through a tall shaft, else the reaction cannot take place. An interesting and instructive comparison of the heat balance sheet is made with an average coke blast furnace, as follows:

HEAT RECEIVED (per 100 kg. of pig iron).	Electric Furnace.			Blast Furnace.		
Electric current	254,900 Cal.	=	70.8%			
Combustion of carbon	105,100 "	=	29.2%	237,500 Cal.	=	84.7%
Hot blast				43,200 "	=	15.3%
Totals	360,000 "	=	100.0%	280,700 "	=	100.0%
HEAT DISTRIBUTION.						
For reductions	170,147 Cal.	=	47.2%	170,147 Cal.	=	60.6%
Heat in iron	26,500 "	=	7.4%	26,500 "	=	9.4%
Heat in slag	12,628 "	=	3.5%	16,100 "	=	5.7%
Decomps'n of moisture	6,496 "	=	1.8%	8,303 "	=	2.9%
Heat in gases	3,080 "	=	0.8%	20,000 "	=	7.1%
Cooling water	77,500 "	=	21.5%	19,000 "	=	6.8%
Radiation and conduction	63,649 "	=	17.8%	20,650 "	=	7.5%
Totals	360,000 "	=	100.0%	280,700 "	=	100.0%

The extraordinary outlay of nearly 40 per cent in the last two items in the electric furnace, as against 14.3 per cent for the same item in the larger coke furnaces leads to the conclusion that as soon as the electric furnace is run on anything like the scale of the blast furnace in question (200 tons per day), a saving of at least one-half of this loss will be made, amounting to a saving of 70,000 calories per 100 kg of pig iron, and since this will all count on the electric current used, it will mean a reduction from 254,900 to 184,900 calories required for the current, or a saving of 28 per cent in power; or, putting it in another way, a reduction from 0.46 hp-year per ton of pig iron to 0.33 hp-year; the reviewer believes it will even be reduced to 0.28 hp-year or less.

Microscopic Examination of Blast Furnace Slag for Cement Making is recommended by Dr. H. Passow, of Blankensee, as especially valuable in determining the value of the slag for the manufacture of Portland cement. Only the vitreous slags have a market value for cement making, and while chemical analysis shows no distinction whatever between vitreous and non-vitreous, the microscope must be constantly resorted to, since it shows at once whether granulation or crushing has yielded vitreous or non-vitreous slag particles.

When the microscope shows that granulation or crushing or the subsequent drying has yielded a too large proportion of non-vitreous particles, then suitable remedies must be applied at once to stop this action. Numerous data and micrographs were shown to substantiate this. (Figs. 9 and 10.)

The microscope is also very useful in controlling the qualities of new blast-furnace slags, the granulated or atomized slag can be examined at once, and the strength of air blast or water current varied to give the best vitreous product. Slags with SiO^2, 27 to 31 per cent; Al^2O^3, 13 per cent to 20 per cent, and CaS, 4 per cent to 8 per cent, are the best reacting slags

for cement manufacture, not needing nearly so fine grinding as those higher in SiO^2 and lower in Al^2O^3 and CaS. Such slags are somewhat more difficult to granulate, however; they must be quickly but uniformly cooled; the Buderuss Iron Works have arranged a trommel which does this very well. The slower reacting slags are easily vitrified, but require larger additions and more fine grinding to make good cement.

The author concludes that the slag-cement business can only

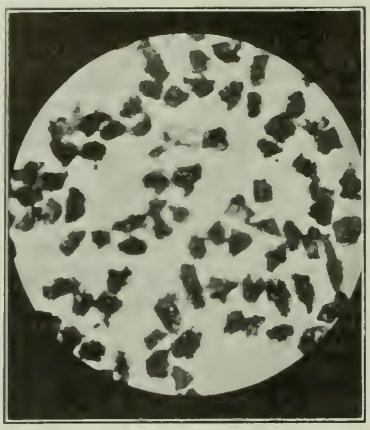


FIG. 9.—NON-VITRIFIED SLAG.

flourish by using the microscope freely as a constant guide in controlling the quality and granulation of the slag.

The Status of Cleaning Blast Furnace Gases in Germany was handled by Ober-Ingenieur C. Grosse, of Metz. The German blast furnaces produce 13,600,000 tons of pig iron yearly, corresponding to 7,650,000 cu. m of waste gas hourly, of which 4,840,000 cu. m, or 63 per cent, is being cleaned. The modern requirements are: (1) the gases must be cooled as

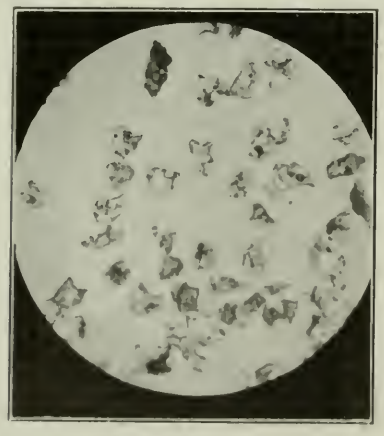


FIG. 10.—VITRIFIED SLAG.

much as possible to free them from water vapor, (2) the gas for heating purposes should be cleaned to 0.1 to 0.5 gram of dust per cubic meter, (3) the gas for gas-engine service should be cleaned to 0.01 to 0.03 grams of dust per cubic meter at 25° C., (4) all apparatus should be simple and reliable. The cooler the gas is made the more moisture is removed and the easier it is to remove dust because of its smaller volume.

The four principal mechanical purifiers are the Zschocke, Theisen, Bean and Schwarz, which purify, respectively, 44 per cent, 33 per cent, 14 per cent and 5 per cent of all the gas cleaned. The Flössel (Fig. 11) has experimentally given better results than the Theisen. The proper cleaning of the gases causes a saving of 20 kg to 25 kg of fuel per ton of iron produced, the blast is usually heated 60° to 100° hotter, the heating-up periods of the stoves are reduced almost one-half, and the "on-blast" periods are lengthened half an hour. The stoves require on an average 42 per cent to 44 per cent of the whole gas cleaned, the blowing engine 16 per cent to 18 per cent, leaving 37 per cent to 42 per cent for outside use. Used under boilers, the cleaned gas gives 10 per cent more steam.

The average boiler campaign between cleanings is raised from one month to three or four months, the hot-blast stove campaign from two or three months to 12 to 14 months. The chimney gases from the Cowpers should not run over 300° C., but higher Cowpers are necessary with the cleaned gas because of its higher heating power; stoves up to 40 m (130 ft.) in height have been constructed with 6 m to 7 m (19 ft. to 23 ft.) diameter. The author thinks that in a short time all the blast-furnace gas produced in Germany will be properly cleaned and utilized.

Large Gas Engine Installations was treated by Prof. Langer, of Aachen. Improvements have been made in reducing the mean effective piston pressure from 5.5 kg to 4.5 kg per cubic centimeter (82 lb. to 67 lb. per square inch), with a regular total load factor of 70 per cent. The fundamental condition of satisfactory working was said to be "a high excess of

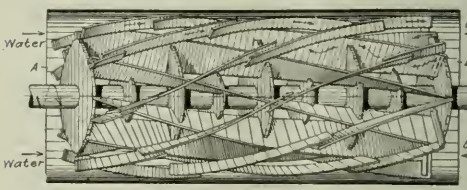


FIG. 11.—BLAST-FURNACE GAS PURIFIER OF FLOESSEL.

air in the mixture." This conclusion was not agreed to by everybody.

The Status of Foundry Casting in Germany was described by Ingenieur Irresberger, of Mülheim (Ruhr). The foundry is no longer neglected in Germany, but now employs technical graduates, special engineers, chemists, mechanical testing laboratories, costly mechanical outfit for molding, transporting, charging cupolas, etc. Great attention has been given lately to sand-blast cleaning of castings, the workman being protected by an air-shield. Many foundries run only on specialties. In Germany 120,000 hands are employed in 1560 foundries producing annually 2,000,000 tons of castings. The traveling cupola charging machines (Fig. 12), sand-blast machines, and other cleaning devices were described at length. Many foundries have a small laboratory simply to determine silicon in the iron. Low-carbon iron is increasingly used for strong castings; many foundries produce it by charging into the cupola Ronay briquettes of soft-steel and wrought-iron turnings.

Cores are often made of sand, meal and glutrose and baked in ovens heated by hot water to 250° C. to 300° C. Such cores are exceptionally resistant during casting, and later fall to powder. Molds are usually dried by burning gas; at some foundries cleaned blast-furnace gas has been used with best results. Blast furnace foundries, which take their melted iron directly from the mixer, are increasing rapidly in numbers.

The Status of the Basic Bessemer Process in Germany was the subject of a splendid report by Superintendent Esser, of Differdingen. The production has risen from 329,000 tons in 1883 and 3,004,000 tons in 1890 to 7,668,600 tons in 1900. The number of converters running in the years named was 47,

70 and 108, and the converter charge averaged 9 tons, 12 tons and 17 tons, respectively. In general the original form of converter has been preserved, but the large new ones have a shallower bath; in the 15-ton converters the bath was 0.7 m to 0.75 m (28 in. to 30 in.) deep, but in the new 25-ton converters now in use it is only 0.45 m to 0.5 m (18 in. to 20 in.). This gives much less resistance for the blast to overcome, requires less blowing-machine power, and saves 1 per cent to 1½ per cent loss of iron. The Jünkerather Company are building a 30-ton converter which will be 6.8 m (22 ft.) high and weigh 70 tons without the charge and 100 tons when charged.

Pig-iron mixers of the rolling type are now in use up to 1000 tons capacity; hydraulic power is used on the converters, but electric motors on the mixers. Electric locomotives are in general use to transport the pig iron, provided with controllers for tilting the ladles, all under the hand of one man. For handling molten steel the author prefers electrically operated casting ladles, but electrically operated high-pressure pumps for raising and lowering the heavy ladles.

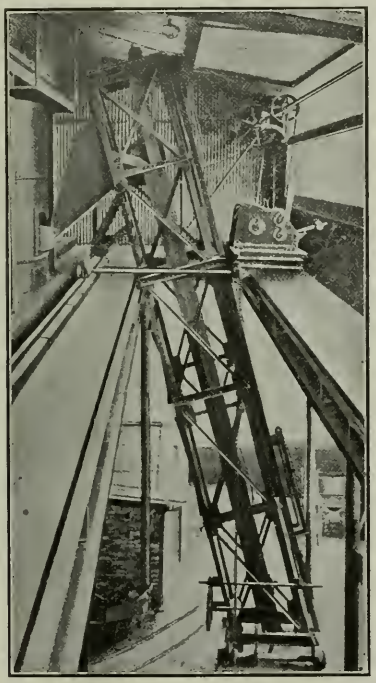


FIG. 12.—TRAVELING CUPOLA. CHARGING MACHINE.

For filling the converters and casting into molds overhead cranes are probably better than any other arrangement. Casting into moulds on trucks, from a ladle held by a traveling crane, has great advantages: the ingots thus cast are better than stationary ingots, because the bumping which they get while being moved brings to the surface gases, oxides and other impurities, and improves their quality. Lime is brought directly from the lime kiln to the converter mouth by a rope railway.

Fig. 13 shows the arrangement of the converter building in Rote Erde.

The converter bodies now stand about 280 charges, the bottoms when furnished with tuyere bricks 95 charges, if made with sample perforations 60-65 charges. This result has been attained mostly by the use of carefully chosen lining material, and above all by using compressed air stamps for ramming in the lining. The same are used for lining the steel ladles, and their service has been raised from 25 up to 50 or even 60 charges.

Where the bessemer works is far from the blast furnaces, it is advisable to run the pig iron into a mixer, and to cast only from the mixer. In this way a uniform quality of pig iron is furnished to the steel works. The great use of blast-furnace gases for power is driving the steel works to the ore regions, so that the iron industry is at present moving away from the coal to the ore.

The "Minette" deposits of Lothringen, Lorraine and Luxemburg are the chief source of the basic pig iron. Dr. Kohlman estimates that there is 5,000,000,000 tons of this ore. There are two principal varieties, the lime gangue variety and the silica gangue, which average about

	Lime Gangue.	Silica Gangue.
Iron	28-37%	32-40%
Lime	12-20%	7-9 %
Silica	6-9 %	12-22%
Phosphorous	0.6-0.7%	0.6-0.7%

The usual ore mixture is 2/3 calciferous to 1/3 siliceous ore. It appears as if there is Minette ore to last a century.

It has been said against the basic bessemer process that it is not possible to make by it steel low enough in phosphorus. The reason why phosphorus is not usually blown below 0.06 per cent is not that the waste of iron is so much greater, but that by this further oxidation the steel is made useless by taking up an unbearable amount of oxygen.

The most important consideration at this point of the process is not whether the steel contains 0.06 or 0.04 or even less phosphorus, but that it should be made with as little oxygen in it as possible; the question to be decided is to keep the absorption of oxygen down and to take out the oxygen which has been absorbed.

If rails are called for with less than 0.04 per cent phosphorus, one would not willingly try to make them in the basic bessemer; it cannot be denied that it might be possible to bring the phosphorous that low, but the danger of "over-oxidation" stands imperatively in the way. The author does not think that a 0.04 phosphorus rail made with this danger of over-oxidation would average any better quality than the ordinary 0.06 to 0.07 phosphorus rail made with the ordinary degree of oxidation; he does not believe that foreigners will continue long to pay 20 marks ($5) per ton extra for such low phosphorus open-hearth rails.

Basic bessemer metal is hottest at the end of the blow, and is always let stand several minutes before casting, in order to give time for the final additions to become well alloyed. As the last of the slag is poured off, the ferro-manganese is added; the charge is then allowed to rest in the converter 7 to 8 minutes, to give a good alloy, allow gases to rise, oxygen compounds to float up, and any other impurities. The thin cover of slag on the bath at this time keeps in the heat and keeps out the air from the bath. Working in this way, the steel is much improved and casts well.

Comparing the basic bessemer with the basic open-hearth process, the comparison is not as bad for the bessemer process as Dichman and some others have tried to make it. From 1000 tons of pig-iron the bessemer makes 890 tons of steel, at a cost of 49.76 marks ($12.44) per ton, while the open hearth makes at the most 1100 tons, at a cost of 51.80 marks ($12.95) per ton. The basic bessemer process has made extraordinary progress in economy and in quality of product, so that it can say, in the immortal words of Mark Twain: "The news of my death is highly exaggerated."

The Pig and Ore Process in Germany, by Direktor R. Genzmer, of the Julienhütte in Upper Silesia, may be considered as a counter-stroke to Superintendent Esser's paper on the basic bessemer process. It is a fine review of the status of this process at the present moment, in Germany.

It would be impossible to make steel with economy in the

open-hearth furnace by oxidizing pig iron by the furnace gases, but it is done economically and quickly by using iron ore as the oxidizing agent. When this is done in furnaces as large as 50 tons capacity, making 200 tons per day, it becomes an active competitor to the basic bessemer process. The Talbot process is not used in Germany, the Duplex process is scarcely known there, and the Monell process seems (under this name) not to be known to Herr Genzmer, so he divides the pig and ore processes into the following:

I. Pig and ore processes, using liquid pig iron below 1 per cent phosphorus:

(a) Taken direct from the blast furnace.

(b) Taken from an ordinary gas-heated mixer.

(c) Taken from a mixer used as a pre-oxidizer.

II. The Bertrand-Thiel or Hoesch process using liquid pig-iron with over 1 per cent phosphorus (basic bessemer pig).

The question of the "mixer" is a highly important one for the steel maker, particularly that of the gas-heated mixer used as a preliminary oxidizer, but many of the German works which have made great improvements in this line refuse to let their improvements be published.

The use of low phosphorus pig iron direct from the blast-furnace is practised at the Hubertushutte in Upper Silesia, the open-hearth furnaces being fired by coke-oven gas. Fifteen

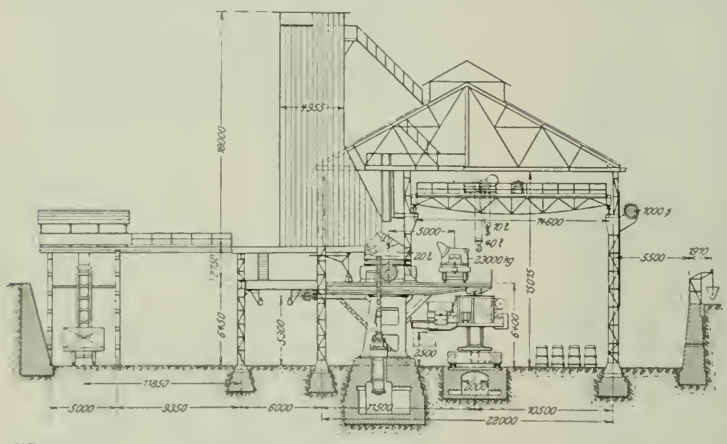


FIG. 13.—CONVERTER BUILDING IN ROTE ERDE.

tons of liquid iron is brought directly from the blast furnace to the 25-ton open hearth, and run into it from the bottom of the ladle.

This is not so easy a way as pouring from the ladle, but it has been found of advantage because the shaking during transport from the blast furnace seems to bring the sulphur (as manganese sulphide) to the surface of the metal, and by tapping from beneath this is left in the ladle. As much as 30 per cent of the sulphur is thus taken out of the pig-iron.

Ore and lime are charged into the furnace before the liquid pig is poured in; if the ore is somewhat preheated the reaction may be quite violent," a foamy slag forms and the ore disappears inside of an hour. Analyses are given of the metal during a heat, in which 224 tons of pig iron was poured upon 4.7 tons of Swedish magnetic ore and 0.9 ton south Russian ore:

	C	Si	Mn	P	S
Pig iron used	3.61	1.89	1.41	0.670	0.044
After reaction (1 hour)	2.51	0.50	0.018	0.005	0.030
At tapping (4¼ hours)	0.087	0.004	0.499	0.067	0.042

The failure to remove the slag at the end of the reaction is the cause of the strong re-phosphorization, which at one time (4½ hours) amounted to 0.140 per cent. More recently, it is said, that this work is removing half the slag at the end of the reaction period, and since it contains 15 to 18 per cent of iron, the output is reduced about 1 per cent by so doing. The end slag carries 9 to 11 per cent FeO and a similar amount of MnO, with 5.6 per cent P^2O^5 and 20.25 per cent SiO^2.

The use of a gas-heated mixer is a great advantage. In it sulphur is always reduced, usually 20 to 30 per cent. Average analyses of pig iron and mixer metal, at the Julienhütte showed:

	Si.	Mn.	P.	S.
Pig Iron	1.93	2.03	..	0.092
Mixer Metal	1.43	1.72	0.43	0.063

The 150-ton mixer is electrically tilted, heated by hot producer gas, with regeneration for air only. Its temperature is kept between 1300° and 1400° C. The coal used is only 8 tons for 400 to 500 tons of pig-iron passed through per day. Only a small amount of slag is made, 1½ to 2½ tons per day, consisting of 27 to 35 per cent FeO, 29 to 37 per cent SiO^2, 13 to 18 per cent MnO, and 0.35 to 0.40 per cent P^2O^5. The hearth is magnesite, roof silica brick; it has run 1½ years without any stop, and repairs are exceedingly small.

In the furnace building, the furnaces of 40-ton capacity are served by overhead traveling cranes. Each furnace has its own producers, the total coal consumed is 20 per cent of the steel output, including mixer, Sunday heating, etc. Four charges are run per day. The charge is 70 to 80 per cent pig iron. At the end of the reaction period, part of the slag is run off, particularly when better quality steel is being made; the first slag carries 20 to 30 per cent Fe.

Mixers of 125 to 250 tons capacity used as oxidizers are in use at a number of German works. At the Georgmarienhutte (Fig. 14) the mixer is built like a Wellman furnace, but the practice differs from the Talbot furnace in that the oxidation is only carried a little way, and then the metal, which is still pig-metal, removed to the steel furnaces. The hearth is magnesite brick covered with burnt dolomite, the roof is Dinas brick. It is heated by purified blast-furnace gas, brought by a 0.8 meter diameter conduit. The same gas is used for the soaking pits, and will later be used in the Siemens-Martin furnaces. After the pig iron is run into the mixer, 5 to 6 per cent of its weight of iron ore and 3 to 4 per cent of lime is added. Per day about 4165 tons is put through this mixer, 354 tons of Swedish magnetite is added, producing 358 tons of slag, containing 6.86 per cent Fe. (24 tons altogether); 90 per cent of the iron of the ore added is reduced into the bath.

	C.	Si.	Mn.	P.	S.
Pig Iron	3.80	0.6-1.0	2.75	0.46	0.05
Mixer Metal	3.50	0.08	1.30	0.15	0.02

In the open-hearth furnaces, per day, 715 tons of ore was used, producing 2.24 per cent of reaction slag with 25.5 per cent Fe and 7.18 per cent of end slag with 0.4 per cent Fe. The total slag production, including the mixer slag, was therefore 14.8 per cent. In the open-hearth reactions 83 per cent of the iron in the ore was reduced; of the total ore used, 85.2 per cent of its iron was reduced into the metal. One 50-ton furnace made 1464 tons of high grade steel in 6 days.

The use of the mixer as a pre-oxidizer has these advantages: (1) Very small repair costs; (2) only two men to handle it; (3) only 2.65 per cent dolomite needed; (4) 103 to 104 per

* [It is the pure and simple Monell process, if the slag formed is removed at the end of the reaction.—Translator.]

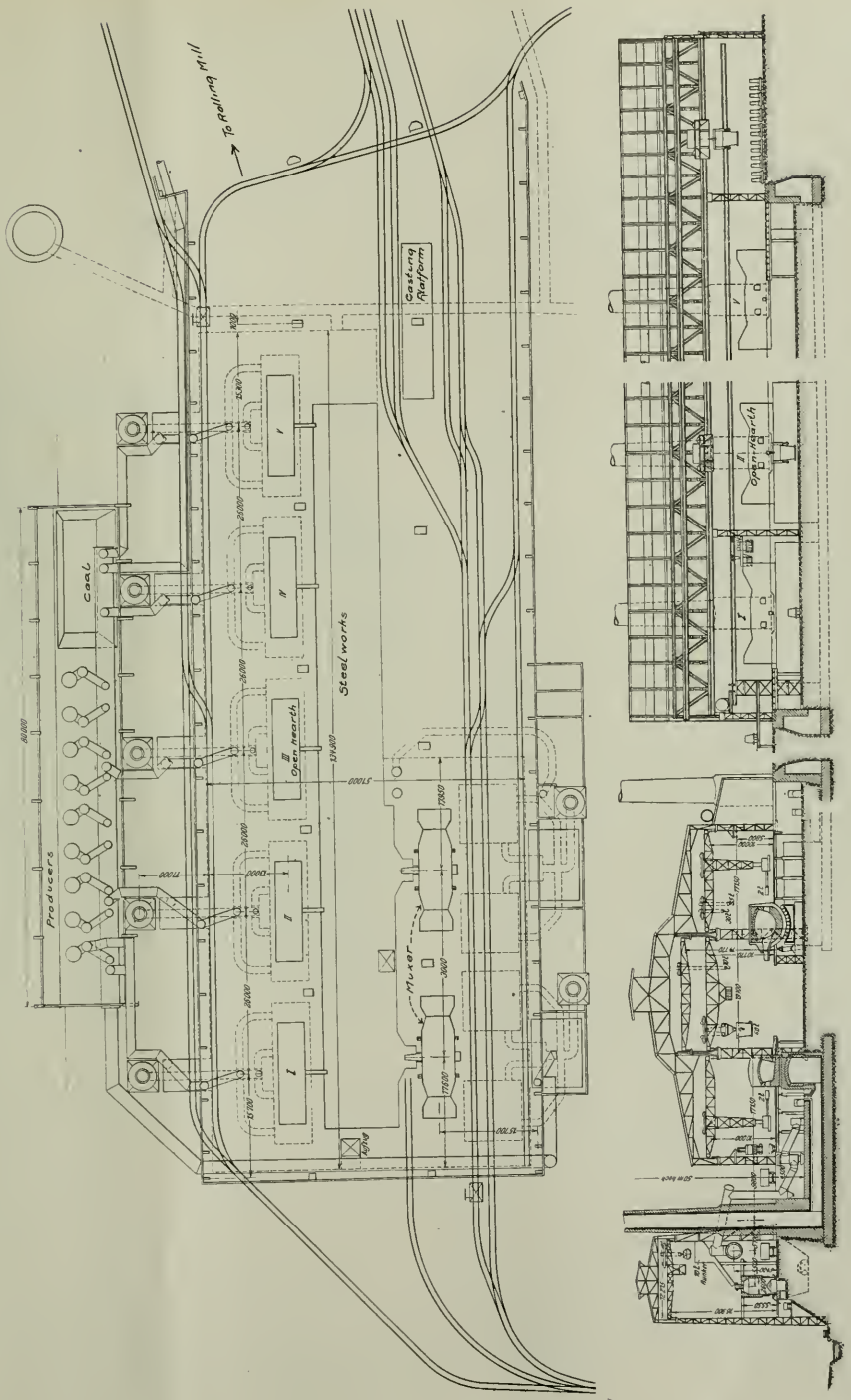


FIG. 14.—OPEN-HEARTH STEEL PLANT OF GEORGSMARIENHÜTTE.

cent output, reckoned on the pig iron; (5) only 16.3 per cent coal consumption in the open-hearth furnaces. It really serves as a preliminary open-hearth furnace; kept always at a moderate temperature, and doing part of the work of the open-hearth furnace while the metal is being kept melted as a reservoir.

The Hoesch process is being worked in 30-ton furnaces in Germany, and some 50-ton furnaces are being built. The pig iron is passed through a mixer, gas-fired, where it is not oxidized. So far, only basic bessemer pig iron, with over 1 per cent phosphorus, is thus worked. The furnace is charged with high phosphorus Swedish magnetite, roll cinder and burnt lime, and the molten pig run in upon it. A charge was, for instance, 3.44 tons Swedish ore, 0.77 tons roll scale, 1.88 tons lime, and on this 23.28 tons of pig iron. After the reaction was complete and the charge well heated, in 2 to 3 hours, the whole was tapped into a ladle and the slag run away. The furnace then received immediately a fresh charge of 2.18 tons spathic iron ore, 1.20 tons of lime and 5.412 tons of scrap, and the metal poured back into the furnace. After 1¼ hours heating the charge was well melted, all the ore dissolved and a good slag made; a further treatment of 1½ hours, with further additions of roll scale, lime and finally ferro-maganese, brought the metal to the tapping point.

	C.	Si.	Mn.	P.	S.
Pig Iron	3.28	0.32	0.96	1.86	0.132
After 1st Reaction	1.46	trace	0.34	0.26	0.082
After 2d Reaction	0.38	trace	0.29	0.09	0.100
Tapping	0.08	trace	0.47	0.04	0.067

Slag.	SiO^2	CaO	P^2O^5	FeO	MnO
After 1st Reaction	11.40	48.86	22.13	4.67	3.93
After 2d Reaction	14.20	33.64	6.25	19.64	13.49
At Tapping	12.40	46.28	5.00	17.03	10.25

The dephosphorization is only certain, according to Springorum, when the slag has 40 to 45 per cent CaO and the SiO^2 is below 15%.* The large amount of MnO in the slag tends to keep down oxidation of the bath, since the Mn reduced from it by the carbon of the metal tends to keep the bath deoxidized.

The original pig iron is low in manganese, but the spathic ore used in the second oxidation reaction is high in manganese, and thus provides the necessary MnO for the slag. The output is 104 per cent on the pig iron used; the first slag is high in soluble P^2O^5, and is kept separate and sold.

By thus working, the open-hearth process is able in Germany to compete with the basic bessemer (Thomas) process, using the same pig iron. With larger furnaces of 50 to 80 ton capacity the output is increased so greatly (an 85-ton furnace at Bochum produces 225 tons per day) that the open-hearth furnace is continually cheapening its product, and perhaps will eventually displace the basic bessemer altogether.

Dephosphorization of Iron in Presence of Carbon was discussed by E. Richarme, of Toaritsinsky-Tavod, Russia. His thesis is that 3 (Fe, Mn, Ca, Mg) $O.SiO^2$ has the same dephosphorizing power as 3 (Fe, Mn, Ca, Mg) $O.Al^2O^3$, or 4 (Fe, Mn, Ca, Mg) $O.P^2O^5$; or 2 (Fe, Mn, Ca, Mg) $O.Fe^2O^3$; or putting it another way, that 4 SiO^2, 4 Al^2O^3, 3 P^2O^5 and 2 Fe^2O^3 all have equal powers of keeping phosphorus *out* of the slag, and FeO, MnO, CaO and MgO have equal molecular powers of keeping phosphorus *in* the slag.

Starting with the calcium silicate as a type, 3 $CaO.SiO^2$, containing 26.3 per cent of SiO^2, the following are equivalent slags as far as dephosphorizing power is concerned: 3 $CaO.Al^2O^3$, 4 $CaO.P^2O^5$, 2 $CaO.Fe^2O^3$. The author then makes use of the molecular weights to get the equivalence of Al^2O^3, P^2O^5 and Fe^2O^3 in terms of SiO^2 and of FeO, MnO and MgO in terms of CaO. He then expresses the index number or "acidity" of the slag as the ratio of the summated acids to the summated acids plus summated bases. The finding of the index number is then simple: e.g.:

$$\%\ SiO^2 + \frac{\%\ Al^2O^3}{1.69} + \frac{\%\ P^2O^5}{1.77} + \frac{\%\ Fe^2O^3}{4.00} = \text{Summated acids}$$

$$\%\ CaO + \frac{\%\ MgO}{0.71} + \frac{\%\ FeO}{1.28} + \frac{\%\ MnO}{1.27} = \text{Summated bases}$$

Applying this formula to the actual slags obtained in the basic steel processes, he comes to the following conclusions, which if reliable are very important:

Slags with acidity ratio 0.32 to 0.26 will remove and retain phosphorus in an oxidizing atmosphere.

Slags with acidity ratio 0.26 to 0.22 will remove and retain phosphorus in an oxidizing or a neutral atmosphere.

Slags with acidity ratio below 0.22 will remove and retain phosphorus in an oxidizing, a neutral or a reducing atmosphere.

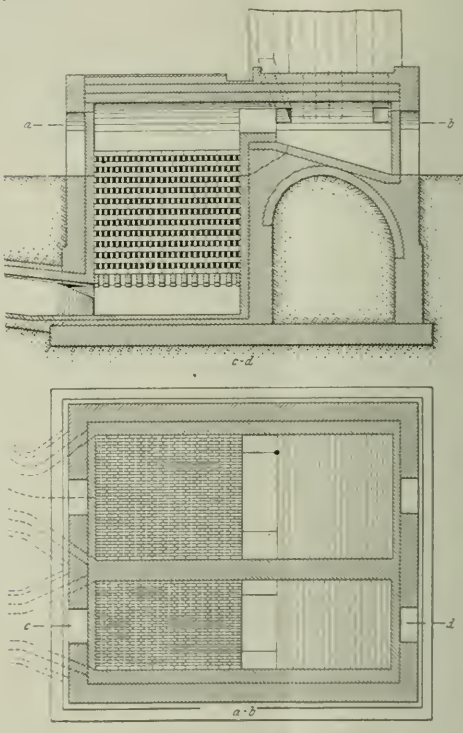


FIG. 15.—ARRANGEMENT OF SLAG POCKETS.

With the latter slags, iron containing carbon can be dephosphorized and all re-phosphorization prevented.

Analyses of a basic steel heat made at Tsaritsin to test this principle showed:

	C	Si.	Mn.	P.	S.
Pig iron	4.00	0.00	1.09	1.93	0.070
Metal 5½ hours	0.79	...	0.23	0.010	0.008
Final metal	0.61	0.32	0.64	0.012	0.006
	P^2O^5	SiO^2	CaO	Fe^2O^3	FeO	MnO
Slag	5.98	9.46	43.95	5.56	17.87	8.56
		C.	Si	Mn.	P	S.
Pig Iron		4.00	0.90	1.90	1.03	0.070
Metal 5½ Hours		0.79	...	0.23	0.010	0.008
Final Metal		0.61	0.32	0.64	0.012	0.006
	P^2O^5	SiO^2	CaO	Fe^2O^3	FeO	MnO
Slag	5.98	9.46	43.05	5.56	17.87	8.56

Index number, or acidity ratio, of the slag, 0.18; metal very quiet in the molds. Test piece, 16 mm (0.64 in.) diameter and 160 mm (6.4 in.) between marks gave 81.2 kg (117,000 lb.) tensile strength and 12 per cent elongation.

Running Open-Hearth Steel Furnaces with Coke-Oven Gas was interestingly discussed by Oberingenieur Terpitz.

*See further in this point abstract of paper by Richarme. [Translator.]

of the Hubertushütte in Upper Silesia. In the new Georgmarienhütte near Ornabruck blast-furnace gas is used for heating a mixer, but attempts to use it for the steel furnaces have so far failed because of the low heating power of the gas Herr Terpitz is personally convinced that in a few years purified blast-furnace gas will be successfully used in the steel furnaces, particularly if it can be enriched a little.

Coke-oven gas is on the contrary richer than producer gas. A plant of 90 Otto-Hofmann coke ovens is used at the Hubertushütte, which treat 320 tons of coal in 24 hours and have 60,000 cu. m to 70,000 cu. m (2,000,000 cu. ft. to 2,500,000 cu. ft.) of excess gas daily, of a calorific value of 3300 Calories per cubic meter. This gas has been used since 1907 for various heating purposes, and in the steel furnaces as far as it would reach. The gas contains: CO^2, 6.5 per cent; CO, 10.6; N^2, 38.7; N^2, 24.8; O^2, 1.0; CH^4, 16.4; C^nH^{2n+2}, 2.0 per cent. There was enough gas to displace 53.15 per cent of the coal used in the producers for running the furnaces.

The high hydrogen in these gases has had absolutely no injurious effect on the steel; the injurious effects usually ascribed to hydrogen are effects caused by too high a temperature in the furnace, and not to the presence of high hydrogen in the gas used.

Since the coke-oven gases are free from ammonia compounds they should give less trouble from nitrogen in the steel than ordinary producer gas, but Herr Terpitz has no definite conclusions on this head; he thinks the influence of nitrogen in steel still doubtful. The steel takes up no sulphur from the gases and Herr Terpitz takes occasion to remark that he thinks it never does, under any circumstances, and that the supposed contamination of the steel by impurities in the gases is wholly imaginary.

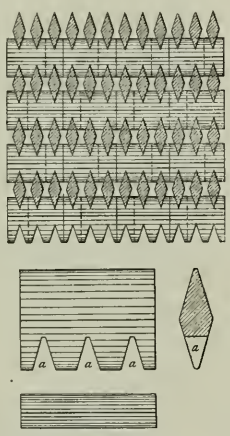


FIG. 16.—CHECKERWORK OF R. SCHERFENBERG.

Concerning imperfect combustion of the hydrogen in the furnace, and its burning in the regenerators, this occurs in their furnaces when fired hard, but not any more with hydrogen than with the CO of the gas; in fact, often less, as analyses show. The higher temperatures of combustion of coke-oven gas have decreased the life of the roof and ports some 8 to 10 per cent. There is no trouble in heating coke-oven gas in the regenerators. If blast-furnace gas and coke-oven gas are both at hand in excess, Herr Terpitz recommends using the former for gas engines and the latter for the open-hearth furnaces. The reviewer would ask why it would not be advantageous for the steel furnaces to *mix* the two gases, since the one being too poor and the other somewhat too rich, a mixture of the two could be easily made with just the desired properties.

Improvements in the Construction of Siemens-Martin Furnaces, by Dipl. Ing. O. Friedrich, of the Julienhütte in Upper Silesia. The rapid development of the open-hearth process, particularly of the pig and ore process, has made great demands on the constructors and workers of these furnaces. The foaming slag rises around the walls even to the ports; fine particles of slag are sometimes carried over by the gases into the regenerators. The voluminous slag also restricts the passage for the heating gases and forces them up against the roof, which then suffers more. The furnaces have also been increased enormously in size, up to steel furnaces for 250 tons (Talbot furnaces) and gas-heated mixers of 1000 tons.

Vertical regenerators are by far preferred to horizontal ones. The air chamber is 1.2 to 1.6 times the size of the gas chamber; 1.4 is recommended as the best proportion, using producer gas. Per ton steel capacity, 1.4 cu. m gas regenerator and 2.0 cu. m. air regenerator space is recommended.

Slag chambers under the furnace are displacing the slag-

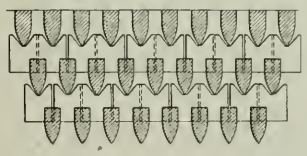


FIG. 17.—CHECKERWORK OF STELLAWERKE.

pockets in the regenerators (Fig. 15). The gases must turn 90° to enter these slag chambers, and this is highly favorable for causing the slag to deposit. The bottom of these chambers is sometimes made removable, on a wagon, so that the fluid slag can be taken away and another wagon inserted very quickly. With the regenerators thus protected, the checker work will stand 2000 to 2500 heats, even on the pig and ore process.

Several of the new forms of checker work in use in Germany are shown in Figs. 16 to 21. Several new forms of gas and air ports are shown in Figs. 22 to 24.

The hearth is usually made 2 to 2½ times as long as broad, in small furnaces; the largest hearths are 4 m (13 ft.) wide and 11 m (36 ft.) long. The bath is seldom over 0.5 m (20 in.) deep. Dolomite is usually used, spread thin upon a background of magnesite bricks. Walls and roof are of silica bricks, except that a lining of magnesite bricks is sometimes run some distance up the walls to protect them from slag in the pig and ore process.

The Present Status of Electric Steel Processes, by Dr. B. Neumann, of Darmstadt, was a review of present systems and their results. At the beginning of 1910 there were 67 electric steel furnaces in operation and 40 building. In 1909 17,773 tons of electric steel was made in Germany, mostly fine steel; but the electric furnace is also making "tonnage" steel, and furnaces of 25-tons capacity are quite possible to build. Without going into a detailed description, the following are some of the latest improvements in electric steel practice:

Héroult Furnace.—The author gave an abstract of the article on the South Chicago plant in the April issue of this journal, page 179.

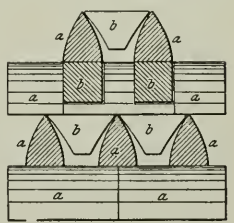


FIG. 18.—CHECKERWORK BRICK OF R. SCHERFENBERG.

Stassano Furnace.—The hearth and side walls are now stamped of burnt dolomite, only the roof is made of magnesite bricks. The roof lasts four to six weeks; the hearth lasts three to four weeks, producing 70 to 100 tons of steel. These figures are for making four heats in 12 hours, and not running nights. When melting scrap and producing steel similar to soft Swedish iron the power used is 750 kw-hours per ton. The bath is deoxidized by silicon and manganese. The rotating furnace has been replaced by a simple tilting construction, which is much cheaper and answers generally quite as well.

Girod Furnace.—The 12-ton furnaces at Ugine are being now charged with 15 to 16 tons of scrap, and 14½ tons are

tapped from them with much better power efficiency.

Chaplet Furnace.—This is used at Allevard; it runs with the same manner of heating and course of the current as the Girod furnace

Keller Furnace.—Has the whole hearth made conducting by

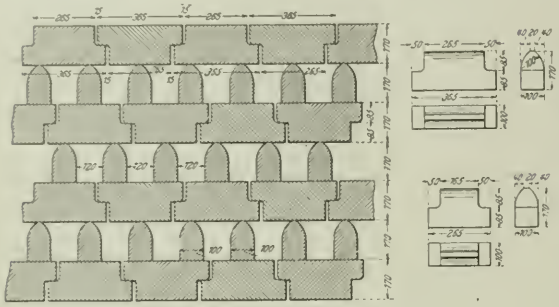


FIG. 19.—CHECKERWORK BRICK OF DIETRICH.

a multitude of small metallic conductors; it has proven itself quite permanent in use. A 3.5-ton furnace of this kind is being installed at Burbach in Germany; this form is preferred to the two electrode Keller furnace.

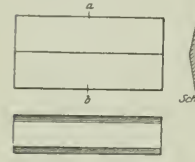
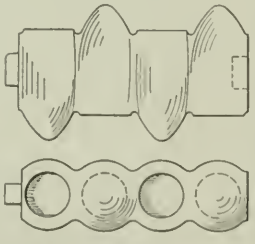


FIG. 20. CHECKERWORK BRICK OF UPPER SILISIAN SCHAMOTTE FABRIK.

Levoz Furnace (Fig. 25).—The furnace body is like a Bessemer converter, but with an electrode passed through the roof and conductors like Keller's hearth built into opposite sides beneath the level of the bath. The whole is mounted on trunnions. This furnace is being tried at a Belgian works.

Nathusius Furnace. — This combines the surface heating of the Héroult furnace with hearth electrodes. Three electrodes of a three-phase system pass through the roof; three steel electrodes, likewise connected with the three-phase system through transformers, are embedded in the hearth, but are covered by the hearth material. As the hearth becomes warm it becomes conducting, and a current of high voltage and small amperage can be sent between these three bottom electrodes, and so the hearth heated to any desired degree (Fig. 26). *T* represents the secondary of a step-down transformer, *T*, that of an auxiliary transformer for heating from the bottom electrodes.



The bottom electrodes can also be then connected in series with the upper electrodes, the furnace then working like a Girod furnace except that the hearth is highly heated by the passage of the current through it. The bath is kept in active motion by the magnetic field created by the current. If the bath is desired to be left standing, the three upper electrodes can be lifted up and the heating left to the bottom electrodes only.

A 5 ton furnace on this system has been working satisfactorily for a year at the Friedenshutte in Upper Silesia (Fig. 27). The upper electrodes receive 550 kva at 100 volts, the lower set, 150 kva. The furnace is 2.73 m (9 ft.) in diameter outside, and 2.17 m (7 ft.) inside, with two doors and a pouring spout. The electrodes are 2 m long by 0.25 m square, and are suspended from beams quite independent of the furnace, so that the latter stands quite free. The roof is of high-alumina firebricks. 5 to 5½ tons of fluid basic Bessemer steel is treated

3½ to 4 hours, using 300 kw-hours per ton of steel. The product is dead soft steel, with 0.002 to 0.017 per cent phosphorus, 0.005 to 0.020 sulphur, and 0.05 to 0.06 per cent carbon.

Kjellin Furnace.—This simple induction furnace has been built up to 8.5 tons capacity. It is not suited for refining purposes, only for melting, and alloying of special steels.

Röchling-Rodenhauser Furnaces.—These are now working up to 8 tons capacity, and a 16-ton furnace is contemplated. This was the first furnace to make electric steel for rails and railway material. The furnaces are started by laying iron rings in the gutters, and pressing the scrap tightly about them; the latter sinters and soon takes part in conducting the current. Molded dolomite bricks are now used to shut off the slag from the central compartment or reservoir, and so keep the iron in the induction rings clean from slag. The bottom of this central reservoir has also been provided with elevations which contract the section through which the electric current flows and produce locally high heating.

When standing over Sunday, it was usual to keep enough current on the furnaces to keep the contents melted; now the furnace is charged, the doors luted tight and the current shut off entirely. After 30 hours' standing the current is put on again, and in 4 to 6 hours the furnace is at running heat.

The secondary current which is sent through pole plates buried in the hearth material is, when starting, run by a higher voltage, in order more quickly to warm up the hearth and make it conducting; as soon as this is attained, the lower voltage is switched on. When cold scrap is being worked, as much as 2 or 3 tons are charged at once into the 8-ton furnace, the doors closed, and the whole melted down at once without opening the furnace. This has been found to take less energy than the continual feeding in of the scrap in small quantities.

The Frick Furnace (Figs. 28 and 29).—The Fried. Krupp Company has put up a 10-ton furnace of this pattern, which is usually run with a 8½-ton charge, of which 6½ tons are tapped. The heat lasts 6½ hours. It is used only for melting down high quality material to fine steel; the material being charged ¼ ton at a time for 4½ hours, and the last two hours

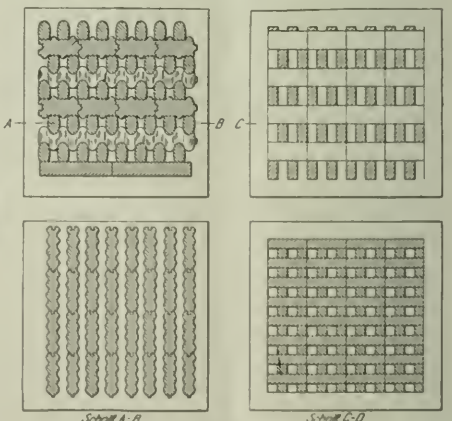


FIG. 21.—CHECKERWORK OF KNOBLAUCH.

being used for heating up to a dead melt. It is an induction furnace which differs from the Kjellin only in the arrangement of the primary windings, which are flat and arranged over and under the melting ring. The outer diameter is 4.4 m

(14 ft.); ring, 2.6 m (9 ft.); breadth of bath, 0.4 m (16 in.).

Primary current, 5000 volts and 265 amp; frequency, 5 per second; power-factor, 0.528; power, 590 kw; calculated amperes in the bath, 52,000; calculated resistance of bath, 0.0002 ohm. Power used, 617 kw-hours per ton of steel. Of the 590 kw, 187 is used to keep the furnace up to heat (to supply radiation and conduction) and 403 for melting and heating purposes; therefore, electrothermal efficiency $\frac{403}{590} = 68$ per cent.

The level of metal in the bath is displaced only 5° from the horizontal, while in the Kjellin furnace it sometimes reaches 24°: the metal in the induction rings rolls actively, which gives

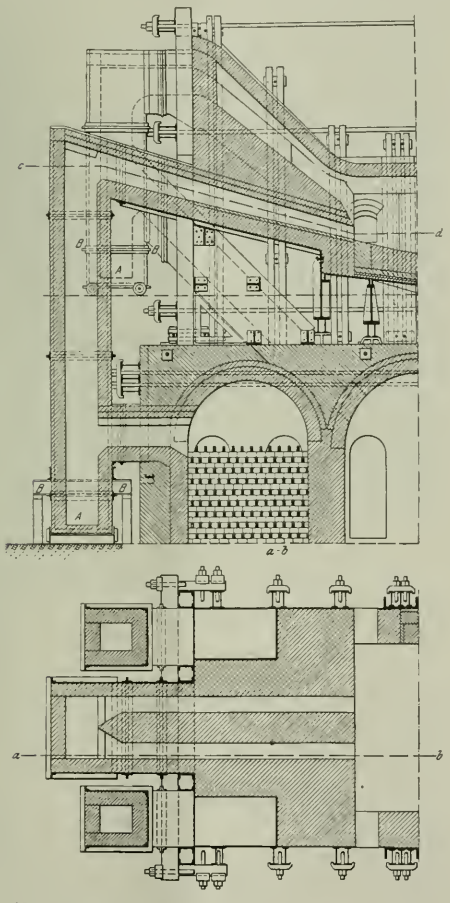


FIG. 22.—ARRANGEMENT OF GAS AND AIR PORTS OF REICHPIETCH.

a good mixture, but makes it difficult to keep the bath covered with slag. In eight weeks this furnace made 180 charges, yielding 1150 tons of fine steel. The campaign was stopped because of injury to the roof; the hearth was uninjured.

Analyses of the steel made show the following properties:

C	Si	Mn	P	S	Tensile strengtb Kg. sq. cm.	Lb. sq. in.	Elonga- tion. %	Contrac- tion. %
0.38	0.23	0.30	0.020	0.024	50.4	72,000	27.5	65
0.77	0.15	0.40	0.013	0.015	83.7	119,000	16.6	46

The Hiorth Furnace (Figs. 30 and 31).—This has been running in Norway, at Helleren on the Jössingfjord, since December, 1909. It has two rings, like the Röchling-Rodenhauser furnace, with a hearth in the middle. The primary windings are divided into four, which are arranged in flat form above and below (embedded in) the two smelting rings. This results in the maximum electrical transformation efficiency and the smallest dispersion losses. The lower embedded windings are cooled by water or air; the upper ones, in the air, cool themselves, and are movable so that they can be lifted up out of the way. By this arrangement, the smelting ring can be enlarged to any extent desired.

The magnet frame is fixed; only the furnace body with the lower windings is mounted for tilting. The furnace holds 5 to 6 tons; the power-factor is given as 0.7 to 0.8, and the

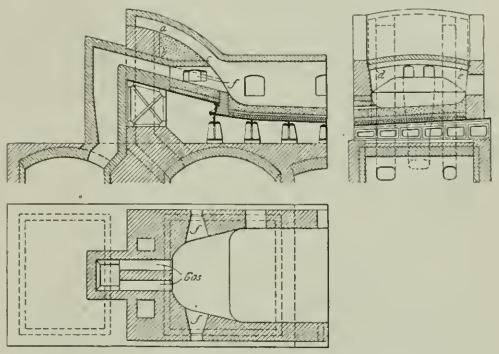


FIG. 23.—ARRANGEMENT OF PORTS BY DIETRICH.

furnace is said to work well, both electrically and metallurgically. The process is only the melting together of pig iron and scrap. Two analyses and tests made of the steel in Stockholm show:

	C.	Si.	Mn.	P.	S.
(1)	0.56	0.14	0.61	0.031	0.01
(2)	1.00	0.16	0.56	0.026	0.015

	Elastic Limit. Kg. per sq. cm.	Lbs. sq. in.	Tensile Strengtb. Kg. per sq. mm.	Lbs. per sq. in.	Elonga- tion. %	C'trac- tion. %
(1)	38.5	55,000	70.5	100,700	12.3	31.7
(2)	46.4	66,300	97.6	139,400	2.3	2.3

Ischewsky Furnace (Fig. 34).—This is neither an arc nor an induction furnace. It consists of a rotating cylinder with radial bricks *a* between which pass iron electrodes *b*. A 1-ton furnace is in operation, making two turns per minute. The current enters by the brushes *A* and leaves by the brushes *B*, which connect with brass springs *c, c*, making connection with *b, b*. The current passes partly through the bath and partly through the material of the roof. If no metal is present, the current passes through the copper strip *e, e* placed beneath the furnace.

The interior of the furnace is first heated by coke, and some soda put in, to make a thin conducting inner layer. The 1-ton furnace operating in the Urals has 36 iron electrodes (1.1 m x 0.3 m); the inside space is 0.75 m (2.5 ft.) diameter and 1.25 m (4 ft.) long; the walls are 0.3 m (1 ft.) thick. It is lined with acid refractory, and the current used is 600 amp at 550 volts, taken in by four brushes.

In conclusion, Professor Neuman says concerning the development of the electrical manufacture of steel:

"The first stage of the development was in the production of high-priced material, making sharp competition with the costly crucible process. Since the electric furnace furnishes much larger quantities of uniform material, it is not dependent on the purity of the material charged, it works cheaper and deoxidizes the metal at least as well as the crucible wall, it enables the adjusting of the carbon, silicon, and manganese much more accurately, and additions can be made such as it is impossible to make in the crucible. Further, high-quality

material can be made by refining scrap and other impure material, which is a field in which the crucible cannot compete.

"For some time, however, the second stage of the development of electric steel has begun, viz.: the use of the electric

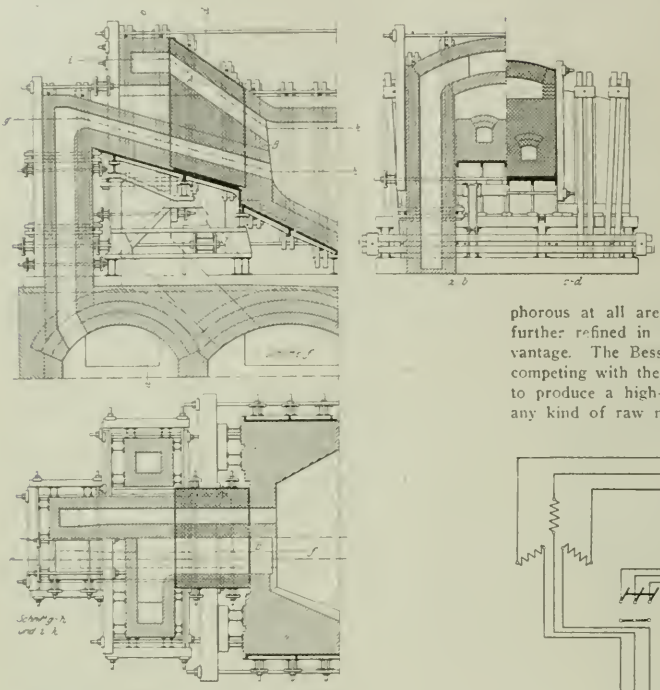


FIG. 24. ARRANGEMENT OF PORTS BY FRIEDRICH

furnace for producing a mean quality of steel to fill in the gap between ordinary steel and crucible quality steel. The electric furnace can be used in combination with the previous steel refining processes to still further refine their product at quite small cost; there is thus produced a nearly sulphur-and-phosphorus-free product, free from segregation, and of great homogeneity, which is particularly suited for high-class rails, tires, axles and springs.

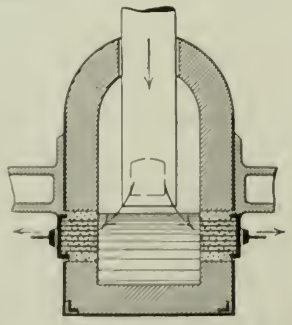


FIG. 25.—LEVOZ STEEL FURNACE.

"According to experience the present used steel refining processes were not used at first because of the cheapness of their product, but because of its superior quality; the electric steel refining will follow the same course. When once a better quality of steel can be furnished for a slightly higher price, this quality is soon used for other and more extensive applications. The electric furnace is well on the way to produce steel in enormous quantities; in the near future it will become a common auxiliary in steel works to supplement the ordinary refining processes, whose entire product will be thus further refined.

"While high-quality steel is mostly made from cold charges, the production of a middle quality in large quantity will require the transfer of fluid metal from converters or open-hearth furnaces, since only in this way is it possible to use the minimum quantity of electrical energy. Since desulphurization in these ordinary furnaces is always incomplete, the advantage of an electrical refining is self-evident. In acid-lined open-hearth or Bessemer furnaces no sulphur or phosphorous at all are eliminated, and the metal from these is further refined in the electric furnace with the greatest advantage. The Bessemer converter may thus find new life in competing with the open-hearth process. It is always possible to produce a high-quality steel in the electric furnace from any kind of raw material, and this will probably make Ger-

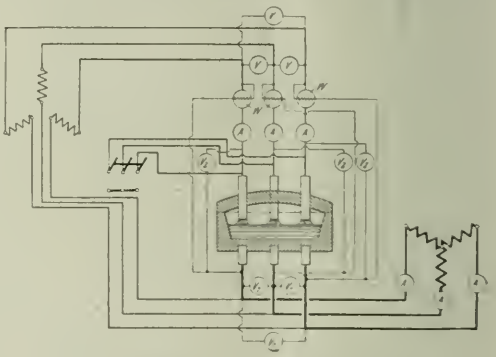


FIG. 26.—CONNECTION OF NATHUSIUS FURNACE.

many independent of importation of special Swedish raw material, a national economy which is very important.

"A large quantity of electric steel is being made up into steel castings. The electric furnace is a convenient melting apparatus in quite small foundries which are not large enough to run a Bessemer or a small open-hearth furnace. It will also be useful in malleable casting foundries. Why could not fluid cast iron be brought directly from the blast furnace of a mixer or a cupola, put into an electric furnace, and there kept hot and receive necessary alloys until its composition was just what was desired, and then cast at just the proper temperature? This keeping hot need only require 50 kw per ton of metal, so that the certainty of obtaining the exact composition required and of casting at the right temperature would in many cases more than repay the cost of the electrical heating.

"A recent unique application of the electric furnace can be here mentioned. The Eicher-Hutten Verein, Le Gallais Metz & Cie., at Dommeldingen, convert pig iron directly into steel in a gas-heated, 20-ton tilting Wellman furnace and three 4-ton Röchling-Rodenhauser induction furnaces (Fig 33). The Wellman furnace acts as an oxidizing mixer. The iron in the Wellman furnace is oxidized down to about 0.26 carbon, 0.082 phosphorus and 0.03 sulphur. This is drawn into the electric furnaces and there refined to the desired quality, using 350-300 kw to a 3.5-ton furnace, in about two hours to steel for

castings and 3 to 3½ hours for best machinery steel. Some analyses are:

	C	Si	Mn	P	S
Pig Iron	3.50	0.60	1.20	1.80	0.12
Mixture in Wellman	1.62	..	0.66	0.46	0.035
Tappings from Wellman	0.26	..	0.30	0.08	0.030
Steel Castings	0.35 to 0.40	0.25	0.50	0.020	0.010
Hard fine steel	0.85	0.23	0.26	0.011	0.009
Chrome steel Cr. 0.87	1.17	0.19	0.16	0.014	0.009

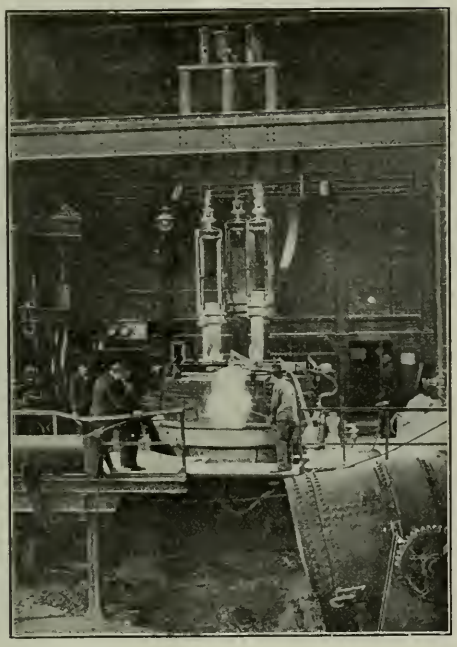


FIG. 27.—NATHUSIUS FURNACE AT FRIEDNHÜTTE.

"The above review gives an idea of the extension of the field of the electric furnace. With such a young branch of industry it is naturally not to be predicted what further possible applications may develop. So much, however, is sure, that the introduction of the electric-steel furnace has opened up a new era of refining to high-quality steel, at a moderate cost."

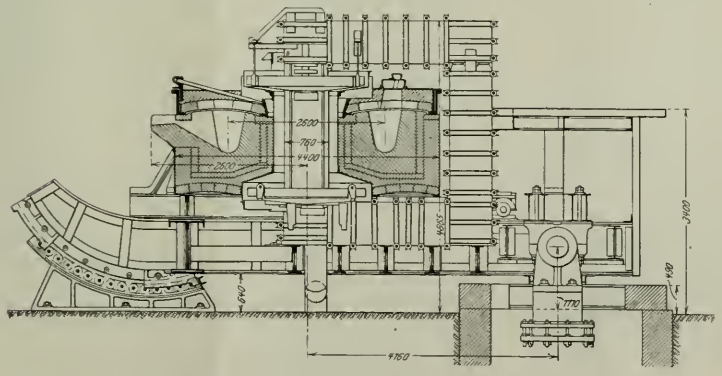


FIG. 28.—LONGITUDINAL SECTION OF FRICK FURNACE.

Acid and Basic Electric Furnaces were discussed by Director O. THALLNER, of Remscheid, from a new standpoint. He tried to find an explanation for the different effects of the acid and basic linings on the steel. All charges melted at a low temperature on basic hearths give steel of poor structure and low physical qualities; but charges which are smelted on a basic hearth with the temperature gradually (not suddenly) raised to a high point, give good steels. Acid hearths, on the contrary, give good steels whether the heat is high or low, and whether the temperature is raised fast or slowly. Director Thallner thinks that the well-known reaction of the carbon in the steel on the silica hearth "supports very strongly the breaking up of the atomic grouping of the steel molecules, and so produces good quality steel." How it does it, or why it should do it, he does not say; and, besides, the assumed atomic groupings are highly hypothetical, resting upon no firmer basis than the following piece of argumentative reasoning: "If changes in the structure of steel are produced by thermal treatment at temperatures below the melting point, one cannot deny similar results from heat treatment above the melting point." One might feel free to deny it until it was proved, or facts adduced to make it fairly probable. The reasons why the basic lining gives poorer results at low temperatures are not given. Altogether, the paper seems merely a piece of metallurgical speculation; the facts adduced are probably correct, and important; the attempted explanations seem far-fetched and improbable.

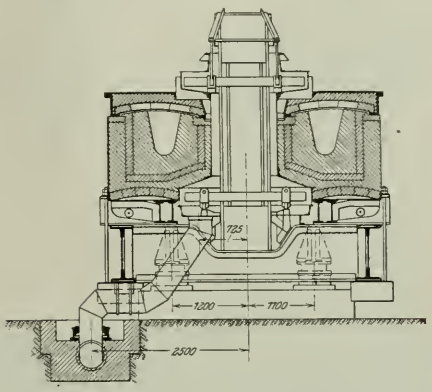


FIG. 29.—VERTICAL SECTION OF FRICK FURNACE.

Current and Voltage in Electric Furnaces, by Dr. Ing. W. CONRAD, of Vienna, records a measurement of the relative amounts of energy developed in the bath of a Héroult furnace and in the arcs above the bath. With the electrodes short-circuited in the metal the resistance of the bath was 0.00027 ohm, there being present 185 liters of melted steel. From this it is calculated that only 0.73 per cent of the energy of the current is expended in overcoming the resistance of the bath. The author attempts to calculate the resistivity of the steel from these data, but the calculation is a miss, because of the irregular shape of the bath. If he were to determine the resistance for a similar shaped bath of a liquid of known resistivity, the calculation could be made.

Notes on Electric Steel, by G. ARNOU, of Paris, deals with

the general quality of the steels made in the Savoy. P. Girod's large works at Ugine has made from June, 1909, to May 1, 1910, 8000 tons of ingots and 850 tons of castings. The Hauts-Fourneaux et Forges d'Allevard have made steel since 1904 in Chaplet furnaces, possessing two furnaces of 3 tons, two of

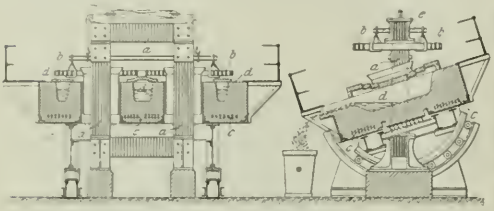


FIG. 30.—SECTIONS OF HIORTH FURNACE.

tons, and a larger furnace under construction. They have old altogether several thousand tons of electric steel.

The absence of gas and oxides and its remarkable homogeneity result practically in two very valuable qualities: a high degree of malleability when hot, and a very strong resistance

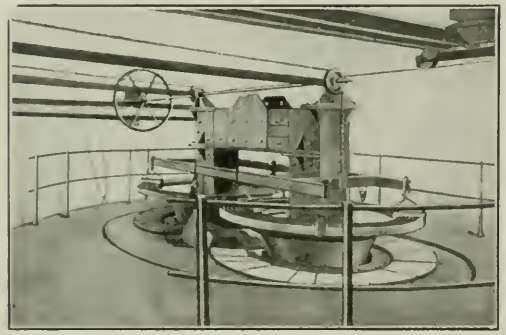


FIG. 31.—HIORTH FURNACE.

to shock. For an equal content of carbon, manganese, tungsten, etc., and given exactly similar treatment, the electric-steel shows 5 per cent more initial magnetism than the non electric.

Mr. Arnou then gives some remarks which explain the differ-

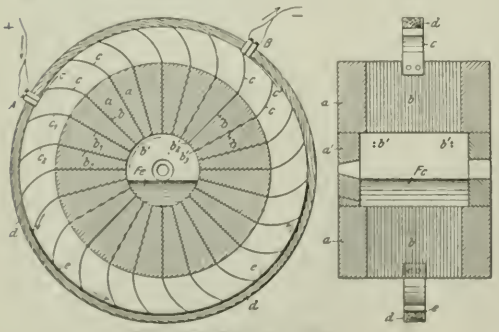


FIG. 32.—ISCHEWSKY FURNACE.

ence between acid and basic practice much more satisfactorily than the paper of Director Thallner. The open-hearth metal always contains dissolved FeO, which on addition of Mn becomes mostly MnO. In contact with a richly basic slag, these oxides are only slowly washed out of the metal; siliceous slags, on the contrary, quickly absorb these dissolved or suspended oxides from the bath. In the electric furnace, with acid lining, these oxides are rapidly removed from the metal, and good steel results; in basic lined furnaces they only come out slowly, by slow heating to a high temperature, and letting stand. In

FIG. 33.—ELECTRIC STEEL FURNACES IN DOMMELDINGEN.

fact, the microscopic observation of these steels shows when the suspended basic oxides (FeO, MnO) have been removed and when they have been left in; the quality of the steel always corresponds.

The electric steel castings, such as those made by the Usine Girod, are of such good quality that they can in many places replace forged steel articles, so that a vast field is here opened up for the electric-steel industry.

On account of limitations of space some further abstracts had to be reserved for publication in later issues.

Sinking Heads on Castings.

At the recent meeting of the Association of Water Engineers, at York, Mr. T. KENNEDY read a paper on the "Effect of a Sinking Head on Large Castings." He described the results obtained in casting some iron columns 32 ft. long and 12 tons in weight, to which sinking heads of 2 ft. 10½ in. long and about 2 tons in weight were given. After detachment, the head was broken, and the fracture showed that the head—which had a considerably greater area than the column—had perfectly maintained the requisite supply as contraction went on in the molten metal of the column. Further, large pockets were formed, and in these there were masses of crystals of pure iron in interlaced groups, while the other materials combined with some of the iron and formed a matrix filling the interstices. The strength of the casting depended on such interlaced groups which were found in identically the same form, but of much smaller size, in the body of the casting where they were completely interlaced and homogeneous as a result of the pressure exercised by the more slowly cooling sinking head. The worst impurities in cast iron as it came from the cupola were globules of slag, free gases and occluded gases. The last named were absorbed physically in the metal, and were, to a very great extent, liberated during the process of solidification. During the pouring more gases were given off from the sand, and for the greater part escaped through the cores and the box joints. The impurities had a lower specific gravity than the iron, and rose to the top if there were a free passage in the mold; if there were not, they were retained, and caused porosity and weakness in the casting. If a long column were cast on a slope instead of vertically the millions of crystals which formed round the mold at right angles to the cooling surfaces greatly hindered the separation of impurities; but if the column were cast horizontally the slag and gases could not get away and would collect in the top side of the casting.

Notes on Chemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

The Failure of Light Engineering Alloys.

Mr. Edward F. Law read a paper before the Faraday Society on June 21 on the failure of light engineering alloys, particularly aluminium alloys, in which he contended that the cause of the disintegration was an allotropic change, and suggested that this occurred in the aluminium, and was not dependent on the presence of other metals, although they facilitated its production. This view was supported by the fact that some samples of "pure" aluminium gave a well-defined break in the cooling curve at 632° C. Other specimens, however, did not exhibit this feature. Aluminium expanded in solidifying and, as Professor Turner had shown in a recent paper read at the Institute of Metals, its behavior during solidification and cooling was similar to that of antimony and zinc.

In opening the discussion the chairman, Dr. F. M. Perken, remarked that other metals besides aluminium broke down in the way described by the author—for instance, chromium, manganese and calcium.

Dr. R. Seligman said some of the questions raised had occurred to him years ago, when he noticed the want of cohesion in aluminium below its melting point. The phenomenon occurred in aluminium of the highest purity; and, therefore, he questioned its being caused by impurities. If an allotropic form of aluminium existed it must be very weak and completely useless. Four years ago he sent some specimens of aluminium to Dr. Harker, who might throw further light on the behavior of this metal.

Dr. J. A. Harker said he had determined the freezing points of the samples of pure aluminium he received from Dr. Seligman, and had looked for the lower point at which the metal became friable. He found a slight indication of a kink in the cooling curve at about 630° C. (the melting point was 657° C.); but otherwise the results were not very satisfactory nor conclusive.

Dr. W. Rosenhain considered that the failure of aluminium alloys was not so great as Mr. Law's opening remarks indicated, or they would not be used in ships in preference to high-tensile steel. With regard to disintegration, he had watched this going on in aluminium-manganese alloys under the microscope. The process resembled miniature volcanic eruptions, fragments of crystals being ejected from the mass. These polymorphic changes were not restricted to aluminium alloys, nor even to metals; they occurred in certain silicates of calcium. He had measured the freezing points of more than 50 specimens of aluminium; but absence of sharpness in the freeze occurred in only one or two instances, and in these the metal must have been contaminated. He had no doubt that allotropic changes in metals were always dependent on the presence of other metals. Dr. Rosenhain was understood to attribute the second break in the curve to a eutectic.

Mr. W. H. Merrett remarked that some years ago he had experimented with a large number of aluminium alloys, and he had invariably found that where the initial freezing points were identical there was afterward a falling off amounting to several degrees. The addition of 5 per cent of bismuth greatly improved the quality of aluminium for casting and turning.

Mr. E. Ristori suggested observations of the effect of putting mechanical work into spontaneously decomposable alloys as soon as they were cast.

Dr. T. M. Lowry observed that most metals when alloyed with iron increase its hardness and retentiveness but decrease its permeability. It was remarkable that in the case of aluminium and silicon the reverse occurred. Could any comparison be traced between the two series of elements with regard to effect on the transition temperatures of iron?

Mr. Law, in reply, said he thought that failure was quite as great as he had stated. [Dr. Seligman signified acquiescence.] Dr. Rosenhain had brought forward several arguments in support of his (the author's) theory. They might not have been intended to support it; but if carefully examined they would be found to do so. He (Mr. Law) had not stated or implied that disintegration did not occur with other metals; aluminium was not unique in that respect. He was of opinion that there was not any eutectic at the second break in the curve. There were three distinct evolutions of heat, and only two could be accounted for with a eutectic when only two metals were present. Fifty freezing points and coolings could always be done without finding the second break when one wanted to obtain it; but when it was not wanted it appeared. The publication of unsatisfactory and inconclusive results of researches might often prove very valuable.

The Electro-Metallurgy of Ferro-Alloys and Steel.

The discussion on M. Paul Girod's paper read before the Faraday Society on June 21 was opened by Dr. F. M. Perkin, who remarked that progress in electric steel refining in England appeared to be slow, but possibly more had been done than was generally known. The cost of power for manufacturing special steels was not so all-important as it was sometimes thought to be.

Dr. W. Rosenhain said that if the best tool steel could be made in the electric furnace from scrap, a revolution in the Sheffield industry must follow. The Sheffield manufacturers fully appreciated the situation, and were experimenting carefully and systematically.

Mr. F. W. Harbord remarked that M. Girod merely claimed advantages in respect of form of the furnace and arrangement of the electrodes. Equally good results could be obtained with any furnace if facilities existed for removal of the slag charged with impurities. The problem of the electric steel furnace was to devise the best mechanical and electrical appliances for the two essentials of refining, viz., oxidation and reduction. Several electric steel furnaces were at work in England. There was not any doubt that steel could be made from scrap in the electric furnace equal to the best crucible steel.

Mr. E. Ristori thought the paper exaggerated the author's claim for priority in respect of electric steel refining, although possibly not in the case of ferro-alloys. He did not consider that the modifications introduced were improvements.

British Aluminium Reconstruction.

At the statutory meeting, on June 29, of the British Aluminium Company (Ltd.), the chairman, Mr. A. W. Tait, said that the company's reconstruction had been most successfully carried out under the terms of the circular of February last. The whole of the shares were issued and allotted, and they had also issued £800,000 of 5 per cent prior lien debentures at the price of £97 per cent, so that the new company would have a total cash sum of £974,435 when the whole of the calls were paid up. The capital of the new company was a total of £2,420,581, made up as follows: £800,000 of 5 per cent debentures, £720,398 of debenture stock, £299,570 of non-cumulative preference shares of £1 each, and £600,613 in ordinary shares. It was very gratifying that only a very small percentage of the shareholders did not take up the interest to which they were entitled in the new company. The company would be able to find the funds for the completion of the Loch Leven Works, and for all that was necessary in regard to other works, and have money left in hand. It was impossible at the statutory meeting of the company to accurately forecast the profits, but the works of the company were in good order, and there was every reason to expect that a considerably larger tonnage of aluminium would be produced during the current year. If the present demand for the metal continued to expand they should be able to get rid of the whole of the output from their various works at a price which would yield a reasonable profit. Of course, no one could ex-

pect that the aluminium industry would recover immediately from the bad time of 1908, and the greater part of 1909, but the market was undoubtedly broadening and prices had already shown some small recovery. There was every reason to look forward with confidence to the future of the reconstructed company.

Market Prices.

JUNE, 1910.

Copper remained steady until the middle of the month, then falling from £56 10 0 to £54 11 0. Since recovered slightly.

Tin has shown a downward tendency, dropping from £149 to £147.

English lead has remained steady at or in the neighborhood of £13.

Iron, Hematite has fallen off from 66/2 to 65/-.

Scotch Pig has fallen steadily from 55/10½ to 55/-.

Cleveland Warrants have dropped from 49/10½ to 49/1; slight recovery later.

	£	s.	d.
Alum, lump, loose, per ton	5	12	6
Antimony, per ton	30	0	0
Borax, best refined crystal, per ton	16	0	0
Sulphate of ammonia, f.o.b. Liverpool, per ton	11	10	0
Sal-ammoniac, lump, first delivered U. K., per ton	42	0	0
Copper sulphate, per ton	17	10	0
Camphor, 1 oz. tablets		1	7
Caustic soda, 48 per cent ordinary, per ton	5	10	0
Bleaching powder, 35 per cent, per ton	4	5	0
Shellac, standard T. N. orange spot, per cwt	3	13	0
Sulphur, recovered, per ton	5	0	0
Carbolic acid, liquid 98/99 per cent, per gal.		1	0
Naphtha solvent, 90 per cent at 160° C., per gal.		2	4
Platinum, per ounce, nominal	6	0	0

Rubber, while touching 9/2 on the 8th, ultimately rose to 10/-, and subsequently again fell to the neighborhood of 9/4.

London, June, 1910.

SYNOPSIS OF METALLURGICAL AND CHEMICAL LITERATURE.

Iron and Steel.

Gases in Commercial Iron.—P. GOERENS, of Aachen, presented a paper on this subject at the International Metallurgical Congress at Düsseldorf. This is reprinted in a recent issue of *Metallurgie*. The first requirement for producing sound steel-castings is the absence of gas in the metal; in steel for rolling and forging the presence of considerable gases is undesirable, since the waste from cropping the ingots will be larger and the blow-holes will roll out to scales on the finished steel. After reviewing the history of the subject, the author describes his tests. Samples of steel were taken at various stages of the process, cast into ingots, and these investigated for dissolved and mechanically-included gases when heated to temperatures of 850° to 1000° C, in a vacuum. The results are contradictory, leaving it very doubtful whether they really represent the composition of the liquid metal. The results must be taken for simply what they are, viz., gases given off by the metal first cast in the air and then reheated to a bright red heat in a vacuum.

Basic Bessemer Metal,
at end of blow

	1	2	3.	4	5.	6.	7.
Volume of gas per 1 Vol. Metal	88	1.95	1.41	1.5	0.50	2.1	2.2
Percentage of CO	20.1	13.8	9.7	6.5	9.5	24.8	9.9
Percentage of CO_2	42.1	45.1	51.8	76.6	64.1	46.5	68.2
Percentage of H^2	21.2	31.8	25.5	11.3	15.0	17.4	20.1
Percentage of N^2	13.6	9.3	13.0	5.6	10.5	11.3	1.8

Basic Bessemer Metal,
after Fe Mn. addition

Volume of gas per Vol. Metal	3.38	1.1	3.3	3.0	4.4	2.7	2.1
Percentage of CO^2	7.75	14.7	8.8	1.5	4.3	5.8	1.1
Percentage of CO	76.3	65.1	60.9	77.2	70.7	73.4	—
Percentage of H^2	6.0	12.2	11.0	4.5	3.5	7.8	22.2
Percentage of N^2	0.95	8.0	10.3	16.8	12.5	13.0	—

Open hearth basic steel

	Before final additions.	After final additions.	From last ingot cast
Volume of gas per 1 Vol. metal	1.7	1.8	3.3
Percentage of CO^2	5.0	2.6	4.2
Percentage of CO	78.5	83.2	81.7
Percentage of H^2	7.4	4.7	10.2
Percentage of N^2	9.1	9.5	3.9

Electric steel ready for casting:

	After removing first slag.	After removing second slag	Finished steel.
Volume of gas per 1 Vol. metal	1.3	1.3	1.9
Percentage of CO^2	7.5	6.4	4.7
Percentage of CO	80.1	84.8	78.3
Percentage of H^2	7.4	4.7	11.7
Percentage of N^2	5.3	4.1	5.3

Crucible Steel:

	Charge 1.	Charge 2.
Volume of gas per 1 Vol. metal	1.9	1.5
Percentage of CO^2	1.3	0.7
Percentage of CO	69.2	71.6
Percentage of H^2	18.6	7.2
Percentage of N^2	10.9	20.5

Miscellaneous Material:

	Steel rail.	Swedish pig-iron.
Volume of gas per 1 Vol. metal	6.1	8.2
Percentage of CO^2	3.7	11.5
Percentage of CO	72.7	70.5
Percentage of H^2	8.3	11.1
Percentage of N^2	15.3	6.9

A careful study of these tests will show practically no connection between the volume of gas and the nature of the steel. It will show, however, that tests made as these were are of no value whatever in forming conclusions as to the nature and amount of the gases in the *fluid* metal, or of the reactions by which they are to be reduced in amount or prevented from spoiling the solid casting. The author, in trying to draw some conclusions, comes to the contradictory result that deoxidizing processes in general enrich the steel in gas, and therefore damage the steel. The truth is, that the tests are incapable of giving any reliable information on the behavior of the fluid steel in casting.

Gold and Silver.

The "De Wilde" Process of Precipitating Gold and Silver from cyanide solutions was devised about 15 years ago by Prof. P. De Wilde of the University of Brussels. In brief, this process consists in acidulating the cyanide solution with sulphuric acid with the addition of cuprous chloride. A yellowish precipitate is formed which consists of an intimate mixture of aurous cyanide and cuprous cyanide. Silver is precipitated as chloride by the addition of sodium chloride with the sulphuric acid. In the April, 1910, *Informes y Memorias* of the Mexican Institute of Mines and Metallurgy, Mr. G WITTEVEEN contributes the results of some experiments performed to determine the practical utility of the scheme as applied to silver solutions. In his description of the process, Prof. De Wilde stated that "the solution (after precipitation) contains all the cyanogen less the small amount combined in the gold-copper precipitate and the very small quantity volatilized as cyanhydric acid." The author of this article believed that this might be true of very weak solutions used for gold ores, but doubted the statement as applied to the strong cyanide solutions used in silver cyanidation.

To test this he took 250 cc. of cyanide solutions of 0.05 per cent to 0.40 per cent KCN acidified each with 0.4 cc. H_2SO_4 to each 0.05 per cent KCN. The solutions were stirred and allowed to stand for two hours, when they were made alkaline with a 6 per cent caustic soda solution. The solutions were then titrated for KCN and the loss found to be almost uniformly to per cent. The same experiment repeated, with neutralization by alkali immediately after acidification, showed the loss to be practically nil. The experiments showed that the loss of cyanide depends upon the time elapsing between acidification and neutralization, and since considerable time must elapse in actual practice the author concludes that the process is impracticable owing to loss of cyanide. Further experiments on working cyanide silver solutions containing 0.25 per cent KCN and 225 grams of silver per ton gave such inconstant results that it was impossible to base on them an estimate of the working costs of the process, but they were found to be prohibitive. The conclusion is that the De Wilde

process offers no advantage in the precipitation of silver bearing cyanide solutions.

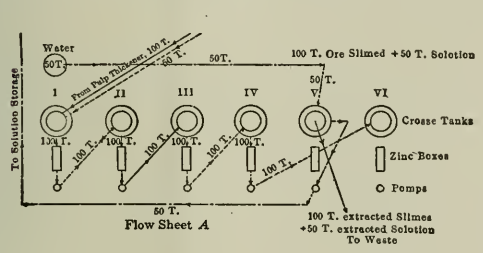
Cyanide Plant of the Consolidated Mercur Co.—This plant is one of the most interesting in the west on account of the developments in cyanidation which have found their beginning here. Cyanidation on a large scale, the treatment of base ores and the vacuum filtration of slime were first successfully tried at this property. The present system is described at length in the June 18, 1910, issue of the *Eng. & Min. Journ.*, by Mr. ROY H. ALLEN.

Two classes of ore are treated; oxidized ore is treated direct, but the base ore receives a preliminary roast to prepare it for cyanidation. The oxidized ore is crushed and passed over screens having ⅝-in. openings, the undersize passing to the fine ore bins. The oversize is crushed again and passed through trommels fitted with ½-in. screens. The oversize from the trommels is cyanided direct, and the undersize is combined with the fine ore. The latter is mixed with cyanide solution through Mercur mixer-classifiers in which the coarse sand is separated from the fine and slime. The coarse sand is subjected to a second similar operation after which it is conveyed to charging bins, whence it is drawn to be mixed with the coarse ore for leaching. The fine and slime are treated in cone classifiers, the spigot product of which is further classified in Dorr classifiers. The sand from these classifiers is sluiced to leaching vats, while the slime is thickened in Dorr Slime thickeners. The treatment of the slime is complete in these thickeners and the pulp is passed from them to the filters. The Mercur filters are of the stationary vacuum type. There are 104 leaves 7 ft. square, a vacuum of about 20 in. of mercury is used, and under this vacuum it takes about 50 minutes to form a cake ⅝ to ¾ in. thick. Fifty minutes are required for the washing and about 20 minutes for the transfer during the discharging process, thus consuming about two hours in a complete cycle.

The base ore is crushed and screened through 1/6-in. stationary screens and trommels and delivered to five straight-line roasters, one Holthoff-Wethey, one Brown and three Jackling, in which it is roasted for about six hours during its passage through the furnaces. Leaching is carried on in 18 rectangular vats, each 25 ft. x 50 ft. x 4 ft. 10 in., with a capacity of 225 tons of sand. Four of these tanks are used for fine sand from the Dorr classifiers, twelve are used for coarse oxidized and roasted ore and coarse sand, and two are held in reserve. The fine sand tanks are filled by sluicing, while the others are filled by hand, the ore being trammed over the tanks, dumped and spread. Cyanide solution is run in from the bottom for about eight hours, allowed to stand for four hours, when leaching is commenced. Strong solution is applied for 24 hours and weak solution for 48 hours. Washing requires 12 hours. Zinc dust is used as a precipitant, and 10 Johnson filter presses are required to handle the solutions. For the year ended June 30, 1909, the following data appear: Mill ore assayed $3.58 per ton; tailing $0.88 per ton; extraction 75.4 per cent; consumption of cyanide, 0.66 lb.; lime, 5.4 lb.; caustic soda, 0.05 lb.; zinc, 0.45 lb. per ton of ore treated. The cost of milling 282,269 tons of ore is given as $1.09 per ton. The cost of roasting, $1.05 per ton of ore roasted.

Simplifying the Cyanide Process.—In our issue for February, 1910 (Vol. VIII, No. 2, p. 72) there appeared a description of Mr. A. F. Crosse's system of slime treatment. The main feature of the process consists of the elimination of the filter press, and precipitation of a clear cyanide solution obtained by decantation. The suggestion has been taken up by Mr. BRUNO MIERISCH, who describes in the *Eng. & Min. Journ.* for June 25, 1910, certain proposed simplifications of the cyanide process. The proposal provides a continuous system of treatment in which the handling of the ore is reduced to a minimum.

In flow-sheet *A* (Fig. 1) there are as many "Crosse" tanks as the number of days required for the treatment of the ore, plus three more of identical design. Thus if the treatment requires three days, the number of tanks is six: three for the treatment proper; one which is being filled with ore; one being emptied and the sixth receiving the overflowing solution. Assuming that tanks Nos. 2, 3, 4 and 5 are filled with pulp, No. 1 is filled with the necessary cyanide solution and receives its charge from the mill, and No. 6 is empty. Air is turned on in all tanks except No. 6. Thickened pulp flows into the first tank and displaces a corresponding volume of solution which flows through No. 1 zinc box and into tank No. 2. Here a similar overflow is effected after mixture with the pulp, and so on to No. 4, from which zinc box the solution is pumped into No. 6. The ore in tank No. 5, having given up its gold and silver, is washed with an amount of water lost by evaporation and by that retained in the discharged slime. The resulting dilute solution is returned to the crushing and grinding apparatus. The cycle of operations advances from tank to tank each 24 hours and goes on continuously. In flow-sheet B zinc box precipitation is eliminated and amalgamating pans substituted. Only five tanks are required. As the solution flows into tank No. 5 the thickened pulp is taken out and directed into the amalgamating pans which contain mercury, zinc amalgam and copper. After about a two-hour treatment the gold and silver are precipitated into the amalgam and recovered as usual.



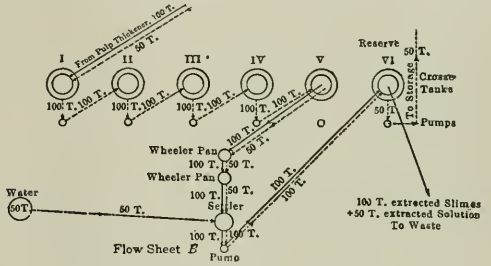


FIG. 1.—SIMPLIFYING CYANIDE PROCESS.

Assuming that tanks Nos. 2, 3, 4 and 5 are filled with pulp, No. 1 is filled with the necessary cyanide solution and receives its charge from the mill, and No. 6 is empty. Air is turned on in all tanks except No. 6. Thickened pulp flows into the first tank and displaces a corresponding volume of solution which flows through No. 1 zinc box and into tank No. 2. Here a similar overflow is effected after mixture with the pulp, and so on to No. 4, from which zinc box the solution is pumped into No. 6. The ore in tank No. 5, having given up its gold and silver, is washed with an amount of water lost by evaporation and by that retained in the discharged slime. The resulting dilute solution is returned to the crushing and grinding apparatus. The cycle of operations advances from tank to tank each 24 hours and goes on continuously. In flow-sheet B zinc box precipitation is eliminated and amalgamating pans substituted. Only five tanks are required. As the solution flows into tank No. 5 the thickened pulp is taken out and directed into the amalgamating pans which contain mercury, zinc amalgam and copper. After about a two-hour treatment the gold and silver are precipitated into the amalgam and recovered as usual.

Neither of the systems has been tried in practice, but the author believes that they offer theoretical advantages which merit trial.

Aluminium.

The Aluminium Industry in France is one of growing importance. The principal deposits of bauxite occur in the southern part of France and comprise two varieties, white and red. The former is low in iron, but high in silica, while the latter is higher in iron and low in silica. Between these two varieties are intermediate grades in which the impurities of iron and silica replace each other. In the June 11, 1910, issue of the *Eng. & Min. Journ.*, TONY CALLOT gives some details of the French aluminium industry. The percentage of Al_2O_3 varies from 55 to 70 per cent, while an unusually pure variety yields

78 to 80 per cent Al_2O_3, with almost no iron, 2 to 3 per cent silica and 2 to 3 per cent TiO_2. The best ore is found in the Department du Var, where it is sold under a formal guarantee to contain 60 per cent Al_2O_3 and a maximum of 3 per cent silica.

Two processes are used in France to produce alumina from bauxite. The Bayer process is used at Gardanne, where the ore is attacked under pressure with caustic soda solution, producing sodium aluminate which is separated by filtration. Alumina precipitates from this spontaneously, or the precipitation may be aided by the addition of some prepared alumina. The precipitated alumina is washed and calcined, when it is ready for reduction to aluminium. In the other process which is used at Calypso, the pulverized bauxite is charged in ordinary soda furnaces with soda. The sodium aluminate thus obtained is dissolved in water, filtered from the iron and then treated with CO_2 to precipitate the alumina. The reduction of alumina to aluminium is accomplished by the well-known electrolytic process. A large quantity of electric horsepower is required. About 100,000 hp is used annually.

Copper.

Basic Lined Converters.—The first successful use of basic lining for converting was by Knudsen, who developed his system of pyritic converter smelting at Sulitelma, Norway. Other more or less successful attempts to use basic lining have been made by other metallurgists in this country for twenty years. Redick R. Moore describes the present practice in the United States in the *Eng. & Min. Journ.* for June 25, 1910. The data given are from operations at Garfield, Utah and Perth Amboy, N. J. The steel converter shell is of ¾ in. steel and is 23 ft. long and 10 ft. inside diameter. There are 32 1¼-in. cast-iron tuyeres. A circular throat 40 in. in diameter is placed at one end; other openings are provided for removal of slag and metal, and for the examination of the tuyeres, and for the introduction of an oil burner. The slag opening and the opening for inspection of tuyeres are at the side opposite the latter, and the opening for the oil burner is at the end opposite the throat. These openings are kept luted with clay when not in use. The lining is of 9-inch magnesite brick except at the tuyeres, where 18-in. brick are used. The oval opening inside the lining is about 6 ft. 3 in. high by 7 ft. 3 in. wide by 19 ft. 9 in. long. The brick are set in magnesite powder, except at the tuyeres, where a mixture of magnesia and linseed oil is used.

When seasoned, an initial charge of 30 to 40 tons of matte is poured in through the throat in the usual manner. About 6000 pounds of silicious ore are added and the blast turned on, the pressure being about 5 lb. for a few minutes, being later increased to twice that amount. Blowing is continued for 30 to 45 minutes until the silica is fluxed, when the slag is poured. Another ladle of matte is added and another charge if silicious ore. This is kept up until the converter is filled to within 2 to 4 inches of the tuyeres with white metal, when the charge is blown to copper. The appearance of the flame is not relied upon when blowing leady or arsenical mattes, but test samples are taken from time to time. Slags are usually lower in silica and copper than those from acid lined converters; they are generally made to carry about 27 per cent silica. With leady mattes this can be reduced to 23 per cent.

Foaming is caused in basic converters when the charge is too cold and when an excess of silica has been added. Leady matte is also prone to foam badly, but foaming can be corrected by the quick addition of a ladle of hot matte and increase of the blast. The basic converter also shows greater radiation and conduction heat losses than the acid converter, necessitating working with larger charges or lower grade mattes and higher blast volumes. Also on account of the lower temperature it is necessary to keep men constantly punching tuyeres.

The advantages of the basic lined converter are stated by the author as follows: Cheaper construction, less building space and lighter crane, large vessels may be used as there is no handling of the same, no clay used for binder in relining, less slag produced for further smelting to recover copper, and lower grade mattes can be converted at a less cost per ton of matte and silicious ore than the same concentration can be effected in the blast furnace. The disadvantages are: Loss of fines due to blowing out of the throat, lost time in relining and repairing, extra careful manipulation required to avoid burning out lining and tuyeres and the formation of accretions, and extra tuyere punching necessary. These advantages and disadvantages will determine the use of the basic converter in different localities.

Electrolytic Refining.—An extensive paper on electrolytic copper refining was presented before the Australasian Institute of Mining Engineers, and published in their *Proceedings*, by G. H. Blakemore. The article describes in detail the procedure at Lithgow, New South Wales, where the author was engaged for some years. Efficient insulation, cleanliness and temperature of the electrolyte are discussed as the three secrets of successful electrolytic refining. The making of starting sheets, the parting of the Dore bullion, filtration and purification of electrolyte, effect of slime suspended in electrolyte, and the methods of assay used are all discussed in detail. The paper is so extensive as to require original reading, as an abstract could not do it justice.

Lead and Zinc.

The Daly Judge Mill at Park City, Utah, concentrates an argentiferous lead ore carrying some pyrite and blende. As described in the July 2, 1910, issue of the *Min. Wld.*, by Mr. L. A. Palmer, the system of concentration is briefly as follows: Wet crushing is practiced, the product of crusher and rolls going to a line of screens of 3, 4 and 6-mesh. The oversize from each screen goes to a 3-compartment Hartz jig which delivers both gate and hutch products in the shape of a finished lead concentrate; middling is discharged over the tailboard. The undersize from the last screen goes to a classifier which makes two feeds for fine jigs making finished lead concentrate through the hutch only. The middling from the two coarser jigs is dewatered, screened and treated in Huntington mills, the product from which is combined with the middling from the fine jigs. These are elevated to a dewatering tank, the overflow from which goes to the main settling tank, the settlings being screened and jigged. From the main settling tank, which is composed of 11 compartments overflowing from one to the other, nine Wilfley tables are fed. The tables make five products, lead concentrate, lead middling, zinc middling, tailing and slime. The lead middling is reconcentrated on Wilfley tables where a lead concentrate is recovered. Another set of Wilfley tables makes a lead-iron concentrate and a zinc middling. Slime is collected in a settling tank and concentrated on a Wilfley table. The tailings of all the Wilfleys except those treating the middling from the first group of nine, are screened on a 10-mesh trommel, the oversize being rejected and the undersize treated on three Wilfley tables. The zinc middlings are retreated to remove as much of the lead and iron as possible before shipment. Concentration is at the rate of 3.49 tons of crude ore into 1 of shipping product. Extraction is about 90 per cent on lead and gold, and 75 to 80 per cent on zinc and silver. Milling costs are about 95 cents per ton.

The Potter Process, in common with the other flotation processes has suffered from misapprehension as to the exact procedure. One point not clearly understood concerns the use of oil. In the *Mining Magazine* (London) for June, 1910, Mr. W. F. Simpson communicates some interesting facts regarding the operation of the Potter process in Australia during his régime as manager of the Zinc Corporation at Broken Hill. The author states that no oil was used in the Potter process, and is almost certain that Potter never anticipated the use of oil. Neither is it true that oil from exhaust steam probably

got into the solutions, for neither oil nor exhaust steam ever found even a tentative place in the process.

The formation of gelatinous silica by chemical reaction between the quartz and fluorspar in the ore, in the presence of the hot sulphuric acid used, is assigned by Mr. Simpson as one of the elements of success of the Potter process on these ores. The colloid-silica doubtless aided in clotting the sulphides into masses which became buoyant under the action of the heat and the generation of carbonic acid which was continuous. It was proved that if the gelatinous silica were not formed, the process would be so impaired as to render it impracticable.

Another point mentioned by Mr. Simpson, is his discovery that absolute aeration of the material to be treated was necessary before its immersion in the hot acid solution. Prior to aeration the mass has been mixed with the acid solution before the mill water had been drained from the particles of ore and they were coated with a film of mill water. On aerating the mass, the improvement was most marked. In a subsequent 1000-ton test, the actual extraction for the entire time was over 81 per cent of the zinc, and at times the extraction ran as high as 92.5 per cent. Mr. Simpson gives the requirements for the Potter process as follows: That the sulphide particles have clean, air-filmed surfaces, and be not over 1 mm. in diameter; that there must be present, though not naturally added, some gelatinous or oily material for clotting the particles together; and that there must be some means of bringing these clotted masses to the surface, either by the generation of gas or by the buoyant effect of heat or vacuum.

Chemical Engineering.

Protal and Protal-Bakelite.—A paper by Dr. F. G. Wiechmann before the Am. Inst. Chem. Eng. deals with "a new product for use in the arts." The base of this new material is vegetable-albumin, under which generic term vegetable ivory, the vegetable caseins, glutens, hemi-celluloses, reserve cellulose, horny albumins, etc., are included. The vegetable albumin, from whatever source derived, is treated with one or more substances, which convert it into a new substance, a plastic eminently well adapted for use in the arts and industries. To this new plastic, the name protal has been given. Any and all materials commonly used in the rubber industry may be incorporated with protal. About one hundred different protal compounds have been produced, and, of course, the properties of these compounds vary with the ingredients employed. Protal can be molded, pressed or otherwise formed, into any desired shape. It is odorless, resilient, and can be cut, sawed, filed, polished, tapped and countersunk, like hard rubber. It can be colored by dyes, and all pigments can be incorporated with it. It is non-explosive and is very resistant to the influence of heat and electricity. Among the great number of protal compounds which have been made, there are some which contain rubber, rubber substitutes, shellac, rosins, asbestos, etc. Some of these products remain plastic and moldable for a long time and possess the remarkable quality of hardening on immersion in water. Compounds of protal with rubber, rubber fluxes and some of the so-called rubber substitutes, exhibit a wide range in their properties. They can be made hard, semi-hard or soft, which of these qualities they are to exhibit being determined by the choice of loading materials and by the conditions of heat and pressure governing their production.

Among the most important compounds of protal is protal-bakelite, bakelite being that most interesting and valuable new product of Dr. L. H. Baekeland, which is, as is well known, a condensation product of phenol and formaldehyde. Protal-bakelite possesses many valuable qualities. It exhibits great resistance to nearly all chemical solvents. It is an excellent electric insulator, is capable of taking a high degree of polish, can be produced in almost every color, and is well adapted to the many purposes and uses for which hard rubber and hard rubber compounds are, at present, almost exclusively employed. It possesses the great advantage over hard rubber of not being subject to oxidation, of not softening on the application of heat and of not being attacked by oils and bodies of a similar nature. Any and all materials, organic as well as inorganic, may be incorporated with protal-bakelite, thus giving rise to a great number of compounds which possess very different qualities and properties and which are adapted to a great variety of uses. Protal-bakelite compounds have, as before said, great dielectric strength, ranging from about ten thousand volts to about twenty-six thousand volts per millimeter. It would be impossible to specify all of the uses to which this new plastic may be put. It need only be borne in mind that this is a plastic which can be fashioned into any shape, which can be molded and pressed, which is capable of taking a high polish, which is not affected by water—cold or boiling—which is resistant to practically all chemical solvents, which can be tooled and machined with ease and which is produced in both flexible and rigid form. In cost, this material compares very favorably with rubber and every day witnesses its introduction into new fields of industry. The manufacture of protal and of protal-bakelite is in the hands of a New York concern, the Protal Company, the works being located at Bridgeport, Conn., and at Yonkers, N. Y. (From advance sheets.)

Problems in Chemical Industry.—A paper on this subject was presented by Mr. John T. Baker, of Phillipsburgh, N. J., before the Am. Inst. Chem. Eng. The number and complexity of the factors involved in chemical operations are so great that many operations are still carried on under the rule of thumb guidance and have not been reduced to a science. On the other hand the trained scientific man is very prone to believe that the matter with which he deals will follow the laws which he has learned, and for this reason he often overlooks valuable facts which the untrained observer sees. The untrained observer ignores laws and systems; tries any suggestion that comes along, and therefore loses much valuable time and labor. The investigator who is successful follows a mean between these paths. A number of practical illustrations of these principles were given. (From advance sheets.)

Electrochemistry.

Ozone.—A paper on the manufacture and industrial applications of ozone was presented by Dr. Oscar Linder, of the Western Electric Company, of Chicago. (By a misprint the author's name was given in our last issue, p. 387, as Dr. Snider instead of Dr. Linder. This is herewith corrected.) The properties and methods of formation of ozone were given. The amount present in pure air was given as about 1 part in 1,000,000. The commercial methods of manufacture of ozone are almost exclusively electrical, although ozone is formed by ultra-violet light and heat. The successful operation of the electric ozonizers depends upon the rise of a high voltage (10,000 to 40,000 volts) and dry air maintaining a low temperature. Concentrations of ozone as high as 30 grams of ozone per cubic meter of air corresponding to 2 per cent by volume have been obtained. Yields of 105 to 250 grams of ozone per kilowatt hour have been obtained.

Ozone has been used industrially for a great many purposes. The purification of the air of theatres, schools, restaurants and public halls is quite common in Europe. Ozonizers made especially for ventilating purposes are now on the market and are made either of portable or stationary type. Large installations for water purification are in successful operation. Ozone is well adapted for sterilizing and disinfecting purposes. It is being tested in the quarantine station in New York and at the Pittsburg Homeopathic Hospital. It is believed that ozone will ultimately supplant formaldehyde in disinfecting rooms and buildings. It has also been successful for the preservation of milk, cream, butter, eggs, fruits, meat, etc. Ozone is also an excellent bleaching agent. (From advance sheets.)

RECENT METALLURGICAL AND CHEMICAL PATENTS

Concentration

A New Concentrator designed along entirely new lines is the invention of Mr. Lucien I. Blake, of Boston, Mass., but at present residing in Denver, Col. Briefly stated, the device consists of a flexible endless belt traveling around a system of inclined rollers, with a sufficient amount of sag between the supporting rollers to form a trough, which is the concentrating

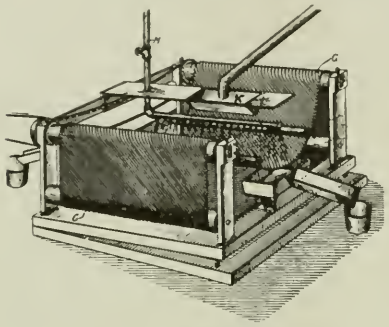


FIG. 1.—CONCENTRATOR.

area. The working surface of the belt is corrugated or riffled and so formed that the riffles run obliquely to the direction of travel of the belt, and in such a direction that the particles of concentrate are carried upward in the trough, while the tailing or reject is allowed to flow to the lower end of the trough. The details of construction may be varied considerably, but the form preferably used by the inventor is that indicated diagrammatically in the accompanying illustration. The flexibile belt *G* may be made of corrugated rubber. Ore in the form of pulp is delivered into a suitable feeding apparatus, indicated by *K*, from which it is fed onto the surface of the belt against the motion of the same. Wash water is introduced through a pipe *M* so located that small jets of water issuing from it will act upon the layer of pulp and aid in the separation of concentrate and tailing. The heavy particles in the pulp will sink into the riffles and travel up the inclined trough, while the lighter particles of gangue will be raised above the riffled surface and be washed off by the wash water. The table thus makes but two products, concentrate and tailing. (964,083, July 12, 1910.)

Smelting.

Blast Furnace Feeding Device.—In the usual operation of a blast furnace such as is used for lead and copper smelting, the gases tend to find paths of least resistance in rising through the column of ore and flux. These paths are usually found along the walls of the shaft. One method of overcoming this tendency to form channels through the smelting mass is to so distribute the charge that the fine matter will lie along the walls and the coarser material in the center of the shaft. This would provide for more even permeation by the gases. Arthur S. Dwight, of San Luis Potosi, Mexico, has patented a device for effecting this mode of feeding mechanically. The device consists of aprons or partitions placed inside the furnace charging doors, and extending both above and below the plane of the feed floor. These vertical aprons are supported in the end walls of the furnace and run parallel with the side walls. In Fig. 2, which is a diagrammatic sectional end view of a furnace, B' is the feed floor, G' the feeding device having vertical side walls g^4 and stays g^7. C' represents the furnace doorway, with sliding door E'. When the ore is charged, the larger pieces of the charge tend to fall toward the center of the shaft upon passing below the apron, while the fine material remains near the side walls. The feeding device presents no obstacle to the usual furnace operations, such as breaking up bridges and crusts, and if the furnace be kept full the amount of cold air which can enter is reduced to a minimum and the size of the flue required to carry away the gases can be reduced as compared with the usual requirements. (959,484, May 31, 1910.)

Reverberatory Furnace.—Various schemes have been proposed to utilize the waste heat from reverberatory furnaces. One of the most successful proposals is the generation of steam in boilers placed at the front of the furnace. The proposal of Frank A. Burns, of Anaconda, Mont., is that reverberatory furnaces be placed in series so that the waste heat from one will aid in effecting the smelting process in the next. He claims a reduced consumption of fuel in proportion to the number of furnaces in the series. The usual amount of fuel would be required in the first furnace in the series, but each succeeding furnace in the battery would require less fuel to be consumed in its fire box. (960,499, June 7, 1910.)

A Regenerative Reverberatory Furnace is the subject of a patent of F. A. Leas, of Oakland, Cal. The idea of the inventor is to provide a reverberatory furnace which can be fired alternately at one end and the other. Oil is the fuel preferred by the inventor of this furnace. From each end of the furnace settling flues extend in any manner desired, but both flues lead to a common short flue, which in turn connects with the stack. In the short flue is a damper of special form, which can be operated to admit air for combustion through the flue leading to one end of the furnace while the products of combustion are passing from the other end out through the stack. When the flue conducting the products of combustion has been heated the operation is reversed by changing the damper, so that the air for combustion now flows through the heated flue, while the flue from the opposite end of the furnace now conducts the products of combustion. Oil burners placed at each end of the furnace are in use alternately, the one in use at any time receiving the preheated air. The furnace thus combines the dust-saving features of the ordinary reverberatory, with the heat economy features of the open hearth steel furnace. At the same time the checker work is eliminated, which is impracticable in a regenerative reverberatory furnace on account of the large amount of dust formed. (902,644, June 28, 1910.)

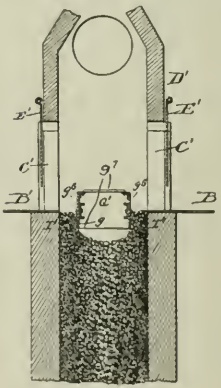


FIG. 2.—BLAST-FURNACE FEEDING DEVICE.

Gold and Silver.

Filtration.—The mechanical treatment of slime pulp is the subject of a number of recent patents, some of which have proved successful, while others are not known to be in use, although their mode of operation is interesting. Erwin Burt, of El Oro, Mexico, has been granted patent on an improved method of filtering slime, in which he prefers to use his apparatus patented some months ago. (931, 267, Aug. 17, 1909). It comprises a rotatable shell in which is placed a filtering medium, on which the filter cake is to be deposited as the shell rotates. The chief difficulty in operating devices of this type has been the lack of coherence in the filter cake during the rotation of the shell, parts of the cake falling off. The object of the present invention is to provide a firm cake. Slime is run into the shell until it is half full. Air or fluid pressure is then gradually generated in the filter shell as filtration pro-

ceeds, until a final pressure of 40 to 50 pounds exists. As the shell rotates at a rate of about 15 revolutions per minute, the liquid portion of the slime is forced out through the filter and a cake forms under pressure sufficient to hold it intact during the necessary subsequent washing. (961,713, June 14, 1910.)

Quite different in form is the apparatus designed by Clyde H. Jay, of Salt Lake City, Utah. The operation will be understood by reference to Fig. 3. The description of the process appeared prior to the issue of patent papers in the *Western Chemist & Met.*, May, 1909. The inventor's idea is to provide a continuous dewatering, agitating, and filtering process for handling slime pulp. Thickened pulp is directed into the cone-bottomed tank containing filter leaves as shown. The vacuum pump is started and by the time the tank is filled with pulp it has been practically dewatered, and ready to receive the cyanide solution with which it is to be treated. Agitation is accomplished by a Pohle air-lift pump placed vertically in the center of the tank. As soon as it is considered advisable, a vacuum is created in the vacuum tank and filtration of metal-bearing solution commences, which is continued until the solution has been extracted. Wash water or weak solution is then applied and filtered off. The discharge of the pulp is effected by directing a current of water into the filter leaves, causing the cake to drop off, when it is sluiced out at the bottom. The inventor claims low cost of installation for the work accomplished; minimum amount of labor; small amount of equipment for tonnage treated, and maximum extraction in minimum time. (963,710, July 5, 1910.)

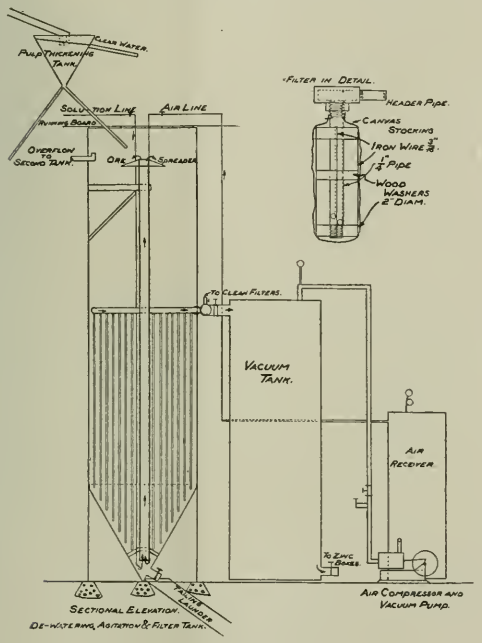


FIG. 3.—JAY CYANIDE PROCESS.

A novel scheme in filtration schemes is that of Clarence M. Chamberlain, of Denver, Colorado, although it does not appear to have advantages over the more common types of apparatus. He proposes to arrange open filter trays horizontally on a steel framework, each filter tray being connected with vacuum apparatus so that slime may be filtered under vacuum. By means of a time wheel fitted with cams, it is arranged to have pulp, weak solution and wash water successively directed into the trays, so that the process of filtration and washing will be continuous. The discharge of the cake is effected by means of scrapers which are so arranged that they may be operated over the surface of the filter trays after the filtration is complete. (959,461, May 31, 1910.)

A better known form of filter, on which patents have been granted to Wilbur A. Hendryx, formerly of Denver, Colo., but now of New York City, may be used as a vacuum filter or as a decanter as conditions indicate. A round tank with a converging bottom is provided with filter leaves radially arranged around a central well. The filters occupy the upper portion of the tank only, the conical portion being unoccupied. At the bottom of the tank there is horizontal discharge pipe in which a screw propeller may revolve to discharge the thickened or filtered pulp. The flow of pulp is downward through the central well, at the bottom of which is a spreading baffle. When the pulp rises to the height of the filter leaves it may be filtered either by decantation, if clear enough, or by vacuum through the filters. The thickened pulp in the bottom can be discharged through the pipe containing the screw, at the end of which is a gate which can be regulated to allow only a small amount of dewatered pulp to pass. If the filter leaves are used, the cake can be discharged from them by a current of water directed through the inside of the filter, and the pulp discharged through the bottom as before indicated. (960,546 and 960,547, June 7, 1910.)

Precipitation on Aluminium.—The use of this metal to replace zinc as a precipitant for gold and silver in cyanide solutions has been suggested from time to time by several writers, and is used in laboratory schemes for analysis of cyanide solutions. Stafford F. Kirkpatrick, of Kingston, Ontario, Canada, has patented the process for commercial application in an apparatus specially devised for the purpose. The most efficient form in which aluminium could be used for this purpose is as a powder, but powdered aluminium does not easily become wet by water and does not mix with solutions. In order to overcome this difficulty the inventor makes use of an agitating device to engulf the metal in the solution and thus cause it to react with the precious metal cyanide solutions. If the precipitation is carried out in a round tank with a conical bottom, a suitable agitator can be made by attaching propellor blades onto a vertical shaft revolving in the center of the tank. This will produce a vortex into which the aluminium will be drawn and thoroughly mixed with the solution. Sodium or potassium hydroxide is added to the solution, and the inventor gives the following equation as representing the reaction which takes place:

$$6NaAG(CN)_2 + 6NaOH + 2Al = 6Ag + 6NaCN + Al_2O_3 + 3H_2O.$$ (959,757, May 31, 1910.)

Copper.

Smelting Pyritic Ores.—An improvement in the process of smelting copper sulphide ores and other sulphides has been patented by John Parke Channing and Frederick J. Falding, of New York City. They propose to so regulate the supply of air for smelting that practically all of the iron and sulphur available for pyritic smelting shall be oxidized, and that the gases resulting shall be available for the manufacture of sulphuric acid. It is further claimed that a higher grade matte will result from these operations. In the case of pyritic smelting of ores not containing copper, the resulting iron slag will be suitable for the production of pig iron in the blast furnace. The regulation of the amount of air used is to be effected by means of valves near the tuyeres. The top of the furnace is to be so arranged that no excess of air will be admitted to disturb the equilibrium established. If it is found necessary to dilute the gases in order to make them better available for the manufacture of sulphuric acid, the inventors propose a second set of tuyeres above the charge, through which the necessary amount of diluent can be introduced. In regard to the use of carbonaceous fuel, it will be seen that it would be de-

sirable to reduce the quantity as much as possible. The inventors claim that experiments have shown that ½ per cent carbon to each 4 per cent available sulphur in the charge is a proportion which gives gases having sufficient oxygen therein to convert sulphur dioxide into sulphur trioxide. It is further stated that practice has determined that the amount of free oxygen should be considerably more than the amount theoretically required to convert sulphur dioxide into sulphur trioxide. Assuming that the gases contain 6 per cent sulphur dioxide, they would require, theoretically, 3 per cent of free oxygen to convert them into sulphur trioxide. It is found, however, that 7 to 8 per cent free oxygen should be present. The richness of the furnace gases in sulphur dioxide should range between 5 per cent and 7 per cent sulphur dioxide. (963,020, June 28, 1910.)

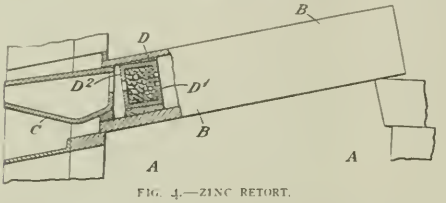


FIG. 4.—ZINC RETORT.

Zinc.

Improved Extraction.—In the ordinary distillation of zinc from its ores it has been found that some sort of mechanical filtration of the metallic fume as it passes from the retort to the condenser improves the quality of the condensed zinc by removing certain impurities, notably lead. Fig. 4 shows the apparatus designed by CHARLES S. BRAND, of Troon, Scotland, for this purpose. *A* indicates the furnace heating area, *B* a retort, and *C* a condenser. The filter container *D* is of fire clay and is filled with material refractory to the temperature employed in the process. A suitable material has been found in pieces of broken clay retorts, silica brick, etc. The container is about 6 in. in length and of a cross-section similar to that of the retort in which it sets. The filtering material is of about 1 cu. in. in size. The ends of the container are perforated to permit the passage of the metallic fume. The apparatus is so arranged that at the completion of the distillation of a charge the container may be withdrawn and the condensed lead poured from it. (963,416, July 5, 1910.)

The Continuous Volatilization of Zinc from its ores is the subject of a patent of HARRY H. HUGHES, of Springfield, Mo. The construction of his furnace provides for the continuous introduction of ore without access of air, for the removal of the molten zinc and the residue, and for the utilization of the carbon monoxide gas generated in the reduction of the ore. Reference to Fig. 5 will make the process plain. The charge of ore to be reduced is fed in at the hoppers 26 and 27, where it is transferred into the retort 3 by means of a screw conveyor. The retort is encased in a revolving combustion chamber, in which the retort is suitably supported as shown. The retort connects with a condensing extension 8, which further communicates with the portion 11, from which the molten zinc is discharged through 33, and the residue through gates 38 and 40. The combustion chamber is heated by oil burners 32, and provision is further made for the combustion of the carbon monoxide formed in the reduction of the ore, by conducting it into the combustion chamber by means of the pipe 35. The inventor claims that every particle of ore and carbon is brought in contact with the heat quickly and at the proper temperature to effect the production of the maximum quantity of zinc; and further that the condensing apparatus is under such control that the minimum quantity of blue powder will be formed. (13,118, Reissued May 31, 1910.)

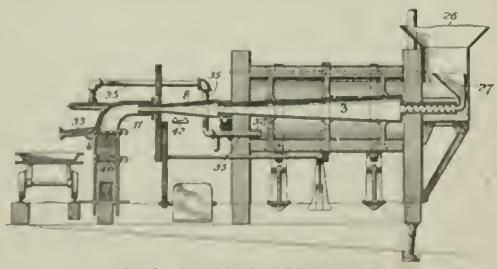


FIG. 5.—CONTINUOUS ZINC PROCESS.

Smelting Lead-Zinc Ores.—For the purpose of cleaning mixed zinc and lead ores containing gold and silver prior to their treatment in the blast furnace, EDUARD DEDOLPH, of Kaslo, British Columbia, proposes to volatilize the zinc and lead as oxides and sulphates, leaving a mixture suitable for the blast furnace. The operation is carried out in a revolving furnace fed with a mixture of the ore and flux which will make a suitable smelting product after the zinc and lead have been volatilized. The fuel may be gas or oil directed into the discharge end of the drum, where a jet of air is also introduced. Two drums can be used in conjunction, the first operating at a temperature of about 800° Celsius, and the second at about 1300 degrees. (959,924, May 31, 1910.)

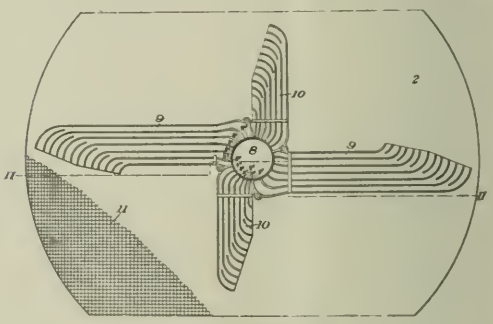


FIG. 6.—GAS SCRUBBER SPRAY.

Chemical Engineering.

Spray for Gas Scrubber.—For the purpose of spraying water or other liquids in gas scrubbers or cooling towers, WILLIAM AHLEN, of Duquesne, Pennsylvania, has patented a device of his invention. In removing finely divided solids from blast furnace gases it is necessary that the sprayed water be uniformly distributed over the cross section of the scrubber. The object of the present invention is to provide such a uniform distribution. Fig. 6 shows a plan of the spray located in the top of a scrubber. It comprises a hollow shaft 8, with rotatable arms 9 and 10. Each of the arms is provided with several passages through which the water is delivered to openings at the ends. Below the spray is a series of screens 11. In operation of the device, water is led into the pipe 8, from which it overflows through outlets into the passages in the arms. The widths of the outlets are proportioned to the lengths of the passages, progressively increasing with the length of the latter. When the water reaches the ends of the troughs or passages it overflows in a plurality of small streams, which fall in a series of annular paths as the arms rotate. The speed is so regulated that no water is thrown from the spray by centrifugal force, and as the numerous small streams fall upon the screens they are broken up into drops so that the effect is that of a fine rain. This offers the maximum cooling and cleansing surface to the rising gases. (961,195, June 14, 1910.)

Micro-Metallograph.

The study of the microstructure of metals with the aid of the microscope and by microphotographs has become in recent years a most important and useful method in metallurgical research and with the aid of improved apparatus enhancing the practical usefulness of microphotography, this method will soon be indispensable in any metallurgical testing or research laboratory.

The micro-metallograph, described in this article, has been developed by the renowned optical works of E. Leitz, Wetzlar, Germany, and embodies various interesting and important improvements. It is designed for visual observation as well as for the photomicrographic study of metallic surfaces. The construction of the apparatus embodies the principle first applied by Le Chatelier together with certain improvements suggested by Dr. W. Guertler and Dr. Wm. Campbell of the School of Mines, Columbia University, the object being placed upon the stage T, while the objective is underneath. In the act of focusing the stage moves together with the object, whereas the objective and the illuminating system remain stationary. This part of the apparatus is its most important feature (Dr. W. v. Ignatowsky, Zeitschrift f. wiss. Mikroskopie, Vol. XXV, No. 4, 1909). The apparatus has the advantage that it is available with any magnification, besides being readily adjustable in such a manner as to obviate detrimental reflections or glares.

The micro-metallograph consists essentially of three parts, all of which are mounted upon an optical bench, viz., a stand with tube and stage, a lamp with iris diaphragm and illuminating lenses, and a photographic camera.

1. *Stand with tube and stage* (Fig. 1). The stand S consists of an upright which carries the tube together with the illuminating appliances. The stage T may be raised and lowered with the aid of a rack and pinion motion and a micrometric slow motion M.

2. *The lamp.* The lamp A in Fig. 1 may be raised and lowered along the upright S_1 by means of a rack and pinion. The carbons are adjustable axially with the aid of the screws m_1 and m_2. The upright S_1 is made to slide along the bench B_2. The lamp is provided with a carrier S_2 capable of displacement along a slide bar b and fitted with an iris-diaphragm and illuminating lenses.

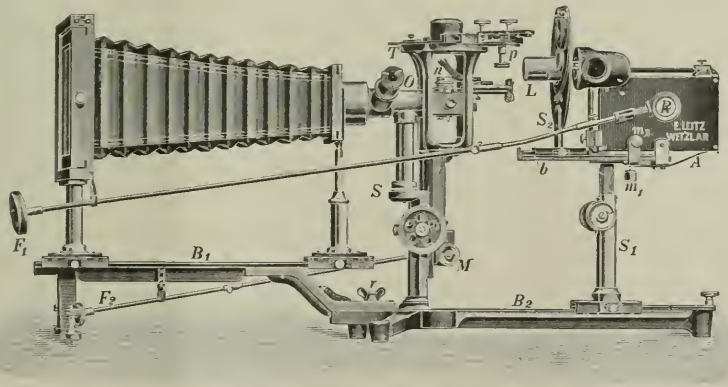


FIG. 1.—MICRO-METALLOGRAPH, STAND WITH TUBE AND STAGE.

The lamp requires an electric current of about 4 amp and with the addition of a suitable resistance may be worked from the ordinary lighting circuit. The lamp is provided with a hood fitted with a dark glass window for watching the state of the carbons. The latter are at right angles to each other. By this arrangement the crater becomes entirely free, which greatly enhances the available amount of light. The arc gap is regulated by means of a hand wheel R or by means of the adjusting key F_1.

3. *The camera.* The camera is ordinarily adapted for 13-cm x 18-cm. or 7-in. x 5-in. plates. The greatest available distance from the focusing screen to the eyepiece is 280 mm. The camera is supported by two uprights, which are capable in displacement along the bench B_1 (Fig. 1). The latter, together with the camera, after untightening the wing-nut r, may be detached from the bench B_2.

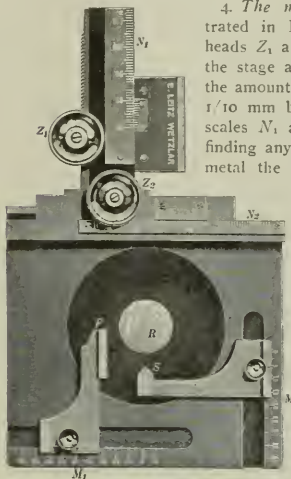


FIG. 2.—MECHANICAL STAGE.

4. *The mechanical stage* is illustrated in Fig. 2. The two milled heads Z_1 and Z_2 serve to impart to the stage a compound cross motion, the amount of which may be read to 1/10 mm by means of two vernier scales N_1 and N_2. As a means of finding any given spot in a piece of metal the mechanical stage is provided with a sliding pointer S, the position of which is registered by a scale M_2. This pointer is placed in contact with a fine file mark on one side of the specimen, while another side rests against the flat F of another sliding arm, the position of which may be seen on the scale. N_1 N_2 and M_1 M_2 furnish accordingly data with the help of which particular points of interest may be recorded and found at any time.

The mechanical stage attaches by means of the clamping screw p (Figs. 1 and 3) to the stage T, after the removal of the stage rings. These rings, of which there are three, fit a corresponding opening in the mechanical stage. In Fig. 2 they are indicated by the letter R.

To operate the instrument for *moderate and high magnification* the lamp A is placed in its lowest position by means of the rack and pinion attached to S_1 (Fig. 1), then moved along the bench B_2 as far away as possible from the stand and the iris-diaphragm I is placed (Fig. 3) in position. The carrier S_2 (Fig. 3) is then moved toward the lamp and the iris-diaphragm closed down to a very small aperture. The screw r (Fig. 3) is loosened and the tube B lowered as far as it will go, after which r is retightened. Finally, the objective side is withdrawn by the knob R_1, and the objective is screwed on. The ocular tube is capable of displacement along its axis and is fitted with a prism, which when the ocular mount is pushed in deflects the rays transmitted through the objective into the ocular. The prism may be put out of action by sliding out the ocular tube so as to allow the rays to pass without obstruction to the focusing screen.

Fig. 4, which shows the path of the rays, will assist the

reader in following the manipulations involved in the adjustment of the apparatus.

Let B mark the position of the source of light, supposing it to be reduced to a point. J represents the iris-diaphragm as shown in Fig. 3. With the aid of the projecting pin-knob s (Fig. 3) the position of the lens L_1 (Fig. 4) is adjusted within the tube B (Fig 3) until an image of the iris-diaphragm is formed at E, i. e., about midway above the upper face of prism P (Fig. 4). The latter may with the aid of the milled knob R (Fig. 3) be turned about an axis at right angles to the plane of the paper and also displaced along AB.

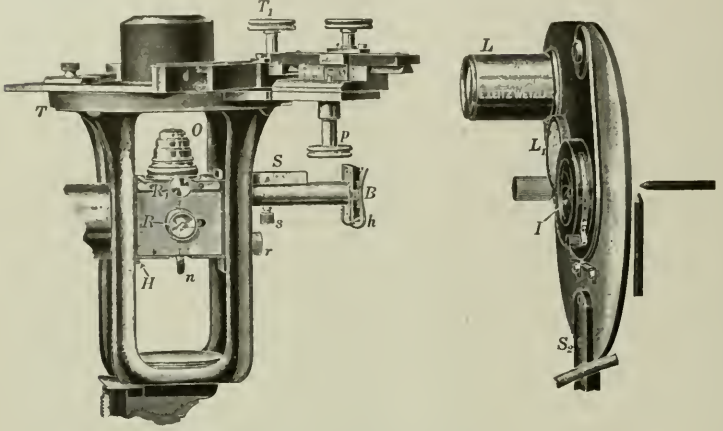


FIG. 3.—DETAILS OF MICRO-METALLOGRAPH.

The pencil of rays proceeding from P, after passing through the objective o, illuminates the object, whence that portion which goes to form the final image in the eyepiece or on the ground glass plate is reflected back through the objective, past the prism P and through prism P_1. The latter is likewise movable about an axis at right angles to the surface of the paper and may be adjusted from without by means of the pin n (Fig. 3).

The movable mount of prism P (Fig. 4) affords a means of so directing the rays through the objective o that any light which is reflected by the lens L may be intercepted by prism P to prevent its passage through prism P_1 and the creation of disturbing glares. The width of the pencil emerging from the prism P_1 which is of considerable importance, may be regulated by displacing the lens L_1, which to this end is contained in a movable mount, the distance of L_1 from P being greater with high than with low powers. To direct the correctly illuminated portion of the object into the field of view it is only necessary to turn P_1 about its axis.

Having completed the adjustment of the illumination, the operator may make sure of the absence of glares, which are particularly disturbing in photographic work, by removing the object and replacing it by a non-reflecting body, preferably black velvet. The field as seen through the eyepiece should then be black. After a little practice this test will become superfluous.

Having secured the proper adjustment as controlled through the eyepiece tube, the latter is put out of action and the image is made to appear on the groundglass focusing screen. Photographs are taken by this apparatus with the aid of an eyepiece as well as the objective, a fitting containing an eyepiece being inserted into the sleeve attached to the camera. The final focusing and the adjustment of the carbons is effected with the aid of the transmission gears F_1 and F_2.

The milled head R and the pin n (Fig. 3) are provided with index marks corresponding to convenient initial positions of prisms P and P_1 from which the operator may proceed to make his adjustments.

The image should be evenly bright and undistorted. The presence of distortions should be regarded as an indication that prisms P and P_1 are not properly adjusted.

Tube B (Fig. 3) is fitted with a slide, which should, of course, be opened at the outset. The spring clips at the end of the tube serve for clamping colored glasses or light-filters.

The back of the carrier S_2 (Figs. 1 and 3) is fitted with a shutter for use during photographic exposures.

To operate the instrument for *low magnifications* the lamp should be racked up, as shown in Fig. 1, and moved up as near as possible to S. The glass plate n (Fig. 1) is mounted with its carrier upon the slider S (Fig. 3) and together with tube B raised to the required height and clamped with the aid of screw r. The slide at the end of tube B should be closed. When microsummars (magnifications 9.5 and 13) are used, the operation of the instrument is somewhat different.

For the purpose of cleaning the illuminating apparatus the prisms P and P_1 may be withdrawn from the tube mounted on stand S (Fig. 1). To do so the hook H (Fig. 3) is released, the objective slide is removed, and the illuminating apparatus withdrawn by means of the button R.

When the object is of such dimensions that it cannot be accommodated on the mechanical stage the latter should be removed and the object placed direct upon the stage T (Figs. 1 and 3). The rings of the mechanical stage fit the gap in T. The range of magnifications with fully extended camera is as follows: Without eyepiece and with microsummars, magnifications 9.5 and 13; with eyepiece No. 2 and objectives 1 to 10, magnifications 26, 46, 73, 132, 210, 305, 370, 465, 650 and 710.

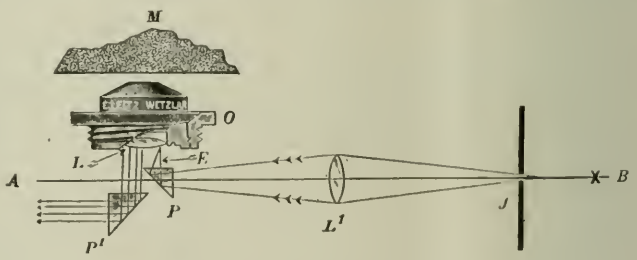


FIG. 4.—PATH OF RAYS.

The optical works of E. Leitz have branches in this country at 30 East Eighteenth Street, New York City, and 1923 Ogden Avenue, Chicago, Ill. This new micro-metallograph has already been introduced in some of the leading metallurgical works in this country, like the Watertown Arsenal, the Scovill Mfg. Co., The Winchester Repeating Arms Co., The American Vanadium Co., etc., and has given complete satisfaction.

High-Speed Forging Press.

A machine which has come into general use very rapidly in the last few years is the high-speed forging press. This machine has been developed abroad and has been extensively used in England and Germany for as much as 10 years, but has been employed in this country, up to a year ago, by only a few of the very large companies who make the heavier class of forgings.

The machine, as shown in Fig. 1, represents the very latest development in forging machines for large work, and Fig. 2 is the same machine, but built with an overhanging frame, which is considered better for the smaller powers. These stand for years of step-by-step progress by Davy Brothers, Limited, of Sheffield, England, who have been designers and builders of—first, steam hammers, and later hydraulic forging presses.

The earlier presses built by this company were of the pump type—that is, the water pressure was derived from a pump which was necessarily slow, with the disadvantage that the water must pass through valves at high pressure, relatively long lines with many joints and numerous drawbacks to a large water system, one being its liability to carry grit which cuts out plungers and packings, another, the liability of damage on account of water hammer.

The next broad step after the direct pump-driven press was the hydraulic intensifier—this made it possible to raise the pressure from somewhere between 1000 lb. and 2000 lb. per

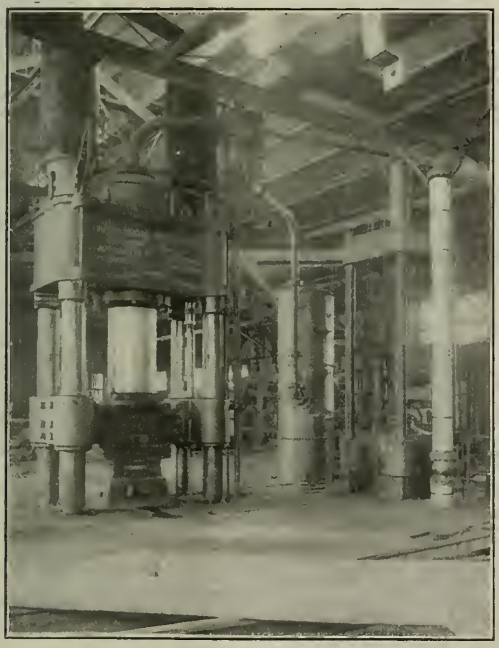


FIG. 1.—FORGING PRESS.

square inch, to 4000 lb. to 6000 lb. per square inch. This makes the main press cylinders and the volume of water correspondingly small, which is a very great advantage in that the press of large power becomes of practical dimensions and the volume of water under this high pressure is relatively small and in an enclosed system, not handled through valves, and only under this high pressure at the instant of delivering the squeeze.

This, of course, was a long step forward, but this machine still required a pump, and the system generally included a large accumulator in addition to the hydraulic intensifier, and was operated by a number of valves and levers. This press was slow, partly owing to the number of valves operated and partly due to the low speed of the hydraulic intensifier.

This intensifier, owing to the fact that the large volume of low-pressure water which must be handled at low speed, was superseded by the steam intensifier. This device took steam direct from the boilers, which acts directly on the low-pressure piston, and gives you at once the high-pressure water required, without the use of the pump. Another great advantage is that

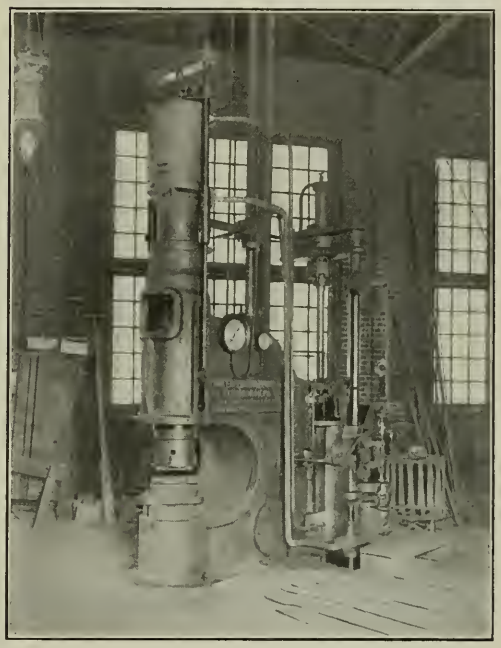


FIG. 2.—FORGING PRESS.

the steam, being highly elastic, can be handled at a very high rate of speed.

There are then two pistons directly connected together, with a ratio of about 40 to 1, and when this device is set vertical, with the water cylinder on top, the return stroke is accomplished by gravity, so that steam is used on one side only. The high-pressure cylinder can then be directly connected to the main cylinder of the press without the water passing through valves. The press head then moves at the same time as the intensifier ram and always a corresponding amount, the intensifier moving about 12 times as far as the press ram. One then has only to control the travel of the intensifier, both as to time and distance, and one has a corresponding control of the presshead, with the advantage that on the presshead the control is as 12 is to 1 better than on the intensifier.

This provides for a quick-working stroke, but it is also necessary to return the presshead quickly. This was accomplished by placing steam cylinders on top of the machine, and also arranged to take steam on one side of the piston only. These cylinders are arranged so that, when forging or intensifier stroke is being used, they do not consume any steam, it simply flowing back and forth in the main pipe.

It now becomes evident that, unless the intensifier be of very large proportions, the stroke of the press will be very short An intensifier of sufficient capacity to give a practical forging stroke, and also give the proper amount of clearance for a

diversified line of forgings, would be impractical owing to its size, and also on account of the enormous amount of steam required to operate it.

It then becomes apparent that the thing to do is to lower the presshead nearly in contact with the forging, by means of the steam balance cylinders. It is then found necessary to have more or less water in the high-pressure system according to the thickness of forging being operated upon, which, of course, fixes the distance in or out of the main cylinder occu-

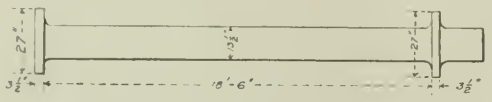


FIG. 3.—SHAFT.

pied by the main ram. This was accomplished by having a tank connected with this high-pressure system and controlled by a check valve. This tank is about half-filled with water, the balance being filled with air at about 30 lb. per square inch. When the presshead is lowered by wholly or partially exhausting the steam from the balance cylinders, the check valve automatically opens, allowing the required amount of water to flow from the tank into the system. This check valve is also arranged to operate with a lever connection so that when it is required to lift the presshead, this can be accomplished by opening the check and allowing the balance cylinders to force back into the tank an amount of water that will give the proper position of presshead.

This action, or long light stroke, takes place always when the intensifier is in its lower position, or at rest, so that it can be readily seen that the presshead can be lowered until it is practically in contact with the forging, and the steam in the intensifier used only for a stroke in proportion to the amount the forging is reduced, and an amount directly in proportion to the resistance met or the work done. This makes a press very efficient in action as, once the presshead is let down on the forging, the low position is maintained until the forging is finished, using no more steam in the balance cylinders as this is forced back and forth from the cylinders to the main steam pipe.

This made a satisfactory press so far as any one stroke was concerned, but when rapid action was required it was found impractical to operate a number of valves or levers for the rea-

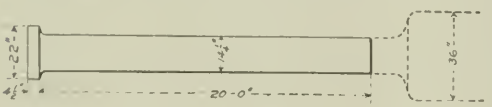


FIG. 4.—SHAFT.

son that if one was moved out of time the result was disastrous to the machine or the work, and it developed slow because it was necessary to move one valve or lever and be sure that the press assumed a corresponding position before moving the next one. This invariably results in the wrecking of the machine through a slip of the operator.

The Davy Brothers, Limited, realizing this, devised the single-lever control, which is patented in the United States as well as abroad. This control operates all valves and permits of putting the press through an entire cycle of motions in one forward and back motion of the lever. The valves cannot be crossed and the press follows exactly the stroke of the hand lever, both as to speed and distance traveled. The lever is so arranged that back of the vertical center line corresponds to the light stroke, which is from 2 ft. to 6 ft., according to the size of the press, and forward of that line is the power stroke from 2½ in. to 6 in. long, according to size of the press.

On account of this arrangement the press can be brought down 2 ft. to 6 ft. on top of a forging and the forging reduced from 2 in. to 6 in. by one forward motion of the hand lever, then the press can be brought all the way back, or any part, by simply moving the lever to the required position, or the lever can be brought back only as far as the center line, which will simply return the presshead the amount of the forging stroke. This forging stroke can then be repeated as often as the lever can be moved. In practice this is as high as 150 per minute in the small presses, down to 60 on the large ones.

This stroke, while it is repeated as often as the blow of a steam hammer, is at a much lower velocity owing to the fact that the press moves only inches where the hammer moves feet. This slow action is very beneficial to the metal because it works it to the center, giving perfectly sound forgings. There is absolutely no danger to the smith owing to the absence of shock or jar. This is very advantageous when shape tools or cutters are used as they can be set up, the presshead brought down on them, and then sighted for position before the pressure is applied.

In cutting or punching, the hand lever can be set for a certain travel and the presshead will not ride forward even though the resistance is suddenly and entirely removed. On account of this, there is no danger from flying tools or work.

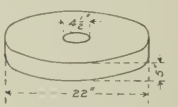


FIG. 5.—DISK.

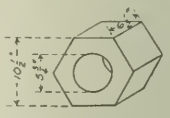


FIG. 6.—NUT.

The machine does not require much foundation, and in working there is no vibration transmitted to surrounding buildings as all strains due to working are taken up in the machine itself.

The United Engineering & Foundry Company, of Pittsburgh, Pa., acquired the sole right to manufacture this machine for the United States, Canada and Mexico about one year ago, and have complete drawings, patterns, jigs, fixtures, etc., for all sizes of these machines, as well as data from the Davy Brothers, Limited, which gives them the advantage of many years' experience abroad on all classes of forgings. They have already built and delivered quite a number of these presses, which are at work in various parts of the country. These presses are working on a diversified line of forgings, which include cogging down ingots, wheel forgings, locomotive forgings, and a general line of forgings.

The high-speed press will do very much more forging than any steam hammer, as can be seen by the following:

Fig. 3.—A 2000-ton press forged the 13½-in shaft, 19 ft. long, with 27-in. diameter flanges, from a 36-in. ingot in two hours, requiring two heats.

Fig. 4.—A 1500-ton press forged a 14½-in. shaft, 20 ft. long, with coupling on one end, from a 36-in. ingot, in one heat, in one hour.

A 1200-ton press on cogging 16-in. x 16-in. ingots to 10-in. x 10-in., and cutting into 18-in. pieces, averaged 16 tons per hour.

Figs. 5 and 6.—A 500-ton press made a disk 22 in. diam-

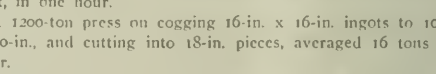


FIG. 7.—SHAFT.

eter, 5 in. thick, upsetting it, from 14 in. square, and punching a 4½-in. hole, as per Fig. 5, in 19 minutes, in one heat. The same size press made and punched a hex nut 10½ in. across flats in 18 minutes, as per Fig. 6, also in one heat.

Fig. 7.—A 300-ton press made a 6¼-in. shaft, 70 in. long, in 12 minutes, from a 7-in. x 7-in. billet in one heat.

Figs. 8 and 9.—A 150-ton press made an eccentric shaft, as

per Fig. 8, from an 8-in. x 8-in. billet, in 13 minutes in one heat. The same press made hex nuts 6½ in. across flats and punched same, upsetting it from 5-in. round stock, in 10 minutes, as per Fig. 9.

The United Engineering & Foundry Company has under-

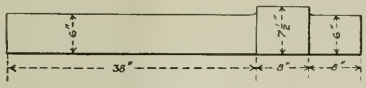


FIG. 8.—SHAFT.

taken to produce a machine in every way first class, among the many refinements of which might be mentioned the following:

The main rams are cast in a chill, which makes them very hard, and are accurately ground to fit the cylinder, which is steel, and bored and lapped out. The intensifier ram is made of hard alloyed steel accurately ground; all joint pins and bushings are hardened and lapped. The material entering into the press is all high grade and nearly all steel. All packings can be removed without dismantling any part of the press, or employing any power outside of the press itself.

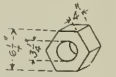


FIG. 9.—NUT.

Jet Machines.

BY OSKAR NAGEL, PH.D.

The working of every jet machine is based upon the fact that a liquid or gaseous jet passes through another liquid or gas (under pressure from a nozzle), causing a motion from the adjoining parts of the mass, so that a vacuum would be formed, if it would not be immediately annihilated by the motion of the particles of the respective substance. The jet draws and sucks in the surrounding mass, mixes with it and transmits to same a part of its own momentum so that both escape together. If now the common velocity is reduced by a counter pressure, a corresponding part of the momentum of the mixture is transformed into pressure which serves to overcome the counter-pressure.

If a steam jet passes through a cooler liquid or if steam is sucked in by a cooler jet formed of a liquid, the steam is condensed in the mixing process and the suction will be the more intense the more complete the condensation.

We now see that in the ordinary jet machine work is produced by using part of the velocity of the passing steam for the transportation of other substances and that in other cases the heat of the steam liberated by condensation is so transformed that it serves for the transportation of other gaseous or liquid substances. In every one of these cases the energy content of the steam is changed. In each case a transformation of energy takes place in the steam.

FIG. 1.—STEAM JET BLOWER.

Since the jet machines on account of their having no moving parts are simpler and more convenient than other machines their application is steadily increasing, especially in the chemical industries. The introductory remarks of this article indicate the most natural and most advantageous uses of the jet machine, namely, to move gases and liquids, the term "move" being used in its widest possible meaning.

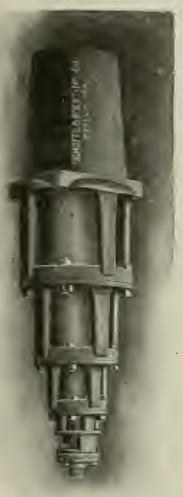


FIG. 2.—CHIMNEY VENTILATOR.

One of the most popular jet machines is the steam-jet blower, Fig. 1, which is mainly used to press air into gas producers or other furnaces and to ventilate cellars, etc.

The chimney ventilators, Fig. 2, which are used to improve the draft of chimneys, also belong to this class, their operation being based upon a series of nozzles in which the motive power of a small steam-jet is utilized for moving the gases; for the removal of acid vapors in chemical work these nozzles are made of lead.

For compressing or exhausting air or other gases, i. e., for vacuum filtration, vacuum distillation, and agitation, the steam-jet exhausters, Fig. 3, are successfully used. (The well-known laboratory type of this instrument is shown in Fig. 4.) A water-jet exhauster for similar purposes is shown in Fig. 5.

Of special interest for the sulphuric acid industries is the lead steam-jet exhauster, and lead blower, Figs. 6 and 7.

The most popular jet apparatus for moving liquids is the injector, which is too well known to require a description at this place. Of greater importance to the chemical industries is the steam-jet syphon, Fig. 8, in which by means of a jet of steam issuing from a nozzle, a vacuum is created which is conveniently utilized for lifting, circulating or transferring liquids. The corresponding apparatus worked by compressed air is called air-jet lift. Its application in the acid industries is easily understood from Fig. 9. The water-jet eductor is built upon the same principle and is mainly used for lifting liquids (Fig. 10).

The above remarks will convey to the reader some idea as to the applications and applicability of jet apparatus. These instruments and their application were developed in Germany by the efforts of the firm of Gebrueder Koerting, Koertingsdorf, near Hanover. In this country these jets are built by the Schutte & Koerting Company, Philadelphia, Pa.

FIG. 3.—STEAM JET EXHAUSTER.

Positive Blowers.

Positive blowers are now being generally introduced in different lines of service for low and moderate pressures up to 8 lb. or 10 lb. per square inch.

The Piqua type of blower has two impellers geared together

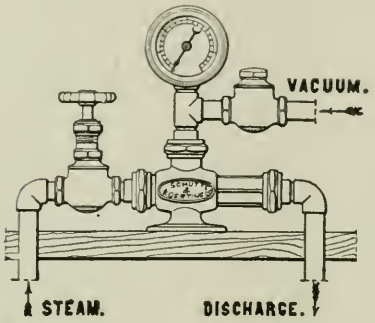


FIG. 4.—LABORATORY TYPE OF STEAM JET EXHAUSTER.

and rotating with very small clearances in rigid semi-cylindrical cast-iron casings. With no internal friction, carefully constructed bearings and flat cut gears dipping in oil, the efficiency of this type of blower is high.

Due to their economy in operation the few parts and simple and rugged construction, and the comparatively low first cost, positive blowers within their pressure limits have a certain advantage over reciprocating air compressors or blowing engines. Also their almost uniform efficiency over the entire range of pressure and volume enables them to readily surpass the centrifugal blower or fan except when this is driven at its exact designed volume and pressure.

For service in the production of grey iron castings, in the foundry cupola, the definite known volume of air delivered, regardless of the changing pressure, makes for an output uniform in quality and increased in quantity.

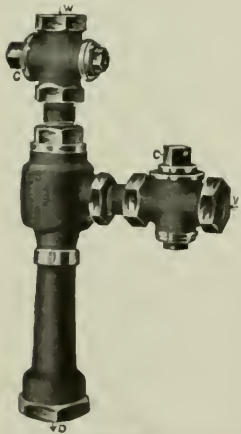


FIG. 5.—WATER JET EXHAUSTER.

In brass and aluminium foundries positive blowers are also giving satisfaction, while for copper and silver-lead smelting the reliability for continuous operation is appreciated.

Blowers for fuel-oil furnaces is another useful application of machines up to 2 lb. and higher pressure blowers have been extensively used to furnish the blast for steel converters.

Among the other uses to which this type of blower is well adapted might be mentioned furnaces for annealing, hardening and tempering; operating sand blast apparatus, agitating, cooling, or evaporating liquids, pneumatic tube conveying, mine ventilation, gas boosters or as exhausters for maintaining low vacuum for gas producers, etc

The Piqua Blower Company, Piqua, Ohio, has just issued an attractive 48-page catalog M-10 illustrating their line of positive blowers. These run from laboratory size to machines displacing 15,340 cu. ft. of free air per minute and delivering at standard pressures up to 16 oz. or 32 oz. and in the high pressure class, 3 lb., 4½ lb., 6 lb. and 8 lb. per square inch. The machines are furnished for belt drive or are direct connected to motors or engines.

NOTES

The General Chemical Company of California, with principal office in New York, has been incorporated with a capital of $3,750,000. The directors include William C. Peyton, of San

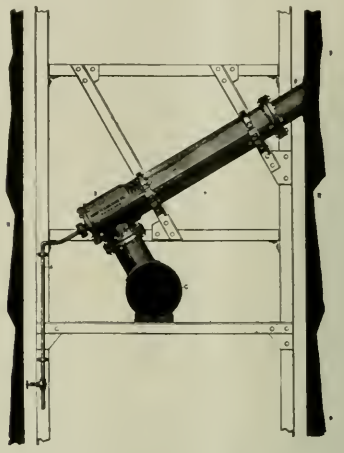


FIG. 6.—STEAM JET EXHAUSTER FOR SULPHURIC ACID PLANTS. (A, STEAM SUPPLY PIPE; B, LEAD EXHAUSTER AND BLOWER; C, GAS SUPPLY; R, CHAMBERS.)

Francisco; Charles W. Millard, Thomas F. Burgess and Frank H. Edmunds, of New York.

The Pierce Amalgamator has been moved from 1654 Champa Street to 1738 Broadway, Denver, Col. This change was made as more roof and better facilities were needed by the company to display their machines, which have been attracting

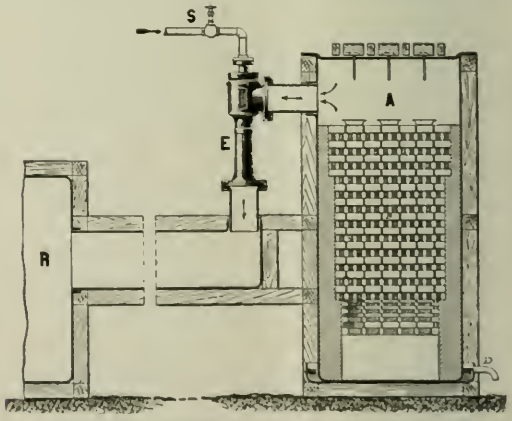


FIG. 7.—LEAD BLOWER FOR SULPHURIC ACID PLANTS.

more and more the attention of metallurgical engineers during the past year. In their new home the company will have spacious office quarters and display room and in connection with this they will have a very complete testing plant in operation ready at all times to demonstrate just what can be accomplished with the Pierce amalgamator.

Recording Instruments.—The Bristol Company of Waterbury, Conn., has just issued "an illustrated index of Bristol's recording instruments for pressure, temperature and electricity, and the William H. Bristol electric pyrometers for high temperatures." This is not a catalog, but a very interesting series of 57 beautifully executed illustrations of the most important Bristol instruments, with a brief and concise statement of their principal features and applications. The illustrations include recording pressure gages, vacuum gages, draught gages, differential pressure gages, water level gages, recording thermometers and pyrometers, recording voltmeters, ammeters and wattmeters, time recorders, recording tachometers, etc.

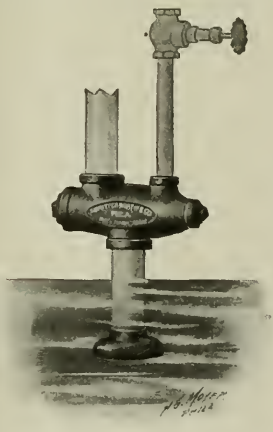


FIG. 8.—STEAM-JET SYPHON.

Sulitelma Mines.—Advice has been received from the Sulitelma Company's mines in Norway that for the month ending May 30 763 tons of concentrates, containing 6.22 per cent copper, were produced by the Elmore process.

The **Brown Instrument Company**, of Philadelphia, has just issued a 45-page illustrated catalog on Brown electric pyrometers, both indicating and recording pyrometers of the thermo-electric type. In the second part the Brown fixed-focus radiation pyrometer is described. The pamphlet contains numerous interesting and instructive illustrations and discusses in detail the methods of installing the various parts of the electric pyrometer and its use in different industries.

The Williams Patent Crusher & Pulverizer Company, of St. Louis, Mo., has just issued a new cement edition of their catalog describing their various well-known crushing and grinding machines, with special reference to their use in the cement industry.

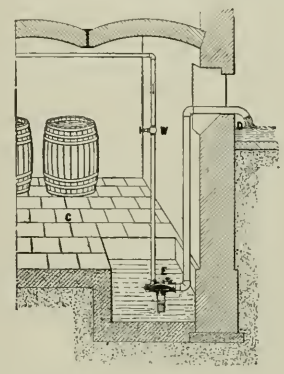


FIG. 10.—WATER JET EDUCTOR FOR LIFTING LIQUIDS.

The **Taylor Instrument Companies** called their branch managers and principal salesmen together to their annual convention in Rochester, N. Y., on July 11. Three busy days were spent by the officers of the companies and the men in reviewing the general conditions in the thermometer industry and outlining new measures and policies for the coming year. The Taylor Instrument Companies are just closing their most successful year. Although the present plant of the companies was occupied only in 1906, the growth of the corporation has been so rapid as to necessitate material additions for the manufacture and handling of "Tycos" products. One building, measuring 100 ft. x 160 ft., will be used exclusively as a wood-working shop for the preparation of all the lumber entering into the products of these companies. A dry kiln is also under construction for the drying of rough lumber. These buildings, with a new wing added to the main building, will make an addition of more than 40,000 sq. ft. of floor space to the present plant.

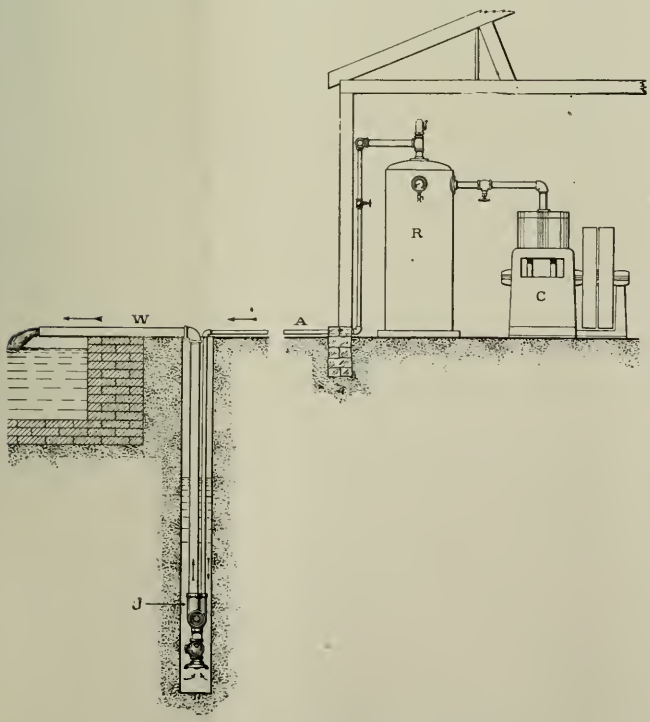


FIG. 9.—APPLICATION OF STEAM-JET SYPHON IN ACID PLANT.

Turbine Fan.—A booklet on a new turbine fan has been issued recently by J. R. Robinson, of Pittsburgh, Pa. This fan is of "mixed-flow" type, i.e., air is taken in axially at the center, turned through 90 deg. and discharged radially by centrifugal force at the circumference. The vanes are curved to pass the air through with minimum friction, and it is claimed that considerable saving in power is effected over that required by various standard fans on the market. The fan is suitable for mine ventilation or for ventilation of power houses and other buildings and for high-pressure work with cupolas, forges, smel-

ters and refineries, furnaces, glass houses, etc. Listed sizes run from 18 in. diameter, 17 in. width, 2500 r.p.m. and 21,000 cu. ft. per minute discharge to 312 in diameter, 105 in. width, 120 r.p.m. and 1,200,000 cu. ft. per minute approximate capacity. A number of these fans are in satisfactory service, particularly in mines, and in addition to a power saving they require a comparatively small amount of space and have proved themselves capable of maintaining an unsually high water gage. In a later issue we expect to give further information with illustrations and data.

Why Solder Perishes.—Under this caption we find in a British contemporary some notes which appear to be based on recent publications in *Zeit. f. Phys. Chemie.* (Vol. 68, 1909, p. 214 and vol. 71, p. 301). V. Haslinger observed some time ago that all the seams soldered with tin of a tinned iron can were broken, because the tin had crystallized. By contact with this "sick" tin other polished tin surfaces could be infected so that they would get rough. E. Cohen has recently investigated this phenomenon and confirmed Haslinger's observations. The explanation is not that the tin passes from one crystalline form into another, nor has the phenomenon to do with the so-called "tin pest" in which ordinary tin changes to gray tin, which occurs below 20° C. In our case higher temperatures favor the crystallization of the tin. What really happens is that "machined tin" is in a forced or metastable state; tinned iron sheets, rolled tin (tin foil), and any brilliantly polished tin are in this metastable state; at higher temperatures the tin gives up its metastable state and crystallizes and the continuity of the soldered seam is broken. E. Cohen and J. K. Inonye have recently extended this investigation to other metals. We intend to deal with this phenomenon which is also of practical interest, in greater detail in a future issue.

Oil and Gas Burning Furnaces and Appliances.—Some interesting circulars have been issued within the past few months by Tate, Jones & Company, Inc., of Pittsburgh, Pa. No. 129 covers fuel-oil burning equipment for open-hearth furnaces and illustrates the Kirkwood oil burner, swinging and water-cooled types, also a reversing valve stand and the oil pumping, heating and regulating system which goes along with the installation. Oil is being used successfully as a fuel in many open-hearth steel furnaces; it is generally the cheapest fuel except in the immediate neighborhood of coal and natural gas fields, and the heat of the furnace is more easily regulated than is possible with producer gas. Even where natural gas and coal are abundant it has been found advisable in many instances to install auxiliary oil-burning equipment for emergency use. The Kirkwood burner is a combination high and low-pressure oil burner in which the oil is fed to the burner at from 15 lb. to 25 lb. pressure, and atomized by a small quantity of compressed air or steam at a slightly lower pressure. An air blast of from 2 oz. to 6 oz. is used to supply the necessary amount of oxygen for complete combustion. This burner is suitable for open-hearth furnaces, glass-house furnaces and leers, small steam boilers, brick kilns, rivet forges, heating and annealing furnaces, etc.

PERSONAL

Mr **J. F. Callbreath, Jr.**, secretary of the American Mining Congress, has been to Los Angeles, making preparations for the meeting of the Congress there Sept. 26 to Oct. 1.

The wedding is announced of Mr. **James M. Camp**, chief chemist of the Carnegie Steel Company, and Miss Isabella R Wightman, of Pittsburgh, Pa., on June 30.

Mr **Louis S. Cates** has accepted the position of superintendent of mines for the Ray Consolidated Copper Company, Ray, Ariz.

Mr. **Robert Kennedy Duncan**, who has been professor of industrial chemistry at the University of Kansas, will hold a similar position in the University of Pittsburg the coming year.

Mr **C. E. Foster**, who has been connected with the Taylor Instruments Companies of Rochester, N. Y., has finished the special work for which he came to this country and has now returned to England.

Mr. **H. W. Hardinge**, president of the Hardinge Conical Mill Company, sailed on July 16 on a professional trip to Sweden.

Mr. **H. E. T. Haultain**, of Toronto, has been appointed professor of mining in Toronto University.

Mr. **E. N. Hawkins** will spend the summer in Denver, after an absence of several years, during which time he has been engaged in professional work in the Northwest.

Mr. **George T. Holloway**, consulting metallurgist, chemist and assayer, announces the removal of his office and laboratories from Chancery Lane to 9-13 Emmett Street, Limehouse, London E.

Mr. **Frederick Lyon**, assistant managing director of the U. S. Smelting, Refining & Mining Company, was in Salt Lake City recently after a trip to the company's smelter at Kennett, Cal.

Mr. **Frederick H. Morley**, of Denver, has gone to British Columbia on professional business.

Mr. **E. L. Oliver** has opened an office at Oakland, Cal., and will engage in consulting metallurgical practice. He has resigned his position with the North Star Mines Company, at Grass Valley, where he developed the Oliver continuous slime filter.

Dr. **Harrison Patten**, of the Bureau of Soils, Washington, is the author of a novel, "The Town and the Trust," just brought out by the Neale Publishing Company.

Prof. **Robert Peele**, of the Columbia School of Mines, New York, has gone to South Africa on a six months' leave of absence to study mining operations.

Dr. **F. W. Traphagen**, professor of metallurgy in the Colorado School of Mines, is spending the summer in Montana in professional work.

Mr. **George J. Young**, of the Mackay School of Mines, Reno, Nev., recently returned after an eight months' leave of absence, during which he visited many European institutions of mining education.

BOOK REVIEWS.

Transactions of the American Institute of Chemical Engineers. Vol. II.: 1909. 308 pages; illustrated. Price, $6. New York: D. Van Nostrand Company.

Besides official notices and reports, the second volume of the *Transactions* contains the following papers: Charles F. McKenna, "The Centering of Great Industries in the New York Metropolitan District"; Charles E. Munroe, "Chemical Industries of America"; Samuel P. Sadtler, "Conservation and the Chemical Engineer"; William D. Ennis, "Efficiency Limits in the Power-Gas Producer"; O. K. Zwingenberger, "Utilization of Low-Grade Fuels in the United States"; A. Bement, "Heat Efficiency of Smokeless Combustion and Heat-Absorbing Capacity of Boilers"; A. Bement, "Chemical Composition of Illinois Coal and the Relation of Heating Value to Composition"; Samuel P. Sadtler, "Creosote Oil from Water-Gas Tar"; John C. Olsen, John S. Weiffenback and John R. Brierly, "Experiments on the Case-Hardening of Steel by Gases"; R. K. Meade, "Automatic Acid Egg"; John G. Dean, "Method of Clay Control for the Manufacture of Cement"; Jerome Alexander, "Colloids and the Ultramicroscope"; Stephen P. Sharples, "Utilization of Waste India Rubber"; Launcelot W. Andrews, "Permanence and Acidity of Hydrogen Peroxide Solutions"; Wm. M. Booth, "Commercial Extraction of Grease and Oils"; F. J. Wood, "Glycerine Refining in Multiple-Effect Stills"; A. C. Langmuir, "Advantages of the Multiple-Effect Distillation of Glycerine and Other Products"; Edward R. Taylor, "Electric Furnace for the Smelting of Iron Ore"; Edward R. Taylor, "Waterways and Water-Power."

The frontispiece is an excellent portrait of the president, Dr. Charles F. McKenna.

Metallurgical and Chemical Engineering

A Consolidation of Electrochemical and Metallurgical Industry and Iron and Steel Magazine

Vol. VIII. New York, September, 1910 No. 9

PUBLISHED MONTHLY BY THE

Electrochemical Publishing Company

239 West 39th Street, New York.

Pittsburgh Office, 201 Oliver Building.
Western Office, 1510 Court Place, Denver, Colo.
European Office, Hastings House, Norfolk St., Strand, London, Eng.

J. W. Richards, Ph.D., President.
Curtis E. Whittlesey, Treasurer. E. F. Roeber, Secretary.
Telephone Call: 4700 Bryant.

E. F. Roeber, Ph.D., Editor.
H. C. Parmelee, Western Editor.

Yearly subscription price for United States, Mexico and United States dependencies, $2.00; for all other countries, $2.50 (European exchange, 10 shillings, 10 marks, 12.50 francs).

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Entered as Second-Class Matter at the Post Office at New York, N. Y., under the Act of Congress, March 3, 1879

New York, September, 1910.

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The Westward Trend.

It is no disparagement to the East, where the foundation of this country has been laid, to acknowledge the fact that the centre of productive activity is steadily moving westward. While the nerve centre of this country will remain in New York, which appears destined to grow in restlessness and nervosity, the productive capacity grows by westward expansion of the development, going on with gigantic strides, of the bountiful natural resources of this country. To every action there is a reaction, and to prove that this westward trend reacts on, and changes, the East, the instance of Mr. Roosevelt's career is sufficient. The policy of the conservation of the natural resources—perhaps the most momentous of all problems in the modern political and industrial life of this country—is born of the necessity to regulate this westward expansion in the best interest of all the people. In this westward expansion the engineer takes an important part. He is the pathfinder. The builders of the roads, the bridges, the railways, are the advance guard of the army that is out to conquer nature. It is essentially the civil engineer, the mechanical engineer, the mining engineer who bear the brunt of this pioneer work. The metallurgist and the chemist follow later, for the same reason why metallurgical and chemical engineering are younger professions than civil, mining, and mechanical engineering. As long as placer mining pays, there is no need for chemical and metallurgical research. But the time comes when the chemist and metallurgist are needed. It often comes "too soon." When the rich ores are exhausted, the simple methods fail and the metallurgist has to devise new ways and means. This is the raison d'être of the rapidly growing importance of the specialized metallurgical engineer in the West and of the slow but sure dissociation of the metallurgical from the mining engineer.

* * *

Chemical engineering, aside from metallurgy, needs a more highly developed civilization to manifest intensively its usefulness. But the time seems to have arrived when the chemist should begin to interest himself more energetically in the West from a professional standpoint. The American Chemical Society recognized this by holding its Summer meeting in San Francisco; and this meeting was so successful that it will be remembered as a milestone in the career of the oldest of our chemical societies which has regained so much youthful vigor and energy in recent years. That the West is bound to gain much from chemistry in years to come, goes without saying. That the pilgrims from the East who participated in this trip to the West brought back for themselves Western enthusiasm is evidenced by the story of the trip told by one of the participants in our correspondence columns in this issue—by the way, a little literary masterpiece. The American Electrochemical Society, though not going so far, follows in the same direction by holding its Autumn meeting in October in Chicago. So far

quite an attractive industrial sight-seeing program has been arranged and it may be hoped that the meeting will attract many Eastern members. "East is East and West is West." But the extremes should meet each other in their own quarters. It is an indication of sound judgment of the managing councils of our chemical societies that they pay due regard to the westward trend of industrial evolution and its importance for the future of chemistry in this country.

Copper-Nickel Steels.

The fact brought out in Mr. David H. Browne's paper before the American Institute of Mining Engineers, that in the bessemerizing of copper-nickel matte, copper and nickel together act like a single metal and that in the monel alloy directly produced from the ore the nickel and copper are contained in practically the same proportion as in the ore, is surprising in many respects. As explained by Mr. Browne in detail, the facts disprove some former theoretical notions on the subject. It is perhaps even more surprising that copper and nickel when added together to steel should behave also to a certain extent like one metal, so that it is possible to replace a certain amount of nickel in nickel-steel by copper, without changing the properties materially. In our last issue we published a very interesting paper by Prof. C. F. Burgess and Mr. J. Aston on copper-nickel-iron alloys, made from pure electrolytic iron, free from carbon. Not less interesting is a paper recently presented by Mr. G. H. Clamer before the American Society of Testing Materials and published in abstract in our present issue. It supplements the Burgess-Aston paper in the practically important point that the steels of Mr. Clamer contained carbon. The problem is, of course, an exceedingly complicated one and further detailed and systematic researches are necessary. But the results so far obtained look decidedly promising, and it really seems that copper has about finished its rôle as a bugbear in steel metallurgy. If monel steel is practically as satisfactory as nickel steel, the considerable price difference between the monel alloy and nickel should have a decisive effect and should open for nickel steels—or rather for monel steels—many new applications in engineering. That even the best informed are still in the dark with respect to some of the most fundamental questions is proven by Mr. Browne's frank statement that by melting copper and nickel together they could not produce an alloy identical in its properties with monel alloy obtained directly from the ore, in spite of apparently the same composition.

Superficial Criticism.

In a recent address before the Technical and Engineering Society of the Colorado School of Mines, Mr. George E. Collins discussed the fallacy of passing criticism on technical methods in use in any district, without a full knowledge of all the conditions. The milling methods of Gilpin County, Colorado, have been subject to drastic condemnation, but when they are examined in the light of the conditions under which they are applied, they are found to be a logical development of those conditions, and, in the main, perhaps, satisfactory. The milling problem of a district like Gilpin County is not a simple one. The diversity of ore and the large number of mines producing small tonnage make the problem complex, with the result that the ultimate practice must be somewhat of a compromise. Thus it is that the superficial observer, visiting mills of this kind, is apt to level his criticism at the methods without full knowledge of the reason for their existence. As a matter of fact, none of the mills may be best suited to the ore from the technical point of view, but may, nevertheless, give the best commercial saving over a long period of operation.

* * *

A favorite form of criticism of milling methods is based on superficial observation of the mineral particles passing into the waste. The customary comment is to the effect that "modern" devices would effect a better saving. Quite probably they would. Perhaps, also, the machines already in use could make a better saving, but at a cost not taken into consideration by the critic. The commercial side of ore reduction is the ultimate criterion of success. Where freight and treatment charges have to be considered, it is frequently a nice problem to solve, whether to ship the bulky crude ore and incur the freight and treatment expense, or subject it to a dressing process with its attendant losses. Generally speaking, the methods existing in an old district cannot be compared with modern methods, and should not, therefore, come under condemnation on that score alone. The methods used in such a district usually represent the combined judgment of many men in one or two generations, and represent a very considerable capital investment. Wholesale scrapping is often not only a sign of progressiveness, but shrewd business policy which soon pays. Many examples from the history of the iron and steel industry prove this. But as long as the owner of a large works in which considerable capital is invested is satisfied to let things well enough alone, progress cannot be hoped for from new methods, but rather from new men working under the given conditions and improving them slowly.

Kick's Law.

An engineering rule, like a scientific law, should be a statement of fact based on a large number of observations which give concordant results. A law thus formulated cannot be said to be infallible, but it is true for our state of knowledge at the time and stands accepted as long as no new facts are discovered which controvert it. There is always the possibility that the law may not stand the test of newly discovered facts, and frequently a generally accepted law may not be questioned until something arises to direct special attention to it. Furthermore, having developed an acceptable scientific law, there still remains the necessity of making proper application of it to the solution of any problem. The recent admirable efforts to determine the crushing efficiency of various machines has directed renewed attention to Kick's law, which is stated thus: The energy required for producing analogous changes of configuration of geometrically similar bodies of equal technological state varies as the volumes or weights of those bodies.

* * *

The proposal to apply the elements of this law to determine relative efficiencies of crushing appliances, from a grading analysis of the crushed product, has not been accepted without question. While no doubt is cast on the mathematical truth of the law as stated, it is doubted whether the law applies to the question in hand. The assumption of the "equal technological state" of the various constituents of an ore is erroneous.

and the application of the law in some instances certainly gives results that cannot be accepted. For example, the comparative efficiency of the stamp and tube mill on Rand banket ore, when determined by the application of Kick's law to the grading analysis, is greatly in favor of the stamp, which does not accord with practical experience. This is due to the conglomerate nature of the Rand ore. Comparatively little energy is required to separate the pebbles from each other and to crush the cement to a fine mesh. All of this makes a good showing for the stamps, especially when crushing coarse. On the other hand, the tube mill has to deal with uniformly hard, coarse ore from which the softer and finer particles have been separated. It is, therefore, unfair to apply Kick's law to this problem.

* * *

On further consideration it will be found that many other ores are of such a nature that Kick's law is inapplicable, on account of the unequal technological state of the constituent minerals. Massive mixed sulphides present one crushing problem, which, however, changes as soon as the constituent minerals have been freed from each other. Similarly granite might come under the law until that point in its reduction is reached where we no longer have granite, but rather its constituents. Probably very few natural ores will be found of equal technological state, and the energy required to crush them cannot, therefore, be determined by the proposed method. It is for this reason, mainly, that metallurgists have not accepted wholly the work of recent investigators, although they recognize progress in the work thus far done.

Iron Ore and Pig Iron Production.

Recent forecasts made in some quarters that this season's movement of Lake Superior ore will fall below that of last year appear to be based upon the assumption that pig iron production in the twelvemonth beginning July 1 last will fall materially short of that in the preceding twelvemonth. That is an assumption which is hardly warranted. In round figures the production in the twelvemonth ended June 30 last was 30,000,000 tons. It is true that the rate of production has since dropped nearly 5,000,000 tons below, but the rate was higher at the end of the twelvemonth than at the beginning, and the maximum rate during the twelvemonth was reached in February, or later than the middle of the period. It is normal for the productive rate to fall off very materially in midsummer. Considering all circumstances it is as likely that production will be above as under 30,000,000 tons in the twelvemonths beginning with last July. Before entering into the arithmetic of the case, a brief résumé of the Lake Superior ore movement may be given. The total movement of Lake Superior ore has been as follows, the figures covering the movement down the lakes and the movement by all-rail routes:

	Gross tons.
1904	21,849,401
1905	34,384,116
1906	38,565,762
1907	42,266,668
1908	26,014,987
1909	42,586,869

The early predictions, made in the winter, when the iron industry was running at a considerably faster gait than the present, were that this season's movement would be about 50,000,000 tons, and more likely over than under that quantity. Lately there has been a reversal of opinion, some predictions recently made being that no more than 35,000,000 tons to 38,000,000 tons will be moved. En passant, it may be noted that if the movement is to be confined to so small a total, a drastic reduction in shipments must be made, for to Aug. 1 21,863,549 tons came down the lakes, and the August movement has been forecasted at 6,500,000 tons, and this represents a gain over 1909 of 5,800,000 tons, so that a mere continuance, after Sept. 1, of the 1909 rate would make a total for the season of about 48,400,000 tons.

* * *

The estimates of annual consumption of iron ore in the United States made by the Geological Survey show a total of 359,000,000 tons in the 10 years ended with 1908. The figures do not take account of furnace stocks, nor of some other details, and on account of the growth of the industry and industrial conditions at the beginning and end of the decennial period it is probably not unfair to assume that there was such an increase in furnace stocks as to reduce the actual consumption in the 10 years to about 350,000,000 tons. The pig iron production in the same period, as officially reported, was 185,632,841 gross tons, which gives a ratio of practically 1.9 tons of ore to 1 ton of pig iron. The ratio, as we shall have occasion to consider later, tends to increase, as the ores mined are steadily, although slowly, growing leaner year by year. In a year of full activity the production of ore outside the Lake Superior region amounts to about 10,000,000 tons. The Lake ores brought down in 1909 were substantially for the blast-furnace year July 1 to June 30. Then we have, roughly, for the year ended June 30 last the following supplies:

Lake ore shipments, 1909	42,586,869
Imports less exports to June 30	1,837,362
Production outside Lake Superior	10,000,000
Total	54,424,231

* * *

Allowing for a slight increase in the outside production we may take 55,000,000 tons. For the 30,000,000 tons of pig iron produced we should have required, at the ratio of 1.9, 57,000,000 tons, showing a deficiency of 2,000,000 tons. For a production of 30,000,000 tons of pig iron in the succeeding twelvemonth we should make up the deficiency and provide against another deficiency, by moving 4,000,000 tons more of Lake ore than in last season. Furthermore, as the iron content of ore is decreasing, the factor of 1.9 derived by the experience of the 10 years 1899 to 1908 is too low, and to cover that we should also move more ore. Finally, it seems to be taken for granted, in the retrospect, that the large imports of the past twelvemonth should have been discouraged as far as possible. Altogether, then, it would appear that to support 30,000,000 tons of pig iron production in the twelvemonth begun July 1 the Lake ore movement should be well in excess of the 42,500,000 tons moved last year, and a movement of between 48,000,000 tons and 50,000,000 tons would not be excessive. From the other viewpoint, a movement of only 35,000,000 tons of Lake ore this season would represent provision for the production of only 25,000,000 tons or 26,000,000 tons of pig iron in the twelvemonth begun July 1. It is utterly improbable that the requirements will be so light as this.

The Western Metallurgical Field.

International Smelting & Refining Company.

The lighting of the first fires in the furnaces of this company's new copper smelter in Utah occurred on July 25, 1910. The incident recalls the announcement of General Manager E. P. Mathewson in the first annual report of the company, to the effect that the plant would probably be ready for operation about Aug. 1, 1910. Following is a brief description of the plant as presented in the company's annual report by Mr. Mathewson, under date of June 1, 1910.

The company's railway, known as the Tooele Valley Railway, six and one-half miles in length, connects the smelter site known as International, with Tooele Junction on the San Pedro, Los Angeles and Salt Lake Railway. The plant consists of a sampling mill, capacity 4500 tons daily, fitted with two complete sets of Brunton sampling machinery and thoroughly equipped in every particular, driven by electric motors. In connection with the sampling mill are sampling ore bins of about 6000 tons capacity, from which the ore is taken to the sampling mill by belt conveyers. After sampling the ore is taken again by belt conveyers either to blast-furnace storage bins of about 3000 tons capacity, or to toaster storage bins of about 5000 tons capacity. From the blast-furnace storage bins the ore is to be removed by the local tramming system in electrically operated cars. From the roaster bins the ore will be removed by belt conveyers passing over a conveyer weighing scale, thence to roasters.

Each roaster building contains 16 roasters of the McDougall type, 32 in all, each with a capacity of 40 tons of dry ore in 24 hours, the smoke from the roasters passing directly into a large dust chamber fitted with hoppers and arranged for the introduction of the system of wire filters such as is in use at the Great Falls plant of the Anaconda Copper Mining Company, at the expense of merely hanging the wires. This dust chamber is connected by a large flue to a stack which is 350 ft. in height and 25 ft. inside diameter at the top. The calcined material from the roasters, together with flue dust from the dust chamber, is trammed to the top of the reverberatory building and discharged into hoppers directly over the reverberatory furnaces, and thence by gravity to the furnaces themselves.

The sample mill is fireproof throughout, having a steel frame and reinforced concrete floors and steel housings for the elevators. The bins are all of steel construction, but the approaches to the receiving bins are of timber frame. The roaster building is all steel with the exception of the roof, which is of wood covered with sheet iron. The office and shop buildings are of steel construction with concrete floors, but with wooden partitions with expanded metal lath on the office end, so that practically the entire plant is fireproof with the exception of the office partitions and part of the roof of the roaster building and sampling mill.

The reverberatory plant consists of five furnaces of the Anaconda type, measuring 19 ft. x 102 ft. on the hearth, capacity 275 tons of calcined material each per day. The reverberatories have four waste heat boilers attached to them, and in addition thereto three hand-fired boilers for spares. These boilers will furnish steam to the power plant which is a brick and steel structure fitted up with two triple expansion vertical engines, each directly attached to 700-kw. alternating-current generators. There are also two horizontal compound engines, two blowing engines for converter blast and two 90-lb. compressors for general utility. In addition there are the usual accompaniments of condensers, feed-water heaters, pumps for feed water and fire protection, and space has been left for the installation of blast-furnace blowers should a blast furnace be decided on later. The building is fireproof throughout, and fitted with an overhead crane electrically operated.

The converter building adjoins the reverberatory building and is a fireproof structure of steel with a clay floor. The converter stands are five in number, manufactured by the Power & Mining Machinery Company, and are electrically operated. There is a compressed air-tamping plant at one end of the converter building, and a 60-ton crane is installed in the main aisle for the moving of heavy material, slags, etc. The matte from the reverberatories will be tapped directly into the converters, and slag from the converters will be taken in large ladles by cross cranes to the reverberatory furnaces and poured in hot. This will make a minimum amount of cold slag to be handled. Copper from the converters will be transferred in ladles to the copper cake molds in the casting shed adjoining, and at first poured from the ladles into the molds and then loaded on cars for shipment. When business increases it will be necessary to put in a casting machine, space for which has been left.

The plant will receive ore from the Utah Consolidated mines over a cableway which is now in operation with a capacity of 100 tons per hour. Recently a contract also has been made to treat ore from the South Utah Company's mines. Mr. Mathewson closes his report with the statement that there is no better constructed or designed smelting plant for copper ores in existence.

Improvements at Cananea.

The cost of copper production at the reduction works of the Cananea Consolidated during 1909, including every expenditure, was 11.64 cents per pound, according to the report of General Manager L. D. Ricketts. This cost has been abnormally high, but improvements have been made which are expected to reduce this figure. Conservation of the water supply has been effected by impounding tailing, and using the clarified water in the mill. Loss in concentration is being reduced by the introduction of Cole drag classifiers which, it is estimated, will raise the recovery to 80 per cent. Considering the grade of material concentrated, this will be very good work. It is expected that all of the improvements and changes in the mill will enable the company to treat 25 per cent more ore than formerly.

The operations of the reduction division have been exceedingly satisfactory, both regarding metallurgical results and costs. There has been a reduction in the cost of beneficiating ores and concentrates, and this reduction will continue. The cost per dry ton of new copper bearing material treated for the past several periods has been as follows:

Year.	Total cost.
1907	$6.82
1908	3.86
1909	3.09

This reduction is largely due to increased efficiency of the reverberatory furnace and to decreased cost of power, and to the fact that barren material has been replaced by marginal material in the converting department. Concentrator slime has gradually replaced barren clay, and carefully selected ore has been mixed with barren quartz in the linings, until near the end of 1909 nothing but ore and slime were used. Finally, it was discovered that American ore was sufficient as a lining without any clay, and at present this ore is being used exclusively as converter lining without mixture of clay, either in the bodies or the lids of the converters, and while the combined silica and alumina amounts to but little over 70 per cent, as compared with 89 per cent in the barren quartz, greater efficiency is obtained, pound for pound, with the former than with the latter lining. This is largely due to the fact that the ore forms a more compact lining and withstands mechanical erosion. Necessarily the slag has become more basic, and now contains 62 per cent ferrous oxide. As a consequence, the converter slag has become a very valuable flux and has very largely replaced the barren iron and lime formerly used.

Experiments are being conducted to determine stack losses, which have been found to be serious. The best method to be used to save the minute percentage of dust in the enormous tonnage of stack gases is hard to determine, but preventive measures are to be introduced first. In line with this idea, a second reverberatory furnace with accessory battery of boilers,

and six additional calcining furnaces, will be installed. This will remove much of the fine material from blast-furnace treatment and largely decrease the quantity of dust made. At the same time, the fine material will be smelted more cheaply and the blast furnace capacity will be increased; directly by taking the fine material from them, and indirectly by increasing the capacity per square foot of hearth area. By this arrangement the smelting capacity will be nearly double the tonnage handled in 1909.

The use of fuel oil for power has proved of great advantage. Fuel bills per ton of ore have been cut almost in two despite the fact that more power per ton of ore is used than formerly. It is expected that this year will see the complete abandonment of all steam plants except those at the power house and concentrators. Power costs have dropped during the past two years from over $16 to under $8 per hp-month, and it is expected that as low as $6 power ultimately will be obtained.

When the smelter and concentrator enlargements now under way are completed, the plant will have a monthly capacity of 6,000,000 lb. of copper, if run on the present grade of ore.

Ore Dressing in the Cobalt District.

About a year ago the Temiskaming Mining Company, of Cobalt, Canada, authorized the erection of a concentrating mill. Following is a brief description of the system installed, as given by General Manager Norman R. Fisher. The low-grade ore is crushed in a No. 5 gyratory crusher and 36-in. x 14-in. rolls to pass a ¾-mesh screen, and transferred to storage bins. The first recovery is made in jigs of the Richards pulsator type, three of these machines being installed to handle material down to 14 mesh. All material finer than this is treated on James tables and slimers. It is found that the fine material is very rich owing to the occurrence of a large quantity of friable silver minerals, such as proustite, tetrahedrite and argentite, which, with fine leaf silver, is released from the gangue in crushing. It is estimated that the jig and table section of the mill will effect the recovery of about 80 per cent of the total extraction, or about 68 per cent of the total silver content of the ore.

Up to this stage the capacity of the mill is about 250 tons per 24 hours, but this section will be operated only during the day shift until such time as it is found expedient to increase the capacity of the stamp section.

The tailing from the jigs is dewatered and delivered into the stamp bins of 350 tons capacity. Six 1250 lb. stamps designed for rapid crushing will handle the jig tailing, which is then classified, the coarse material passing to James tables and the fine to Callow thickening tanks preparatory to treatment on slime tables. The capacity of the stamp section will be 80 to 90 tons per 24 hours.

Provision is also made for the installation of regrinding machinery to treat the table tailing if found advisable, and all tailings are to be conserved with a view to future treatment by lixiviation should it be found the mechanical loss renders this an economic necessity. It is expected that the mill will effect a saving of 85 per cent of the silver content of the ore, at an estimated average cost of treatment, including maintenance and power, of about $1.75 per ton.

The Non-Ferrous Metal Market.

The matter of greatest interest in the metal market for the past few weeks has been the proposed curtailment of copper output for the purpose of reducing the stock on hand and allowing the market prices to regain some degree of firmness. It may be said that the Phelps Dodge and the Calumet & Hecla interests have reduced output, and that Amalgamated and the porphyry coppers will undoubtedly do the same. The July output of the refineries has already indicated the curtailment, and it is expected that the next three months will witness a return to a more stable equilibrium between supply and demand. With one or two exceptions in the minor metals, the market has been characterized by general dullness.

Copper.—The reported curtailment has had a noticeable effect on the consumers, some of whom have bought heavily for future delivery as well as for immediate use. Prices have remained fairly steady at 12⅝ cents for lake and 12½ cents for electrolytic. There is a tendency to higher prices.

Lead.—The market has been quiet and without feature. Few transactions have been noted, the dullness having extended throughout the month. The St. Louis quotation is 4.27½ to 4.30 cents; New York, 4.40 cents.

Zinc.—Both demand and supply have been noticeably small, but prices have been generally firm. Consumers are not using more than their immediate demands require and no change is looked for until consumption increases. St. Louis quotations are 5 to 5.02½ cents; New York, 5.15 to 5.17 cents.

Tin.—Metal for immediate delivery has commanded a premium over future, which is a rather unusual condition. The New York market has been firm at 33¾ cents.

Other Metals.—The market for aluminium has been quiet with demand falling off somewhat. No. 1 ingots command 22.5 cents per pound. Prices on antimony have been nominal with practically no demand for the metal. Various brands are quoted at 7.5 to 8.2 cents per pound. The price for quicksilver also has fallen off and the metal can now be obtained for $46 per flask of 75 lb. on large orders. Nickel is quoted at 40 to 45 cents for large contracts, with higher prices for small deliveries.

The Iron and Steel Market.

The reduction in output during July and August has been less than was expected, the rate of pig iron production averaging well in excess of 25,000,000 tons a year during the two months as contrasted with an average rate of 30,000,000 tons during the preceding 12 months, and a maximum rate of 31,000,000 tons, reached in February last. That the output was absorbed can hardly be doubted and in such a quiet market the performance was well nigh phenomenal.

The trade reached July 1 with order books almost depleted in several lines, and production in the past two months has been sustained largely by specifications upon old contracts and by new purchases. As the buying trade had accumulated large stocks in the winter and in the second quarter had been making every effort to reduce them, it is certain that the engagements in July and August were only for absolute requirements, and probably represented less than the actual rate of consumption. The situation as to stocks must be well liquidated, and the inference is reasonable that buying will improve materially in September and October. A start in this direction appears to have already been made, for representative steel interests report that their August bookings have well exceeded those for July. While it is true that July was a poor month in this respect, since so many buyers were curtailing shipments on account of inventories and closing for repairs, still the showing is favorable.

In pig iron it is not so certain the position as to stocks has been liquidated. It is possible, but by no means certain, that consumers have reduced stocks enough to balance the increase which has occurred since Jan. 1 in furnace stocks. This, of course, refers only to merchant iron, less than one-third the total output. The steel works are carrying less pig iron than three months or six months ago. The total of merchant pig iron stocks is of less consequence than the manner in which they are carried, and as to this there is some unfavorable news from the Central West, it being reported upon good authority that an important aggregate of merchant pig iron has been carried by middlemen and furnaces upon bank accommodations which are being curtailed, in some cases forcing sales at prices which are based upon exigency rather than upon cost of production.

Current buying of steel products is largely by representatives of small consumers. Structural bookings have been relatively light in the past two months, in bridge, viaduct and large building work. The railroads have been particularly light buyers. It is believed that they are "playing politics" and will not loosen the pursestrings until after the November elections.

However, it is now understood that the larger steel-car plants have steel car orders on books to carry them into December, and it must be remembered that with the new system of accounting required since July 1, 1907, by the Interstate Commerce Commission the railroads are carrying car depreciation accounts which will, irrespective of net earnings, provide large sums for the purchase of new steel cars as old wooden cars wear out.

The Amalgamated Association has declared off its strike inaugurated 14 months ago against the open-shop policy of the American Sheet & Tin Plate Company, subsidiary of the Steel Corporation. This does not affect the sheet situation materially, since the company was already operating as many sheet mills as needed, but it will effect a considerable increase of output of the tinplate, deliveries of which were much behind. The company is starting two tin-plate plants which were entirely idle on account of the strike and is recruiting forces at other mills which were short handed. This will probably increase the company's tin-plate output 30 or 35 per cent in September as compared with August, equivalent to about 20 per cent of the country's total output and comes opportune, as the tin-plate requirements of September are heavy on account of winding up the canning season.

Pig Iron.

At the beginning of August the pig iron markets were stagnant, but toward the close of the month buyers began to show considerably more interest. The usual small enquiry broadened out, and a few large enquirers for extended delivery appeared. There have been slight declines since our last report in nearly all prices, the most extensive being in the valley market, in which Bessemer has dropped 25 cents, basic 75 cents and foundry 50 cents. Southern has been slightly firmer in the past fortnight than it was on Aug. 1, although not quotably higher.

There are only one or two producers in the Birmingham district willing to do $11 at furnace, and that only for early delivery, other quotations being at $11.50. We quote prices f.o.b. valley furnaces as follows, freight to Pittsburgh being 90 cents: Bessemer, $15.25; basic, $13.75 to $14; No. 2 foundry, $14 to $14.25; forge, $13.35, malleable, $14.50.

Steel.

The actual market movement in unfinished steel has been light. Prices have sagged slightly on Bessemer, while rods have declined fully a dollar a ton, owing probably to the advent of new capacity. We quote, f.o.b. Pittsburgh or Youngstown mills: Bessemer billets, $24.50; sheet bars, $25.50 to $26; open-hearth billets, $26; sheet bars, $26.50 to $27; rods, $28 to $29.

Finished Products.

Early in August steel bars dropped definitely $1 a ton, on attractive business, selling at $1.40 per 100 lb., but $1.45 has been obtained on odd lots for early delivery, as the bar mills are still comfortably filled with specifications. The shading of nominal sheet prices has increased slightly as to galvanized, reaching $5 per ton and making the regular market on 28-gage galvanized $2.25 per 100 lb., while black sheets are occasionally shaded more than $3 a ton, which was possible a month ago. The tonnage of sheet business is large, but insufficient to engage fully the greatly increased capacity. Tin plate continues in good demand and all available capacity is in requisition. Plates and shapes, although in indifferent demand compared with capacity, are fairly well held by the large mills at the former minimum of $1.40.

Subject to shading on flat and corrugated sheets of $3 a ton for black, $5 a ton for galvanized and $1 a ton for blue annealed, we quote finished steel products as follows, f.o.b. Pittsburgh, except where otherwise stated:

Standard steel rails, $28 for Bessemer, $30 for open-hearth, f.o.b. mill, except Colorado.

Plates, $1.40 for tank quality, ¼ in. and heavier.

Shapes, $1.40 for beams and channels, 15 in. and under, zees and angles 2 x 3 and larger.

Steel bars, $1.40 to $1.45, base; iron bars, $1.40 to $1.45, Pittsburgh; $1.37½ to $1.40, Chicago; $1.37½ to $1.45, Philadelphia.

Wire nails, $1.70, base; plain wire, $1.50, base; painted barb wire, $1.70; galvanized, $2.

Black sheets, 28-gage, $2.40; galvanized, $3.50; painted corrugated roofing, $1.70 per square; galvanized, $3; blue annealed, 10-gage, $1.75.

Tin plates, $3.60 for 100-lb. cokes.

Electric Iron Ore Reduction in Shasta County, California

We have repeatedly referred in our columns (see especially the papers of Lyon and Bennie on pages 251 and 252 of our Vol. VII) to the pioneer work of the Noble Electric Steel Company at Héroult-on-the-Pit, Shasta County, Cal., with the electric shaft furnace for iron ore reduction. Credit for this work is chiefly due to Mr. H. H. Noble, president of the Noble Electric Steel Company, and to Prof. D. A. Lyon, who designed the electric reduction furnace and is in charge of its operation. In the early experiments Dr. Héroult, the distinguished electric-steel engineer, also participated. The place of the smelter has been called after him.

It is now reported that the progress of the work is quite satisfactory and that the operation is now passing the experimental to the commercial stage.

On July 21 a party of 22 steel men, engineers, bankers and capitalists came to Héroult from San Francisco as the guests of Mr. H. H. Noble, to inspect this first electric pig-iron plant in America. The party included Mr. S. T. Wellman, of the Wellman-Seaver-Morgan Company; Mr. Patrick Noble, of the Pacific Rolling Mills, of San Francisco; Mr. F. E. Neitzel, of the Union Iron Works, of San Francisco; Mr. S. P. Mooney, of John A. Roebling's Sons Company; Mr. Richard B. Carr, of the Carnegie Steel Company; Mr. Chas. M. Gunn, of the Columbia Steel Company, of Portland and San Francisco; Mr. Edward J. Schneider, of the American Bridge Company; Mr. R. S. Moore, of the Moore & Scott Iron Works, of San Francisco; Prof. Edward O'Neill, of the University of California, and Messrs. J. H. Meyer, A. P. Giannini, R. J. Tyson, H. D. Walker, E. E. Mead, W. A. Halstead, C. B. Morgan, A. S. Carman, C. R. Downs, G. K. Weeks, and Dr. Ferdinand Stable.

The visitors arrived at noon just in time to see the electric furnace tapped, the amount of pig iron being five tons. In the evening the furnace was tapped again. It was stated that the furnace had been running without a hitch for 11 days and that it would be kept running as long as the California Power Company could supply the required power. The furnace is being tapped every six hours. At the day of the visit there were 150 tons of pig iron piled up in the smelter yards.

The furnace has proven so successful that the erection of five more electric furnaces has been decided upon and the five transformers for the same have already been ordered.

Charcoal is used as reducing agent in the iron ore reduction and the buildings occupied by the charcoal and by-product plant are as large as the smelter itself. Mr. H. M. Soper is in charge of the same. Wood alcohol, acetic acid, tar and creosote oil are obtained as by-products of the charcoal manufacture. The Mammoth Copper Company at Kennett has now a lot of mining timber at Heroult to be treated with creosote oil as a preservative.

Four lime kilns are completed. Two are running and two more are ready to start up.

The Noble Electric Steel Company, and its president, Mr. H. H. Noble, must be congratulated on the efficient and successful way in which they are progressing in their pioneer work.

Chicago Meeting of American Electrochemical Society.

Headquarters of the Chicago meeting, to be held on October 13, 14, 15 (Thursday to Saturday), will be at the La Salle Hotel, where, in the assembly rooms on the top floor of the hotel, sessions for the reading and discussion of papers will be held on Thursday and Friday morning. A third professional session will be held on Saturday morning at the University of Chicago.

For the afternoons various excursions will be arranged, including visits to the famous steel plants of the U. S. Steel Corporation at Gary, Ind., and at South Chicago, where a 15-ton electric Héroult furnace is in operation, and a visit to the Buffington Cement Works. On Thursday evening a subscription dinner will be held and on Friday evening a smoker.

CORRESPONDENCE.

Checker Work.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—In the report in your August issue on the Fifth International Congress for Mining, Metallurgy, Applied Mechanics and Practical Geology, recently held at Düsseldorf, an account is given of the paper of Dipl. Ing. O. Friedrich on "Improvements in the Construction of Siemens-Martin Furnaces" in which he refers to the various forms of checker work which are now finding favor in Europe for use in regenerators.

We believe it will be of interest to many metallurgists in this country to know that we have recently concluded with the owners of the Scherfenberg patent a contract for the exclusive manufacture and sale in this country of the bricks illustrated in Figs. 16, 17 and 18 in your article.

These forms of checker work have very many advantages over ordinary checker brick—the heating surface is increased, there are no flat surfaces upon which the slag can deposit and finally close the gas passages, the distribution of the gases is perfectly regular and the checker work is easy to erect and perfectly rigid after erection.

For use in sulphuric acid towers and for other chemical purposes we have already supplied large quantities of these bricks, made from acid-proof stoneware, and among the many other uses for which they are well adapted, the carburetion of water-gas offers a very promising field.

New York City. THE DIDIER-MARCH COMPANY.

Defects in Refractory Brick.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—In METALLURGICAL AND CHEMICAL ENGINEERING for May, 1910, appears an article by Mr. Gilbert Rigg, of the New Jersey Zinc Company, pointing out some defects in refractories. The defects in manufacture mentioned are:

1. Insufficient purification of material.
2. Superannuated methods of manufacture.
3. Burning at too low temperature.

I would like to add to the above list of faults in manufacture the failure of the manufacturer to provide proper shapes in molding. It is common practice to cut or grind the brick into proper shape after burning. This is, in most cases, a mistake. A brick that is perfect can be ground to almost any shape with little or no damage, but can never be cut with a brick hammer or chisel without suffering in quality. This is due to what stone cutters call "opening of the material" by the jarring attendant on the cutting. But only a perfect brick can be ground without damage, and since perfection is a quality seldom attained it is better to make the necessary shapes and avoid grinding. The reason for this is that the surface of a brick as made in the mold is usually a better surface than can be attained after this has been removed. The pressure used in making the brick makes the surface less porous and harder than any other part of the brick. This surface is sometimes destroyed by improper burning or sudden cooling.

Another defect in making refractories consists in forcing the clay out of the machine in such a way as to cause little cracks in the corners. The bricks are afterward pressed to give them a perfect shape, and at the same time these cracks disappear; but they reappear when the bricks are used as soon as hot slag comes in contact with the brick. A number of other defects might be mentioned, but I wish to say a word for the manufacturer.

The consumer who buys "firebrick" in the open market must not be disappointed if he gets just "firebrick"; again, the consumer who orders brick of a certain chemical composition ought to be satisfied when he gets what he ordered; and, finally, when a consumer orders a first-class refractory he ought to be willing to pay for it.

Going back to objection No. 1 in Mr. Rigg's paper, "insufficient purification of material," I would change it to "improper preparation of material," and would call attention to the fact that in order to get the best results a year or more is required in the preparation of the clay, including handling it a number of times. This means a larger plant, more working capital, and consequently added cost. The consumer too often expects goods made in this manner at the same price as those which are made from clay on the same day that it is taken from the bed where nature deposited it.

Mr. Rigg's second objection is "superannuated methods of manufacture." The writer recently visited a plant where good refractories were made, as refractories go, and was surprised to see the methods, which, putting it mildly, appeared to be not the best. When the manager was asked about it he replied that the lot of brick then being made was for a firm whose custom was very desirable, and that the profit on the whole order was negligible. "But," he added, "we hope to hold their business long enough to educate them to the value of good refractories."

This is too commonly the case. The manufacturer may give good value for the money and still not give satisfaction. When the consumer and the manufacturer get together with a perfect understanding of the needs of the one and the difficulties of the other Mr. Rigg's severe indictment will be only an interesting bit of the history of refractories.

Golden, Col. JOHN C. BAILAR.

The Western Tour of the American Chemical Society.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—Our voyage to San Francisco on the American Chemical Society's special train was a wonder trip. Ever since we left Chicago we had not a single dull day. Our train was composed of a buffet and baggage car, a diner, nine sleeping cars, and an observation car. The Santa Fé Railroad deserves our unreserved thanks. Never have I traveled on any railroad as comfortably as on the Santa Fé.

We were 110. At Chicago lunch and refreshments were offered by the Chicago Section. First stop was made at Kansas City, Mo., where some members came on. Then followed our ride through the long Kansas prairie with its agricultural richness.

The sightseeing began with Colorado Springs, Pike's Peak, Manitou, Garden of the Gods, etc. Then we passed through the desert of New Mexico, a short stop being made at Albuquerque. Indian settlements were seen everywhere with their picturesque costumes and adobe houses. The excellently oiled roadbed of the train and the fact that our locomotive was

oil-cured made it possible to remain on the balcony of the observation car almost during every stage of the trip, even in the alkali desert. It was fearfully hot, 110-120° sometimes, but the air was so dry that the heat did not incommodate us and did not even soften our collars.

The petrified forest, to which we were taken from Adamana in carriages and wagons (a distance of some 12 miles), gave us a good idea of the bleak Arizona desert. We were carted over the open, dusty prairie in the blazing sun. The samples of the petrified trees which we picked up blistered our fingers, and when after our long ride from the wonders we had seen we came back to our comfortable train everybody made a dash for the buffet car and the negro porter had a hustling time to supply drinks to our parched throats.

Next morning we awakened in a fine forest of pine trees and we were at the Grand Canyon, the most imposing spectacle I have ever seen. Some of us took horses and through a narrow trail along precipices followed the guide who led our party deeper and deeper in the hot canyon. As I am writing now I am still suffering (after about two weeks) from blisters on my unprotected sunburned hands. My wife made the trip, too, and bravely enjoyed the whole performance. What a luxury it was to have a bath in the hotel after our return from a day in the saddle.

If the petrified forest was hot, this was hotter. But the precipices make you forget the heat when your mule walks on the edge of the trail and there are a few thousand feet just below you. The clerk at the hotel described the canyon as follows: "Between hell and the canyon trail there is but a screen door."

The same night we proceeded toward California. Beautiful Redlands, more beautiful Riverside with its never-to-be-forgotten orange groves, its magnificent streets, its picturesque and comfortable Mission Hotel, giving me for the first time an instance of a city reclaimed from the desert by irrigation and transformed into a triumph of municipal æsthetics.

For the first time in my life I saw an American city which looks finished, and beautifully finished at that. From the moment you enter Riverside until you leave it, everything seems harmony and good taste and displays and proclaims high aspirations of its citizens.

Our party had already been met at the Grand Canyon by Mr Ralph A. Gould, chairman of the local committee, welcoming us to the State of California, and just before reaching Riverside a committee of chemists from Los Angeles met our train. On the following day at Los Angeles our fellow chemists had prepared a full day of excursions and entertainments and began to give us a foretaste of the reception which was awaiting us in 'Frisco.

During the night we proceeded on our way, and early in the morning were awakened in the desert near Lang, where automobiles and a little improvised freight train took us to the borax mines, several miles between cacti and yucca plants intermingled with dried-out sagebrush and covered with alkali dust. At the borax mines of the Sterling Borax Company a big sign, "Welcome American Chemical Society," greeted us in the glistening sun. But abundance of beer cooled with ice imported from our train made us soon forget the temperature.

Strange to say, although the temperature may be as high as 110° F., it always feels very comfortable in the shade.

We left at noon never imagining that the next morning our nice comfortable train was going to be reduced to a horribly deformed mass of débris. I still wonder how we escaped. The car in which Dr Bancroft and Mrs. Bancroft, Dr. Parsons and Mrs. Parsons, Mr. Adamson and family, Dr Baekeland and Mrs. Baekeland, Dr. Love, Dr. Meyeda and wife were traveling was lifted up in the air, the front trucks wrenched off and tossed ahead over the wreck, so that its front end stood about 50 ft high in the air, ready to topple over in the ravine, but held back I do not know how. One car—the dining car—was shot ahead bodily over the upturned locomotive and the buffet car was shot into the sand bank and was twisted like a piece of cardboard, while the splinters and wreckage were shot high up into the barren hill. Our baggage was lost in the wreckage, and several of the party were digging lustily away in the sand bank.

The locomotive, with its wheel in the air, was buried with its front part into the sand, and the emptying oil tanks made a river of oil, the rush of which made me first suppose that our car had rolled into the stream. Afterward the smell of the oil made us think the gas pipes had burst. All this, of course, while we were still prisoners in our car and before we had succeeded in breaking out. The engineer, whose reckless driving around the mountain curve had caused the accident, was pinned underneath, together with the fireman, both dead. Dr. Whitney managed to rescue the conductor, who was in a much disturbed condition and was buried in one of the wrecked cars. The crew, conductors, brakemen, porters and everybody connected with the train behaved admirably; nobody seemed unduly excited and everybody did the right thing at the right time and tried to help others. Several were injured, but made slight of their wounds or bruises.

The ladies of our party, too, were very self-possessed, although most of them had scant time to dress. There was not a complaint, not a sign of emotional weakness in our whole party. The last seven sleeping cars had hardly been damaged, and some had not even left the track. The landscape where all this occurred was one of those large California ranches with the Sierra in the distance, and our train left the track at a curve around a hill known as Rocky Point, near the Salinas River, half way between King City and Metz.

The news soon became known by telephone. Cowboys came galloping over the plain. Mexican Indians and Japanese came from the ranch. Then from the next city came automobiles rushing at full speed between cacti and sagebrush. Some thoughtful people brought basketsful of excellent fruit. Surgeons and nurses arrived in automobiles from Salinas, and before long a rescue train arrived to enable us to continue our voyage.

A wreck train and crew started immediately to work—all Mexican Indians and Japs—to build a new track and remove the wreck, and I heard the foreman telephone that the track would be clear within six hours.

As to our reception in San Francisco I shall not attempt to describe it. A splendidly managed hotel is the St. Francis, where we all were housed and where all meetings took place, and from the moment we arrived until we left, our generous, cheerful California friends heaped festivities and entertainments upon us without intervention.

We in the East cannot realize what an enormous country we live in, and anybody who has not visited the Pacific Coast cannot have the proper conception of the United States. And what big-hearted, enthusiastic fellows one meets here; no weaklings, no drones, no pessimists here. Whoever beholds the new San Francisco rebuilt in no time after a most disastrous calamity which would have discouraged any city in the Old World can understand the latent power, the constructive genius, the executive abilities of these empire builders of the West.

Everything seems big, generous here. From the giant trees in the forests and the endless wheat fields and orchards to the very men and women. And then our trip to Portland and Seattle, through those virgin forests, with trees towering as high as church spires, rushing foamy torrents, snowclad mountains, and now over Vancouver and the Canadian Pacific in the heart of the Canadian Rockies, mountains and glaciers and forests which are large enough in size and in extent to build several Switzerlands and several Tyrols, endless unsettled portions of the beautiful earth eagerly awaiting sturdy men to take advantage of its bountiful resources, while our race is degenerating in congested, vile cities.

Glacier House, X Y Z

Glacier, B. C., Canada

The Effect on the Solubility of Gold When Ore Is Crushed Between Iron Surfaces.

BY J. M. TIPPETT.

There are several types of machines commonly used for comminuting ores, each of which has its friends, some favoring the stamp, others the chilian mill, and still others the tube-mill. A combination of the stamp and tubemill is frequently used, or of chilian mill and tubemill. Regardless of which combination is used, the main idea is to comminute the ore to a desired degree of fineness at the lowest possible cost.

A general impression has prevailed that grinding to a certain degree of fineness is all that is necessary in order to obtain the best extraction in cyanide solutions, but, speaking from an experimental point of view, I have found that fine grinding by attrition between two iron surfaces, such as the bucking-board and muller or the coffee-mill grinder, gives unsatisfactory results, and is apt to mislead one experimenting on an ore which would give better extraction results if it was ground to the same degree of fineness between flint surfaces.

When experimenting on the Portland Gold Mining Company's chlorination tailing dump in 1905 I found that grinding this material between two iron surfaces put the gold in such a condition that it was not readily amenable to solution in potassium cyanide. The dump material was 14-mesh and contained an average gold value of $1.60 per ton.

Leaching tests were first tried on lots ranging from 2 lb. to 2 tons. This soon proved unsatisfactory owing to the fact that much of the gold was encased in the coarser particles of the tailing, and the extraction never exceeded 50 per cent. Crushing to 20-mesh and 30-mesh gave no better results. Grinding in an iron mortar with cyanide solution, or grinding on a bucking-board so that all would pass 200-mesh, using a strong cyanide solution and long agitation, also proved of no avail. At the time this seemed rather strange considering the extreme fineness to which the ore had been ground. No better extraction was gained by grinding the tailing to 200-mesh than by treatment at 14-mesh just as it came off the dump.

Finally a 16-gal. beer barrel was fitted up as a tubemill with a central shaft, at one end of which was a pulley belted to the main lineshaft in the concentrator. This crude device was filled half full of pebbles, 50 lb. of tailings and sufficient cyanide solution and allowed to grind for several hours. Immediately the extractions rose to 70 per cent and 80 per cent.

The conclusion arrived at regarding the poor extraction obtained when the tailing was ground between iron faces was that the gold particles possibly became coated or glazed with a film of metallic iron, which made it impossible for even strong solutions of potassium cyanide to attack them.

At about the same time Mr. Herbert W. Fox, superintendent of the United States Reduction & Refining Company's works at Colorado City, was experimenting on their tailing dump, which contained the same class of material as our own, with the result that their cyanide plant was built and put into commission a short time before the Portland company's mill was completed. Both plants are using tubemills with silex lining for grinding the tailings treated in the cyanide plants.

To further investigate whether comminuting ore by means of attrition between two iron faces really had an effect on the solubility of the gold contained, a large quantity of $20 roasted ore was subjected to a series of tests. The extraction on this ore, ground to 14-mesh, by treatment in a chlorination barrel with a normal charge of chemicals (10 lb. of bleaching powder containing 35 per cent available chlorine and 27 lb. of sulphuric acid per ton of ore) was 90 per cent to 92 per cent. The same ore ground in a silex-lined tubemill in cyanide solution gave an extraction of 94 per cent to 96 per cent.

Portions of this same ore which gave the above extractions were then taken and subjected to a series of tests, after having been ground to 100-mesh, as follows: for the first series the portion was ground in a coffee-mill grinder; for the second, on a bucking-board, while the third was ground in a small tube-mill. Each portion was then panned and the concentrate eliminated, thus reducing the gold content of the portion to be treated, as shown by the assay of the heads in each series. Taking out the coarse gold particles by concentration after grinding and before the application of chemicals should have helped the extraction, but as shown in the series in which the ore was ground between iron faces, it did not aid extraction and if the coarse gold had been left in the ore the extraction would have been still poorer.

Series No. 1.

Ore ground in a coffee-mill grinder. Head after panning assayed 0.68 oz. gold.

(a) Four hours' treatment by agitation in 0.2 per cent chlorine water solution made acid with 5 lb. sulphuric acid per ton of ore; proportion of solution to ore, 2:1, or 8 lb. chlorine per ton of ore; after treatment the solution contained 0.07 per cent chlorine, showing a consumption of 5.2 lb. of chlorine per ton of ore. Tailing assayed 0.52 oz. gold. Extraction, 23.5 per cent.

(b) Conditions the same as in (a) except 10 lb. of sulphuric acid were used instead of 5 lb. After treatment the solution contained 0.05 per cent chlorine, showing consumption of 6 lb. chlorine per ton of ore. Tailings assayed 0.35 oz. gold. Extraction, 48.5 per cent.

(c) Conditions same as in chlorination barrel on the basis of 10 lb. of bleaching powder containing 35 per cent available chlorine, and 27 lb. sulphuric acid per ton of ore. Agitation for four hours. After treatment there were 1.06 lb. of available chlorine per ton of ore. Tailing assayed 0.61 oz. gold. Extraction, 1 per cent.

(d) Four hours' treatment by agitation in a 2-lb.-per-ton cyanide solution; proportion of solution to ore, 3:1; protective alkalinity, 0.35 lb. lime. After treatment the solution contained 1.8 lb. cyanide per ton. Tailing assayed 0.18 oz. gold. Extraction, 73.5 per cent.

(e) This was made an extreme case. Forty pounds of bleaching powder, containing 35 per cent available chlorine, and 110 lb. sulphuric acid were used per ton of ore. Agitation for four hours. Tailings assayed 0.04 oz. gold. Extraction, 94.1 per cent.

Cost of bleaching powder per ton of ore	$0.80
Cost of sulphuric acid per ton of ore	1.10
Cost per ton of ore for chemicals	$1.90

Series No. 2.

Ore ground on a bucking-board. Head after panning assayed 0.78 oz. gold.

(a) Four hours' treatment by agitation in a 0.285 per cent chlorine water solution made acid with 5 lb. of sulphuric acid per ton of ore. Proportion of solution to ore, 2:1. After treatment the solution contained 0.132 per cent chlorine, showing a consumption of 5.72 lb. chlorine per ton of ore. Tailing assayed 0.36 oz. gold. Extraction, 53.8 per cent.

(b) Condition the same as in (a) except that no sulphuric acid was used. After treatment the solution contained 0.158 per cent chlorine, showing a consumption of 4.68 lb. chlorine per ton of ore. Tailing assayed 0.44 oz. gold. Extraction, 43.5 per cent.

(c) Treatment the same as in chlorination barrels on the basis of 10 lb. of bleaching powder containing 35 per cent available chlorine and 27 lb. sulphuric acid per ton of ore. Agitation for four hours. After treatment the solution contained 0.061 per cent chlorine, showing 1.22 lb. chlorine per ton of ore still available. Tailing assayed 0.67 oz. gold. Extraction, 14.1 per cent.

(d) Four hours' treatment by agitation in a 4-lb.-per-ton cyanide solution; proportion of solution to ore, 3:1; protective alkalinity, 2 lb. lime. After treatment the solution con-

tained 3.5 lb. potassium cyanide per ton. Tailing assayed 0.12 oz. gold. Extraction, 84.6 per cent.

Series No. 3.

Ore ground in a small tubemill. Head after panning assayed 0.64 oz. gold.

(a) Four hours' treatment by agitation in a 0.153 per cent chlorine water solution made acid with 5 lb. sulphuric acid per ton of ore. Proportion of solution to ore, 2:1, or 6.12 lb. chlorine per ton of ore. After treatment the solution contained 0.09 per cent chlorine, showing a consumption of 2.52 lb. chlorine per ton of ore. Tailing assayed 0.04 oz. gold. Extraction, 93.7 per cent.

(b) Treatment the same as in chlorination barrels on a basis of 10 lb. of bleaching powder containing 35 per cent available chlorine, and 27 lb. sulphuric acid, per ton of ore. Agitation for four hours. Tailing assayed 0.07 oz. gold. Extraction, 89 per cent.

(c) Agitation for four hours in a 2-lb.-per-ton cyanide solution; proportion of solution to ore, 3:1; protective alkalinity, 0.2 lb. lime. After treatment the solution contained 1.8 lb. cyanide per ton. Tailing assayed 0.06 oz. gold. Extraction, 90.6 per cent.

While it is probable that comminution on a commercial scale in the Chilian or Huntington mill, or grinding pan, would not produce such a noticeable effect on the extraction as is shown in an experimental way, using the bucking-board or coffee-mill grinder, I am inclined to believe that grinding effected by attrition between two iron faces is likely to put gold in such a condition that it is not susceptible to the action of cyanide solutions, and that this condition is more marked than when grinding to the same degree of fineness between flint surfaces.

Portland Mill, Colorado Springs, Col.

The Fuel Requirement of the Rotary Cement Kiln.

By W. S. Landis, M. S.

The purpose of the present paper is to discuss from the theoretical and the practical standpoints the the fuel requirement of the rotary cement kiln, more particularly under the conditions existing in the Lehigh cement region. Only minor changes, however, are required to adapt the whole discussion to the operating conditions of any other region into which the rotary kiln has made its way.

In the Lehigh district cement rock (an argillaceous limestone) or cement rock and pure limestone are ground very fine and fed into the kiln dry, coming out in the form of a sintered mass technically called "clinker." Five hundred and sixty pounds of the raw material, called "mix," are fed to the kiln to make a barrel (the unit weight) of clinker, weighing 380 lb.

In the region in question the mix will average 33 per cent, or 188 pounds per barrel of clinker, of carbon dioxide, existing in the raw material as calcium carbonate. The only reactions taking place in the charge in the kiln are the driving off of the carbon dioxide and the sintering together of the rest of the ingredients to form the clinker The mix is artificially dried before fine griding, so that here we are not even concerned with the driving off of water from it. (Where clay is used in the charge the dehydration must be taken into account if done in the kiln.)

The calculation of the heat requirements of these reactions is quite as simple as the reactions themselves. The heat of decomposition of calcium carbonate is 1020 pound-calories per pound of carbon dioxide, so that to decompose the carbonate present requires $188 \times 1020 = 191{,}760$ pound-calories. The sintering of the residue of ingredients is really the chemical union of the silica, lime, and alumina, etc., giving out heat in the reaction.

By the combustion of a given weight of mix with a coal of known calorific power in a bomb calorimeter the author has evaluated the heat of formation of the clinker as 200 pound-calories per pound. This figure is not quite as accurate as it is hoped will eventually be attained, as not all the difficulties of performing this reaction perfectly in the bomb have successfully been overcome, but its limit of error is well within that of the more commonly accepted thermo-chemical data, and it will well serve the purpose of the present paper. The formation of clinker will then generate $380 \times 200 = 76{,}000$ pound-calories.

The net heat absorbed in the kiln per barrel of clinker, to be supplied from external source, is:

Heat of decomposition of carbonates	191,760 lb.-cals.
Heat of formation of clinker	76,000 "
Net heat absorbed	115,760 "

From a number of calorimeter determinations of the calorific powers of the long flame bituminous coals used in the Lehigh region for firing the kilns this figure was found to average 7400 pound-calories per pound of coal (practical or lower calorific power). Our kiln will therefore require a minimum of 15.7 pounds of coal to produce the reactions necessary for clinker formation irrespective of its kind and size.

It must be remembered, however, that the dissociation of calcium carbonate requires a temperature of 700° to 800° C., depending somewhat on the partial pressure of the carbon dioxide in the atmosphere of the kiln, and also that the formation of the clinker only takes place at temperatures between 1200° and 1300° C. Unfortunately, we have not up to the present attained that perfection of metallurgical apparatus so that we can maintain such temperatures in a continuously working apparatus without considerable losses of heat. Our figure for minimum coal requirement which we have just calculated, therefore, has but little significance to the kiln man operating under industrial conditions. The inevitable losses which occur in the production of such high temperatures on the large scale demanded bring up the fuel consumption to many times this theoretical figure.

Taking up these losses in detail in the light of a large number of heat balance sheets which the author has made on kilns varying from 60 to 135 feet long, operated under various superintendents, and always under actual working conditions, the following facts and conclusions have presented themselves:

In a kiln burning pulverized bituminous coal of 70 per cent total carbon and a mix containing 33 per cent carbon dioxide, perfect combustion should result in a chimney gas of the approximate composition (dry): carbon dioxide 36 per cent, nitrogen 64 per cent by volume. Both the coal and the mix being dried before grinding, the only moisture that would accompany this gas would be that from the combustion of the hydrogen in the coal. (This composition was calculated from the coal consumption of 65 pounds per barrel of clinker, which we will later arrive at.)

In no case has the actual chimney gas analyzed anything like this figure, when the samples were taken under the actual operating conditions. An average of these analyses (dry) taken on mills in various parts of the region was carbon dioxide 14.5 per cent, oxygen 0.5 per cent, nitrogen 76 per cent, showing the presence of large excess of air. This air gets into the kiln through at least two places: one intentionally through the nozzle feeding the coal into the kiln, the other through leaks around the lower end of the kiln and through various sight and poke holes invariably left open.

The air entering through the burner is under direct control of the operator; by means of it he regulates the length and character of the flame No attempt seems to be made to control the leakage, as very few of the plants are equipped for the analysis of stack gases and the interpretation of the results. The height and sizes of chimneys seem to have been a matter of guess; they are all too high, and poor regulation of drafts is gotten by opening a door leading to the outside at their base, so admitting cold air.

The enormous amount of heat carried out through the chimney of a cement kiln even with perfect combustion would cause one to suppose that in the present financial state of the industry this item was closely watched. Actual kilns burning from 80 to 100 pounds of coal per barrel of clinker, discharging the products of combustion into the stack at 500° C. (average of actually measured temperatures) lose up the chimney 130,000 pound-calories, for the lower fuel consumption with perfect combustion, to 275,000 pound-calories for the higher fuel consumption with excess air such as in the average analysis given above. These figures have been taken from balance sheets of actually operating kilns, the first figure from a 135-foot kiln using 84 pounds of coal and the second figure from a 60-foot kiln using 100 pounds of coal per barrel of clinker.

In terms of pounds of coal these figures represent 17 and 37 pounds respectively, a difference of 20 pounds chargeable to a lengthening of kiln and elimination of excess of air. The excess air in the shorter kiln carried out 110,000 pound-calories, leaving to the advantage of the longer kiln 35,000 pound-calories. In terms of pounds of coal the fuel consumption of the smaller kiln was 15 pounds too high because of excess air in products of combustion, and 5 pounds too high because of the kiln being too short for efficient heat regeneration.

Using the actual minimum fuel consumption we shall calculate later, perfect combustion without excess of air, and a gas temperature of 500° C., the chimney loss would be about 100,000 pound-calories, representing a coal consumption of 13.5 pounds for this item.

A curious coincidence in connection with the thermal study of the kilns as actually operated was the fact that the temperature of the gases leaving the kilns, irrespective of the fuel consumption and length of the kilns, was in all cases very close to 500° C. Actually on a 60-foot kiln 490°, 100-foot kiln 525°, and 125-foot kiln 510° C. At first glance it looks as if the length of the kilns had but little to do with the extraction of heat from the gases passing through them, but really such is not the case. This temperature is dependent not only on the extraction of heat from the gases, but also on the fuel consumption, amount of air used in combustion, and in radiation losses of the kiln. It was just a coincidence that in all cases studied these items always balanced so as to give nearly the same temperature.

Actual data has not been published as to the running of the new 260-foot kiln on the Hudson, but it has been hinted that the stack temperature is close to 300° C. It has been mathematically proved that the most economical temperature as far as the chimney is concerned is to have the gases enter at 300° C. There is nothing to be gained by introducing the gas into the chimney at 500° C. The heat in the gas equivalent to the difference in temperatures of 500° and 300° C. had better be used for drying purposes. With gases at 300° instead of 500° C. our coal requirement for the chimney could be cut down from 13.5 pounds to 8 pounds per barrel, of course, only if the heat equivalent to the difference in temperature was intercepted in the kiln.

The loss of heat in the hot clinker leaving the kiln is a large item. The clinkering temperature is absolutely fixed so that practically all kilns discharge clinker at the same temperature, 1050° C. for this region. The specific heat of clinker is $0.207 + 0.00001gt$. The heat in a barrel of clinker at 1050° C. is 90,500 pound-calories, equivalent to 10.5 pounds of coal.

Practically all American kilns discharge this hot clinker to a conveyor and run it to cooling towers, taking extra precautions to lose all the heat contained. Abroad it is run through an under-cooler, giving up two-thirds to one-half its heat to the air entering the kiln. Such regeneration easily results in a saving of from 5 to 7 pounds of coal, an item well worth considering where we are turning out millions of barrels of cement annually. The cost of an under-cooler is small in comparison with the saving that results from its use.

Finally we have to consider the item of radiation and conduction loss. Numerous measurements made of the temperature of the outside of kilns of all sizes have failed to lead to any concordant results as far as calculation of the radiation loss is concerned. It seems as if the complex nature of the surfaces in the average kiln-room which receive the radiant heat so complicate the calculation that up to the present a direct estimation of this loss is not possible.

One thing is certain, the surface of the kiln will not radiate more heat than it receives by conduction through the lining. If we can back up the shell with a suitable heat insulator and then inside of that put the alumina bricks, or other lining used, we are going to cut down the radiation losses of the kiln considerably. Magnesia or alumina bricks are too good heat conductors to rest directly against the shell of the kiln for economical operation. Years ago an asbestos layer was tried in just such a composite lining as advocated, but its yielding character precluded mechanical stability.

What is wanted is a more firm material of low heat conductivity, and it has occurred to the author that bricks of infusorial earth such as the "moler," which have been described repeatedly in this journal would well fill this qualification. Tests made in the author's laboratory have shown an abnormally low heat conductivity for these brick; they will stand up sufficiently rigid to temperatures of 700° to 800° C. The great shrinkage can be taken care of by burning them at a temperature slightly higher than they will afterwards be called upon to withstand. The regular refractory lining of alumina bricks can be laid right against the moler bricks. I firmly believe the present heat losses by radiation could be cut down one-half by the use of such a composite lining.

Some radiation and conduction losses taken from heat balance sheets of various kilns are here reproduced:

60-ft. kiln, old lining	268,000	lb.-cals.
100 " " new lining	176,000	"
135 " "	255,000	"

The figure on the 100-foot kiln is the lowest value ever found by the author for this loss, and represents the nearest approach to perfection in a single-material kiln lining. It corresponds to a coal consumption of 23.5 pounds. A composite lining as described should reduce the loss to 100,000 lb.-cals., a coal consumption of 13.5 pounds per barrel of clinker.

And now in conclusion, to sum up the coal consumption of our kilns, we should have, conditions as indicated:

Reactions	15.7	lbs.
Chimney loss, 500°, combustion perfect	13.5	"
Clinker loss, no regeneration	10.5	"
Radiation and conduction loss, ordinary lining	23.5	"
Total	63.2	"

These figures represent conditions as actually met with in one kiln or another, though not all on the same kiln at the same time. If, however, they can be attained individually, there is no reason why under intelligent supervision all the conditions which originally assured them should not be applied simultaneously to the same kiln. The nearest approach met by the author in practice to the attainment of this coal consumption was 71 pounds, though unfortunately in that kiln combustion was not quite complete and the lining was old and thin. The average consumptions varied from 85 pounds for a 135-foot kiln to 100 pounds and over for a 60-foot kiln.

If we go a step further and embody the reasonable possibilities mentioned in the body of the paper our coal consumption should be:

Reactions	15.7	lbs.
Chimney loss, 300°, perfect combustion	8.0	"
Clinker loss, half heat regenerated	6.0	"
Radiation and conduction loss, composite lining	13.5	"
Total	43.2	"

I expect to see in the near future in this region cement burned with less than 50 pounds of coal per barrel. Industrial

conditions are forcing the small concerns, who cannot afford the necessary experimental work on a large scale, out of business. The larger concerns have just awakened in the past year to the great saving to be made in the cost of manufacture by an intelligent supervision of the fuel consumption and kiln design. The rotary kiln is far from perfection and our foreign friends, realizing its great limitations, are even now at work on a new type of kiln which will bring the fuel consumption as low as 35 pounds per barrel.

Lehigh University, South Bethlehem, Pa.

Hardness Tests on Alloys of Nickel and Copper with Electrolytic Iron.—II.

By G. A. Rousn.

In a previous paper[1] the writer has discussed the effects of the addition of nickel and of copper to electrolytic iron, and the present paper is a continuation of this investigation, giving the effects of the addition of these elements together to iron, it being a part of the extended study of electrolytic iron alloys that is being carried out in the Chemical Engineering Laboratories of the University of Wisconsin, under grant from the Carnegie Institution.

In view of the results obtained from the nickel and copper series of alloys with electrolytic iron, it is of interest to note the results of combining both metals in one alloy. This particular part of the investigation is of interest in two ways: first, in view of the possibility of the hardening properties of the two metals being more or less additive so that the copper will increase the already high values obtained with the nickel alloys, and, second, if the copper does not actually detract too much from the value of the nickel additions, the probability of being able to substitute the new commercial alloy, Monel metal, in place of the more expensive nickel as the addition agent.

With this in view, alloys were made, in the same way as before, with from 2 per cent to 20 per cent of Monel metal and tested for hardness by means of the scleroscope. On account of the close agreement that was noted between the added quantities and the actual analysis in the previous series and the fact that Monel, being a commercial alloy, is subject to slight variation, it was not thought necessary to make any analyses of these alloys, but to simply record them according to the proportions of Monel that had been added to the melt. In the calculation of the ultimate composition of the alloys the average analysis of the Monel was taken as showing two-thirds nickel and one-fourth copper, the impurities not being taken into consideration since they could not be controlled.

The hardness curve for the forged samples rises rather

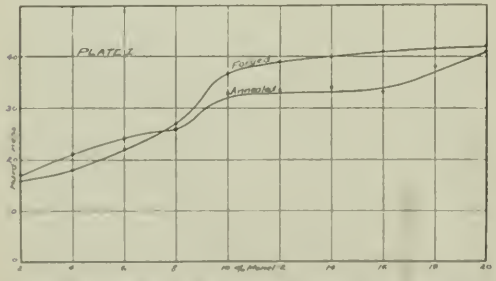


FIG. 1.—HARDNESS OF FORGED AND ANNEALED MONEL ALLOYS.

rapidly up to 10 per cent, and then more gradually to the end of the series. In the annealed samples the rise up to 10 per cent is not quite so abrupt, and from 10 per cent to 16 per cent is practically a straight line; from 16 per cent to the end of the series the rise is about the same as that below 10 per cent

The variation of the readings from the mean is less in this series than in either the nickel or copper series. Up to 7½ per cent the curve for the annealed samples lies above that for the

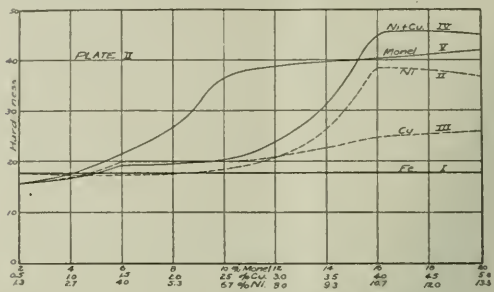


FIG. 2.—COMPARISON CURVES FOR FORGED ALLOYS.

forged samples, and the indications are that the curves would cross again slightly above 20 per cent.

The results are given in Tables I and II, both being plotted together in Fig. 1.

TABLE I.—HARDNESS OF FORGED MONEL ALLOYS

Per cent Monel.	Number.	Maximum.	Minimum.	Average.
2	1			16
4	3	18	18	18
6	1			22
8	3	28	26	27
10	3	42	34	37
12	3	40	37	39
14	1			40
16	3	42	40	41
18	2	43	40	41½
20	2	43	41	42

TABLE II.—HARDNESS OF ANNEALED MONEL ALLOYS.

Per cent Monel.	Number.	Maximum.	Minimum.	Average.
2	2	17	17	17
4	3	21	21	21
6	2	25	23	24
8	3	28	24	26
10	2	34	32	33
12	2	34	33	33½
14	3	40	31	34
16	3	41	31	33
18	3	42	35	38
20	3	41	40	41

In Figs. 2 and 3 are given the comparisons between the single additions of nickel and copper, and their addition together. First (I) the base line for iron is given in the forged condition at 18 and in the annealed condition at 14. Along with this are plotted the variations from the iron value caused by the addition of different amounts of nickel and copper (II and III). These values are then combined to form a fourth curve (IV), which is the result that might be expected if the hardening properties of the two metals when added together are strictly additive. The fifth curve (V) is the result actually obtained from the additions as stated, in comparison with what is to be expected from curve IV.

In the forged condition the hardness does not reach quite such a high value as might be expected, if the hardening powers of the two metals were additive, but the maximum value is reached at a much lower percentage, so that the small difference in hardness is more than balanced by the economy of the smaller addition. In the annealed samples the hardness is much above anything that could be predicted from the results

obtained from the single metals, the difference being particularly pronounced between 8 per cent and 16 per cent.

The comparison of the nickel and Monel curves shows several points in favor of the Monel. The maximum available hardness in both the forged and annealed condition, without the addition of excessive quantities of Monel, is at about 11 per cent or 12 per cent against 16 per cent for the pure nickel, the maximum values in both cases being practically the same. This leaves a large margin in favor of the Monel, a smaller quantity of a cheaper material giving the same results as a larger quantity of a more expensive material.

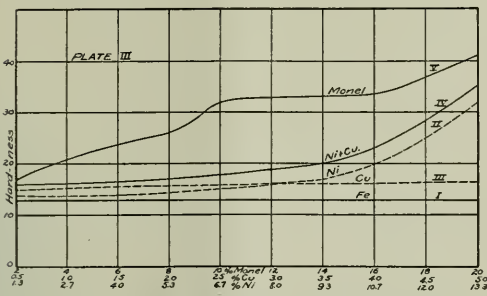


FIG. 3.—COMPARISON CURVES FOR ANNEALED ALLOYS.

As can be seen from Curves I and II, in Figs. 2 and 3, the addition of pure nickel to iron gives no appreciable increase in hardness below 6 per cent, while the addition of the same quantity of Monel gives a marked increase in the hardness, and it is within this range that most of the nickel-steels for construction work are found.

It is possible that some commercial use may be made of the fact that at about 7½ per cent Monel the hardness of the material is the same in both the forged and the annealed condition. With the pure nickel it is necessary to go to 21 per cent to obtain similar conditions.

Chemical Engineering Laboratories,
University of Wisconsin,
Madison, Wis.

The University of Pittsburgh is mailing copies of its *Bulletin*, Vol. 6, No. 7, giving a description and collection of data on the School of Engineering, with special reference to the Co-operative System. The 52-page booklet (June 15, 1910) contains notes on the equipment, requirements for admission and synopsis of courses for the following departments of the engineering school: civil, mechanical, electrical, chemical and sanitary. The degree of B. S. is awarded. The "co-operative plan of engineering instruction" is somewhat new and is being tried out by the university in order to take advantage of the unusual opportunities offered by the great industries of the Pittsburgh district for practice shop training to supplement the necessarily more theoretical work in school. Several of the large industries of Pittsburgh have made special arrangements to give practical training to the technical graduate and better fit him for a position in the organization. The Westinghouse Electric & Manufacturing Company, for instance, has a well-organized post-graduate two-year course. The co-operative plan of the university makes use of these and similar facilities. It has developed "a plan whereby the student, while spending in school the amount of time usually devoted to instruction in our best engineering institutions, will work four terms of three months each, in the engineering industries of the Pittsburgh district. By this plan the student gets the usual theoretical course and in addition 12 months of practical work, all in a space of four years, the school work being arranged so that successive groups of students will furnish continuous service to the employer."

The Application of the Grog in Firebrick Manufacture.

BY GILBERT RIGG.

In fireclayware the grog plays two important parts. Frequently it is more refractory to heat than the bond clay, and serves, therefore, to make the whole mass more refractory. Again, it forms a skeleton tending to hold the mass rigid and four-square, diminishing shrinking and preventing irregular shrinkage or deformation.

The type of firebrick which consists of a highly refractory grog and a less refractory bond clay has many points to recommend it, especially for moderate temperatures (say, up to 1300° C.), and for exposure to deleterious vapors. For grog we may use one of the highly refractory flint clays of a non-shrinking kind, or if it shrinks on burning it must be thoroughly burned before being incorporated in the mass. The bond clay should be highly plastic, softening at cone 25 or thereabouts and burning dense at 1000° C. From such ingredients an exceedingly dense and hard brick can be made, especially if the burning is carried out at a sufficiently high temperature to vitrify the bond.

The chemical composition of the grog should be such as to render it inert both chemically and physically. Pyrite and similar fluxes are highly objectionable. So also is free silica. Silica has the property of expanding on burning and in doing so will rupture the bond as the following experiment shows:

A certain bond clay of high plasticity and good quality was selected, made up with its own weight of a grog composed of the same clay burnt. Another lot of the clay was mixed with its own weight of coarse quartz sand of approximately the same size as the grog.

Specimens of both masses were burned at 1000° C. for 24 hours.

The clay and grog mass had a tensile strength of 322 lb. per square inch and the clay and sand mass 105 lb. per square inch. The former shrank 3.2 per cent on drying and the latter 2.1 per cent. On burning the grog mass shrank an additional 1.4 per cent, while the sand mass expanded again, the final shrinkage being only 1 per cent instead of the 2.1 per cent found on drying. This expansion of the sand grains strained and weakened the bond and reduced the breaking strain accordingly. By using more of the sand a mass was obtained that could be crumbled between the fingers after burning.

The question of the value of the determination of the breaking strain as an indication of density and compactness in a refractory mass deserves a short digression. From my own experience I believe that in judging the compactness of masses composed of a given clay, whose breaking strain is known, and equal quantities of different grogs, it serves as a valuable indication. Clearly the addition of comparatively coarse-grained, non-plastic material to a clay mass cannot increase its resistance to rupture under tension. All experiments go to show the contrary, just as occurs with concrete. For example, a certain bond clay, which burnt alone had a breaking strain of 326 lb. per square inch, fell to 213 lb. when mixed with its own weight of a burnt flint clay crushed through a 10-mesh screen.

The resistance to rupture of such a mixture of clay and grog depends upon the compactness of the mass, in other words, upon the degree of closeness with which the constituents adhere. Any separation between grog and clay, or through the particles of grog or the interstitial clay, at an angle to the direction of tension, will lower the breaking strain, and it is just such planes of separation or shrinkage cracks which give destructive vapors and fluxes their chance to enter the mass.

The breaking strain serves to measure the degree of coherence of the mass and in consequence supplies useful information as to the most suitable temperature of burning. When the breaking strain reaches a maximum it is certain that the mass is knit together as closely as the material it is composed of is

capable. Vitrified masses are usually exceedingly strong. It serves also to check the duration of burning. Some clays continue to gain in strength when kept for a week or more at the same temperature.

Then we have the question of shrinkage. The determination of the shrinkage of a mass is sometimes misleading. Some masses have a low shrinkage on account of their chemical composition. In others the low shrinkage is only apparent and is due to the internal rupturing of the mass by a mass of recticulated hair-cracks. The low breaking strain betrays this condition at once.

The application of the grog in such a manner as to give the greatest possible density in the mass presents certain problems of interest.

In the first place it is obvious that the individual grains should neither shrink nor crack. Then their shape should be such as to encourage close packing and interlocking, and the question of size needs consideration in connection with the work to which the bricks are to be put.

It is highly desirable that the grog should not shrink nor crack either during burning or during service. Yet it is not unusual to find firebricks with round grains of flint clay from ¼ in. to ½ in. diameter, cracked and ready to fall out. Some flint clays, though shrinking very little, if at all, have a marked tendency to crack and break up on heating into small, angular and exceedingly hard fragments, forming an excellent grog. When such flint clays, however, are worked into the mass green, in pieces larger than they would naturally break into on heating, they will split up inside the mass. No bond can possibly get into the cracks so formed, and such a grog becomes a source of weakness rather than strength.

Rounded grains have the further drawback of not packing well. Angular fragments pack much closer into one another, and for this reason are to be preferred. A tightly packed, non-shrinking, non-cracking grog tends to give a dense brick and what is highly important a brick that is true in shape. The more permanent the skeleton the truer will be the body.

The question of sizing is an important one. Where the grog is of coarse grain, the mesh work of bond clay will be coarse and the shrinkage on burning will cause the bond to open in a series of cracks.

Such ware is porous, but under certain circumstances this may be useful, as such goods expand and contract but little as a whole. In gas retorts, for instance, this plan is followed.

But in firebricks that have to withstand slags and destructive vapors, this plan is bad. What is needed in such a case is to get the grog skeleton as closely and densely packed as possible so as to minimize as far as may be the tendency of the bond to open.

In working out this question it occurred to me that useful results might be obtained by taking a hint from concrete mixing. In the latter operation, motives of economy make it important to use no more cement than is necessary to give the strength required. With properly graded stone, gravel and sand a considerable economy can be obtained over haphazard sizing and mixing.

The principle involved is that of filling the space as completely as possible with the stone, etc. If all the fragments are of the largest size used, the unfilled space is at a maximum. By adding the next size of material the spaces between the large pieces are partly filled. With yet another size more space still is occupied, and so on until the amount of cement is reduced to a minimum.

It can be shown that if the materials are graded into a number of different sizes by screening (mechanical analysis), then "concrete which works the smoothest in placing and gives the highest breaking strength for a given percentage of cement, is made from an aggregate whose mechanical analysis, taken after mixing the sand and the stone, forms a curve approaching that of a parabola, with its beginning at zero co ordinates (o) and passing through the intersection of the curve of the coarsest stone with the 100 per cent line—that is, passing through the upper end of the coarsest stone curve."

The foregoing passage occurs on page 195 of Taylor and Thompson's "Concrete, Plain and Reinforced." In succeeding paragraphs of that work a simple method of drawing the curve for any given set of sizes is shown.

I have made some trials of the application of this principle to grog-clay masses, with interesting results.

Starting out with a grog composed of well-burnt fireclay, crushed through a 10-mesh screen, the material was first separated on a series of screens into sizes. From these sizes a curve was calculated which gave the percentages of each size, which should be present. The different sizes were weighed out according to the percentages so determined. The table shows the difference between the composition of the original and the "rectified" grog.

Size.	Original.	Rectified.
Through 10 on 16	33.5 per cent	24.2 per cent
Through 16 on 25	18.1 per cent	14.0 per cent
Through 25 on 40	17.9 per cent	16.5 per cent
Through 40 on 60	10.8 per cent	10.2 per cent
Through 60 on 100	8.4 per cent	10.3 per cent
Through 100	11.3 per cent	24.8 per cent

The following mixes were made up:

(1) Seven parts of good, dense burning, fireclay, with nine parts ordinary grog.

(2) Seven parts of the same clay, with nine parts of rectified grog.

(3) Six parts same clay, with nine parts ordinary grog.

(4) Six parts same clay and nine parts rectified grog.

All these mixes were worked as nearly as possible in the same way and for the same time, pressed into molds, dried and burnt at 1000° C. for 24 hours.

	1	2	3	4
Pounds per square inch.....	270	341	233	301

After burning for 48 hours at 1100-1200° C. we had

Pounds per square inch.....	284	376		

This amounts to a gain in strength of nearly one-third. Such close sizing is, of course, not feasible outside the laboratory, but valuable results can be obtained by much rougher sizing, well within commercial possibilities, as shown by the following experiment:

A grog was used giving the following mechanical analysis:

Through 10 on 16.....	42	per	cent.
" 16 " 25.....	22	"	"
" 25 " 30.....	9	"	"
" 30 " 50.....	12	"	"
" 50	15	"	"

From this another grog was made up:

Through 10 on 16.....	24	per	cent.
" 16 " 30.....	26	"	"
" 30	50	"	"

These two grogs were worked up with the same bond clay as in the previous experiment and burned at 1100° C. Seven parts of clay were used to nine of grog. The following figures were obtained:

	Original grog.	Sized grog.
Shrinkage strain...	243 lb.	371 lb.
Shrinkage	4.84 per cent	5.44 per cent

The sized grog blocks shrank on burning more than the ordinaries, due, I think, to the fact that there was less internal cracking. Naturally, the densest mass would tend to have the smallest dimensions.

I have no opportunity of testing this out on a commercial scale, but I believe it would be worth while for some brickmaker to try it out. It must be borne in mind that the strength of these blocks made up with the original grog is high. Much of the firebrick, particularly the higher refractory stuff I have come across, is vastly lower than either. Some 2-in. bricks I have recently examined could be pulled in two with the hands.

Where a high refractory coefficient is obtained by cutting

down the bond clay to a minimum it seems to me that everything that can be done to make that bond clay do every pound per square inch of bonding possible should be done.

And even where the clay is of the same refractory coefficient as the grog, the increased density and strength of the ware would probably repay the trouble for many purposes.

New Jersey Zinc Company.

Concentration of Fine Sand on a Belt Vanner*

BY T. M. OWEN AND J. F. STEPHEN.

The Junction North concentration mill at Broken Hill, N. S. W., uses side-discharge belt vanners to concentrate the galena remaining in fine table-tailing. The vanner belt is supported on 3½ in. rollers spaced at 12 in. centers along the frame. The belt offers a dressing surface of 10 ft. 6 in. by 3 ft. 8 in., or about 38.5 sq. ft. The belt is made of two-ply, 32-oz. duck, with a coating of rubber on each side. The machine is given 245¾ in. strokes per minute and the belt travels at the rate of 15 ft. per minute. The dressing surface has a rise of ⅝ in. in its length, and a fall of 3 in. across its width. The machine makes but three products, which are discharged at the side of the belt, the concentrate being washed off by a strong water spray.

The feed supplied to these machines has rather peculiar properties. It first comes into the mill as a primary product from the rolls, being the undersize from a ½ in. by ¾ mm slotted screen; it then passes over Krupp tables, which take out a lead concentrate, sending the whole of the remaining pulp to Callow screens. These screens are fitted with a 40-mesh brass wire cloth, aperture 0.4216 mm or 0.0166 in. The undersize from these Callow screens constituted the feed of the vanners under consideration. The feed consists essentially of a mixture of quartz, rhodonite, garnet sandstone, galena and blende, the greater portion of the gangue being quartz and sandstone in varying proportions. Coming almost direct from the first crushing, the feed is very sensitive to any changes in the physical properties and grade of the crude ore; at one time the predominant gangue may be quartz, and a little later consist almost entirely of garnet sandstone. Thus the machine has to separate a lead concentrate from a gangue of two distinct types, light and heavy. Inasmuch as the types are present in varying and continually changing proportions, the separation may be regarded as one difficult to effect efficiently.

The results of the test given herewith represent the work of one vanner operating under typical conditions for one hour.

Concentrate.—Dry weight, 240 lb., or 14.2 per cent; bulk assay, 71.4 per cent lead.

Sieve.	Per cent weight each grade	Assay lead per cent.	Per cent lead to total lead.
On 40 mesh ...	0.20	64.8	0.18
" 60 " ...	8.70	79.6	9.61
" 100 " ...	6.17	80.6	6.90
" 120 " ...	14.17	79.8	15.68
— 120 " ...	70.76	68.9	67.63

Middling.—Dry weight, 600 lb., or 35.7 per cent; bulk assay, 8.8 per cent lead.

Sieve.	Per cent weight each grade	Assay lead per cent.	Per cent lead to total lead.
On 40 mesh ...	1.00	34.7	4.19
" 60 " ...	13.34	16.2	25.66
" 100 " ...	4.03	7.2	3.46
" 120 " ...	12.88	4.1	6.26
— 120 " ...	68.75	7.4	60.43

Tailing.—Dry weight, 842 lb., or 50.1 per cent; bulk assay, 4.4 per cent lead.

Sieve.	Per cent weight each grade	Assay lead per cent.	Per cent lead to total lead.
On 40 mesh ...	8.91	2.6	5.53
" 60 " ...	29.74	2.2	15.55
" 100 " ...	16.01	2.0	7.60
" 120 " ...	11.96	1.6	4.57
— 120 " ...	33.38	8.4	66.75

From the foregoing figures a feed sample is built up, giving the following:

Feed.—Dry weight, 1682 lb. Average assay, 15.34 per cent lead.

Sieve.	Per cent weight each grade	Assay lead per cent.	Per cent lead to total lead.
On 40 mesh ...	4.849	5.4	1.70
" 60 " ...	20.897	10.0	13.58
" 100 " ...	10.336	9.4	6.31
" 120 " ...	12.604	15.0	12.31
— 120 " ...	51.314	19.7	66.10

For purposes of comparison a sizing on a bulk sample of vanner feed is appended. This sample was taken at the close of the above test, and a close similarity will be noted between this bulk feed sample and the built-up feed sample given above.

Vanner Feed.—Bulk assay, 14.6 per cent lead.

Sieve.	Per cent weight each grade	Assay lead per cent.	Per cent lead to total lead.
On 40 mesh ...	5.3	8.8	3.2
" 60 " ...	21.7	10.2	14.8
" 100 " ...	6.8	9.0	4.1
" 120 " ...	15.6	10.7	11.2
— 120 " ...	50.6	19.7	66.7

Concentrate.—Figuring from the bulk assays of the products, the extraction of lead appears to be 65.5 per cent in the form of a 71.4 per cent lead concentrate; from the various sizing samples comes out rather higher —66.71 per cent as a 72.1 per cent lead concentrate. The extraction of lead from the — 60 + 120-mesh material is very high, being about 78 per cent; from the — 120-mesh material the extraction is slightly lower, 68 per cent; and from the + 60-mesh material the extraction is extremely low, being but 47 per cent on 60 mesh and 7 per cent on 40 mesh.

For an explanation of this large variation of extraction between the fine and coarse sizes it is necessary not only to refer back to the sizing tests, but to consider what happens to the material when it is fed to the vanners. An examination of the feed dissection shows that there is a gradual increase in values as the particles increase in size, due, no doubt, to the difference in crushing resistance between the gangue and the mineral. As extraction depends largely upon the grade of material treated, it is natural to expect the higher extraction from the finer size. In the foregoing test this is shown to be true up to a certain point, and it is here that it becomes necessary to consider the effect of the machine.

The efficient concentration of galena from gangue on a machine of this type depends largely upon the property of adhesion which this mineral seems to possess. Size of particle and specific gravity are other properties which affect the separation. The dressing belt is a plane surface, and can only be expected to deal with such material as will settle on it. When a pulp is fed onto this surface the heavier particles naturally settle first. The lighter particles travel farther on and do not settle until the water current has retarded. If the current is running right across the belt, particles of a certain size and weight—depending on the strength of the current—do not have a chance to settle at all, and go into the tailing without any treatment whatever. The drop in extraction on the — 120 mesh material is due to the loss of such particles. Had sieves finer than 120 mesh been available the effect of this loss of finer particles would have been more striking.

As the settled material travels along with the belt it is subjected to the cross flow of the dressing water, and the gangue, possessing but little "cling," is washed off more easily than the

*Abstract of a paper read before the Scientific Society of Broken Hill and published in the *Transactions* of the Australasian Inst. of Min. Eng.

mineral. The lead gradually becomes less as the belt travels forward and the galena particles become more exposed to the direct action of the water. When these particles come under the strong flow of the concentrate spray, some of the larger sizes are washed and become associated with gangue material, finally going to the middling. The escape of these particles to the middling, combined with the low assay of the coarse sizes of the feed, accounts for the low extraction of coarse lead shown in the vanner concentrate.

Middling.—The proportion of this product is at once noted as being high; the explanation of this lies in the fact that the machine was not set up for experimental purposes. To cope efficiently with the changing character of the feed it was necessary to safeguard the grade of the tailing by cutting off a large proportion of middling. With a very uniform feed the amount of this product could be greatly reduced without materially affecting the grade of tailing. On some of the experiments the middling was reduced to 15 per cent of the total weight of feed, and then assayed but 9.7 per cent lead. It is considered that, with a steady rate of feed, these results could be obtained regularly.

An examination of the sizing table of the middling shows that the high lead values are contained in the coarser sizes—the 40 mesh assaying 34.7, and the 60 mesh assaying 16.2 per cent lead, while the bulk assay is 8.4 per cent lead. An inspection of the dried sample shows it to contain large particles of free lead. These particles are forced into the middling by the concentrate spray, as explained above. The low assay value of the —120-mesh size—7.4 per cent lead—goes to prove that the strong flow of water from this spray forces but a small quantity of fine lead into this product.

An efficient separation of the free lead contained in the middling can be effected by mixing the product with the vanner feed and returning it to the machines. The same proportion of coarse lead again goes to the middling, while the surplus is forced into the concentrate as a clean product. As there is almost enough feed of the class already described to supply the seven vanners available, but a small quantity of the vanner middling is returned to the machines—only sufficient to keep a fairly constant rate of feed. The remainder is mixed with the Wilfley feed, which is a coarser product, and contains very little sandstone.

Tailing.—An examination of the sizing table of the tailing shows that the — 120-mesh product assays 8.4 per cent lead, and contains 66.7 per cent of the total lead held in this product. The 120-mesh material assays but 2.1 per cent lead. It has already been stated that when the feed pulp comes onto the machine, particles of a certain size and weight, depending on the strength of the water current, do not have a chance to grip the belt, but go directly to the tailing compartment. Such particles are found in the material which passes the 120-mesh sieve, and are unsuitable for treatment on a sand vanner. It is commercially unprofitable to separate such particles by screening, but a partial separation can be effected by delivering the feed to the machine with as little water as possible. Particular attention has been paid to this point, and the average consistency of the feed pulp is one part of solids to 1.65 parts of water by weight.

A washing test of the — 120-mesh material contained in the tailing showed that 46.5 per cent of the total lead contained in the tailing is present as a very fine slime, and can be easily removed. A separation on this principle is effected in the bulk tailing from the sand vanners by running them to a small settler which discharges thickened sand at the bottom, and slime and water at the top. This slime is settled and returned to slime vanners for retreatment.

Summing up the results of the above test it was found that the machine handled material at the rate of ¾ ton per hour; that in one operation, despite the presence of large quantities of heavy garnet sandstone gangue, 14.2 per cent by weight of this material was discharged as a high-grade lead concentrate containing 65.5 per cent of the total lead value; that 35.7 per cent by weight of the material handled was turned out as a middling product, containing free lead in a form eminently suitable for subsequent recovery; that 5.25 per cent by weight of the material handled was present as slime in the tailing and could be separated in a form suitable for retreatment; that 44.85 per cent by weight of the material handled was discharged as a finished product suitable for dumping, containing but 7.33 per cent of the total lead value and too low grade for retreatment.

The largely varying amounts of garnet sandstone in the feed make very little difference in the working of the vanner, but, in the presence of any quantity of garnet sandstone, the Wilfley table fails to separate a clean concentrate. To further explain this point the results of two typical tests are given: A No. 5 Wilfley table was fed with material of the same type as that treated by the vanners. Following are the results:

Test No. 1.—Gangue containing a very small percentage of garnet sandstone—

Concentrate	72.2	per cent lead,
Middling	12.5	" " "
Tailing	4.1	" " "
Slime	17.2	" " "
Extraction	61.2	" " "

This shows good work, but not quite as good as that of the vanner.

Test No. 2.—Gangue containing a large percentage of garnet sandstone—

Concentrate	61.8	per cent lead.
Middling	25.0	" " "
Tailing	3.3	" " "
Slime	20.4	" " "
Extraction	51.4	" " "

These figures show a low-grade concentrate with a high-grade middling, combined with a rather poor extraction of the lead values. When the table is working it is at once apparent that it fails to give a good separation; the sandstone rides on top of the galena, and the division between concentrate and middling is poorly marked. Below the point where the concentrate is cut off an intimate mixture of sandstone and galena, assaying from 40 to 50 per cent lead, extends for about 5 in. This material is too low grade for concentrate, and had to be run into the middling, explaining their high grade. The extraction is very poor compared with that of the vanner.

The authors conclude that in the treatment of fine, heavy garnet sandstone sand the belt vanner is a more efficient machine than the Wilfley table. The proportion of middling returned for retreatment is 15 to 20 per cent of the total feed; the proportion of finished products (tailing and concentrate) is 80 to 75 per cent of the total feed, and the remaining portion of 5 per cent is slime which is available for retreatment or stacking.

Fixation of Atmospheric Nitrogen in France.—It is reported from France that the company "La Nitrogène" has just started a plant at Roche de Rame, Hautes-Alpes, for the production of nitric acid from atmospheric air by means of arc discharges. They are using the Pauling process, operating under license from the Salpetersäure-Industrie-Gesellschaft in Gelsenkirchen. The Pauling process was described and illustrated in our vol. VII, p. 430, the first plant using this process being located at Innsbruck. In the new French plant water power is also used; the hydro-electric installation comprises four units of 2000 hp each. The furnaces are arranged in two rows in a separate building. Each furnace has two arcs in series. Three-phase currents at 4000 volts with a frequency of 50 periods are employed. The phases are divided equally between three furnaces. At present the capacity of each furnace is 500 hp, but this will probably be raised to 1000 hp. In normal operation the yield of nitric acid is stated to be 60 to 70 grams per kw-hour. The whole plant is almost automatic, so that little hand labor is required.

Cupro-Nickel Steel.[1]

By G. H. CLAMER.

The iron-nickel and iron-copper series of alloys are of particular interest as showing what surprising results are obtained by the joint addition of copper and nickel. For example, alloys can be produced carrying from 5 per cent to 20 per cent of copper, which without nickel would be extremely hard, brittle and red short, but these become, by the addition of nickel in the proportion of from 20 per cent to 50 per cent, highly ductile, easily forgeable and easily machinable. Such alloys are capable of being rolled into sheets or rods and can be machined without difficulty—in fact, they are quite soft.

The following are tests of two typical alloys within these ranges of proportions:

	No. 1.	No. 2.
Copper, per cent	9.00	19.50
Nickel, per cent	22.00	45.00
Carbon, per cent	0.22	0.20
Ultimate strength, pounds per square inch	101,000	98,230
Elastic limit, pounds per square inch	57,300	54,150
Elongation, per cent	42.5	35.0
Reduction of area, per cent	61.5	47.9

These tests were made on rolled rods. The elastic limit, it will be noted, is quite low in proportion to the ultimate strength, which naturally accounts for their softness. Alloys at the lower end of the series can be bent cold to an angle of 180° without fracture, whereas those at the higher end cannot be bent so far. Alloys at the lower end of the series will resist corrosion in a highly satisfactory manner, and those at the upper end are practically non-rusting. Samples which have been exposed for the past year in contact with concentrated foundry gases, and at Atlantic City in contact with salt atmosphere, have demonstrated these facts. In addition to this, the alloys have a very low coefficient of expansion.

Just what uses such alloys might be put to has not yet been determined, but no doubt quite a number will suggest themselves, e.g., as a substitute for the present 25 per cent to 35 per cent nickel-steels and probably for quite a number of others. For the cupro-nickel steel can be produced at a very much lower cost, not only because of the substitution of copper for nickel (copper selling at the present time at less than one-third the price of nickel), but because the alloy can be made from monel metal, a commercial alloy of copper and nickel, containing copper and nickel in the correct proportions for making such a steel. Monel metal sells for about the price of copper.

It is also possible to make the steel directly from Sudbury nickel ores, by simply smelting and bessemerizing them to eliminate the iron to the extent desired, roasting the matte produced, and finally reducing the combined oxides resulting from roasting the matte. Either by the use of monel metal for making the steel alloy, or by the direct production of it from the ore, it is thought that the cost of production might be sufficiently reduced to make it attractive for roofing, gutters, spouts, ventilators, and for many other uses where its resistance to corrosion and its low coefficient of expansion will warrant the cost.

Sheets of cupro-nickel steel can be produced to sell at a very much lower price than copper sheets and probably for two to four times the price of commercial coated iron and steel sheets; but since the alloy is about twice as strong as the soft steel and three times as strong as copper sheets, thinner gages can be used. The alloys at the higher end of the series are non-rusting.

Steels within this range of proportions of copper and nickel have a crystalline structure, such as is possessed by nickel steels carrying above approximately 25 per cent of nickel, or with a somewhat lower percentage of nickel if carbon is present in an appreciable amount—in other words, like nickel steel within the crystalline range—the copper apparently acting very much like nickel. It will be noted that the physical properties are very similar to those of the same nickel steels without copper.

Noting the curious and unexpected results obtained with these high copper-nickel alloys, I became curious to know what would be the effect of replacing part of the nickel in nickel steel within the pearlitic range of copper, and accordingly made such steels with from 1 per cent to 8 per cent of monel metal, in which the proportion of nickel to copper is, roughly, 2.5 per cent of nickel to 1 per cent of copper. Here again the copper really acts like so much nickel, at least when present in these relative proportions, and it is possible, therefore, to produce a steel of practically the same physical properties as nickel steel at very much reduced cost. In these steels the nickel, iron and copper are in the same relative proportions in which they exist in some of the prominent mines at Sudbury, where vast deposits exist, average analysis of these ores being as follows:

Copper	0.8 to 2 per cent.
Nickel	2 to 5 per cent.
Iron	35 to 52 per cent.
Sulphur, Gangue	Remainder.

It will be seen that, by the operation of reducing this ore to matte, calcining the metal to remove the sulphur, and reducing the calcined product, this steel can be produced almost to any desired formula.

The valuable properties of the nickel steels—i.e., steels carrying below 6 per cent of nickel—have, of course, long been recognized, and recently the value of copper as a valuable constituent of steel, instead of a metal to be looked upon as a detrimental impurity, has also become recognized. It is probably not quite so surprising, therefore, that the joint addition of copper and nickel within the range of proportions here considered is productive of valuable results. The addition of these two metals in combination seems to have the same effect upon the steel as if they were individually added, the copper in its effect really being about the same as so much added nickel. It is possible, therefore, to replace part of the nickel in nickel steel by copper without materially altering its physical properties. The following table gives the physical properties of some of these alloys:

No.	Monel metal, per cent	Carbon, per cent,	Ultimate strength lbs, per sq. in.	Elastic limit, lbs. per sq. in.	Elongation, per cent.	Reduction of area per cent,
1	4	0.11	83,600	66,300	26.5	65.4
2	3	0.10	75,300	64,600	28.5	71.4
3	2	0.15	75,500	63,700	28.0	65.4
4	4	0.67	151,000	120,000	8.0	15.8

Recently Messrs. Burgess and Aston, working quite independently of me, have confirmed these results with alloys which they have made with electrolytic iron, which has been quite gratifying.

Although I have not had an opportunity to study all the properties of some of the alloys within these ranges, i.e., the pearlitic and crystalline alloys, nevertheless, it would seem from the tests so far made, that these alloys may become of commercial value, especially as we now have at hand a commercial monel metal—and the further possibility of producing them directly from Sudbury ores.

* * *

In the discussion which followed, Dr. JOHN A. MATHEWS, of the Halcomb Steel Company, of Syracuse, N. Y., said that the general trend of scientific work in connection with the influence of copper on steel has been to show that, in any ordinary quantities in which it is likely to be found, its effect is not marked one way or the other, or may even be beneficial.

Dr. Mathews then gave some data on a particular experiment with cupro-nickel steel which is of more than ordinary interest, owing to the fact that the heat of nickel steel containing

[1] Extracts from a paper read at the recent Atlantic City meeting of the American Society for Testing Materials. The author begins with a review of recent work on the properties and constitution of alloys, with special reference to iron-nickel and iron-copper alloys and to the investigations of C. F. Burgess and J. Aston on this subject. The present paper is of special interest in connection with the article of Burgess and Aston in our last issue (page 452) on the strength of alloys of electrolytic iron with monel metal.

copper was melted in their Héroult electric furnace, and that the pig iron used in this heat was produced in Canada in the Héroult smelting furnace which was installed there some years ago, that is, the entire heating from the ore to the finished product was by means of electricity. This is probably the first heat of steel so made in the United States.

The pig iron as produced from certain of the Canadian ores carries a considerable amount of nickel and copper, and it was with a view to seeing what results could be obtained from this pig iron that they made a cold heat, charging cold pig and ore into their furnace. This heat gave steel of the following analysis:

Carbon	0.44	per cent
Silicon	0.034	" "
Manganese	0.50	" "
Phosphorus	0.013	" "
Sulphur	0.013	" "
Nickel	3.62	" "
Copper	0.48	" "

It must be remembered that all the nickel and copper present were contained in the original pig iron used. They had no definite analysis in mind when they started this heat, but were more interested in finding what grade of steel would result from the grade of pig used. The ingots were clean and worked very satisfactorily under the hammer, and the 0.48 per cent of copper present seems to have exerted no deleterious influence as regards the heating or working.

For the sake of comparing this steel with another steel of approximately the same analysis, they selected an open-hearth nickel steel which analyzed as follows:

Carbon	0.46	per cent
Silicon	0.066	" "
Manganese	0.70	" "
Phosphorus	0.021	" "
Sulphur	0.034	" "
Nickel	3.36	" "

An examination of this heat showed that it also contained about 0.10 per cent of copper.

The physical tests to be mentioned later were made from round bars, ⅞ in. in diameter, and the greatest care was used in their heat treatment. Tests were made in duplicate, and the annealing of the bars from the two heats was done at the same time, so as to avoid any possible difference in the annealing temperature. It was noted that the open-hearth steel machined a little more freely than the electric-furnace steel containing 0.48 per cent of copper, but here the superiority ended. The tensile tests of the material as rolled gave the following values:

	O. H. Nickel.	Cupro-Nickel.
Elastic limit, lbs. per sq. in.	74,625	72,400
Tensile strength, lbs. per sq. in.	122,000	115,000
Elongation in 2 in., per cent	16	22
Reduction in area, per cent	34	51

In the annealed condition the physical tests were as follows:

	O. H. Nickel	Cupro-Nickel.
Elastic limit, lbs. per sq in.	64,750	63,750
Tensile strength, lbs. per sq. in.	119,000	107,300
Elongation in 2 in., per cent	17	25
Reduction in area, per cent	37.5	48

Samples of both steels were heated to 1500° Fahr., quenched in oil, and drawn back to a temperature of 800° Fahr and maintained at that temperature for 20 minutes. In this condition the results of physical tests were as follows:

	O. H. Nickel	Cupro-Nickel.
Elastic limit, lbs. per sq. in	154,500	154,000
Tensile strength, lbs. per sq. in.	175,000	172,500
Elongation in 2 in., per cent	9.75	13.25
Reduction in area, per cent	30.8	49.1

In all of the above tests, the elastic limits and tensile strengths are not very different for the two steels, but the great superiority of the cupro nickel steel in elongation and reduction of area is most noticeable. The presence of the copper certainly has not in any way decreased the toughness of the steel, as manifested in its percentage of elongation and reduction of area, but the most remarkable difference in the two steels was noticed in samples which had been quenched as before at 1500° Fahr. and drawn back to 600° Fahr. In this condition, both steels showed an elastic limit of about 185,000 lb. per square inch, and a tensile strength slightly above 200,000 lb. per square inch; but, while the open-hearth steel was decidedly brittle after this treatment, giving only 2 per cent elongation and 4 per cent reduction in area, the electric-furnace product gave 12 per cent elongation and 46 per cent reduction in area.

So far as this single test is concerned, therefore, there is absolutely no evidence to show that the copper has exerted a harmful influence upon the nickel steel, and we must either ascribe its superiority to the presence of copper, or to the superiority of the electric method of melting, or to both acting jointly. From an extended experience with the electric-furnace product as compared with the open-hearth nickel steels in carbons varying from 0.10 up to 0.50 per cent, there no longer remains any doubt in Dr. Mathews' mind as to the superiority of the former over the latter.

Recent Advances in Electric Pyrometry.

An interesting review of progress in pyrometry was given in a recent paper by Mr. Robert S. Whipple before the Birmingham Metallurgical Society. The paper, which is printed in full in the London *Engineering* of July 22, discusses in succession the advances made in gas thermometry, resistance thermometry, thermo-electric pyrometry, and radiation pyrometry.

We give here in abstract those parts of Mr. Whipple's paper which deal with electric resistance and thermo-electric pyrometry.

Resistance Thermometry.—There is not, according to Mr. Whipple, any striking advance to report in this branch of thermometry. A valuable paper by Dr. Harker on "High-Temperature Standards of the National Physical Laboratory" (*Philos. Transact.*, Royal Society, A, Vol. 203, pages 343 to 384), gives the results of a series of comparisons made between the resistance and thermo-junction standards and the nitrogen-gas thermometer up to 1000° C. These results entirely justify the use of Callendar's parabolic formula up to 1000° C.

Waidner and Burgess (*Bull. Bur. of Standards*, Vol. 6, No. 2) have since confirmed Harker's results in an investigation of the behavior of resistance thermometers made of platinum wires of varying degrees of purity. The following is a brief summary of their results, but those interested should refer to the original memoir:

1. Temperatures defined by the resistance thermometers of pure platinum calibrated by Callendar's formula at 0° C. and 444.70° C., the boiling point of sulphur, agree with temperatures on the accepted gas scale from 0° C. to 1100° C.

2. Thermometers made of impure platinum and calibrated in the same way do not define the temperature scale as those of pure platinum, the divergence increasing with the impurities in the metal.

3. If the thermometers of impure platinum are calibrated at a fourth point, such as the melting point of silver, and the Callendar formula modified accordingly, then the scale is brought into close agreement with that of the thermometer of pure platinum.

4. For temperatures above 900° C. the constants of platinum thermometers undergo gradual changes, necessitating frequent recalibration in work of high precision. These changes are greatly reduced by annealing the thermometers for several hours at a temperature higher than that at which they are to be subsequently used.

5. In pure-wire thermometers these changes are extremely small, very much less than in the case of those made with impure wire. In the latter case the thermometers appear to im-

prove with time as though the impurities (such as iridium) evaporate by successive heatings.

In the construction of resistance thermometers some improvements have been introduced, the most important being that due to Dr. Haagn (report of Reichsanstalt, *Zeit. f. Instrumentenkunde*, 1907, No. 4, page 116), who suggested to Messrs Heraeus, of Hanau, the construction of a thermometer in quartz, the wire being wound on a rod of quartz and protected by a very thin tube of quartz fused over it. This thermometer can be made as small as 25 mm long and 3 mm in diameter, and yet be comparatively robust. By this arrangement the wire is protected from the injurious effect of gases and mechanical strain; it also very quickly attains the temperature of the surrounding medium. It is, however, handicapped with a drawback that quartz devitrifies if exposed for any length of time to temperatures as high as 1000° C.

Two new methods of winding the thermometer coils for avoiding strain on the wires are shown in the paper by Waidner and Burgess, previously referred to; one, in which, by means of a modified form of mica frame, a flexible mounting is given to the wire (Fig. 1), and the other, due to Northrup, in which stouter wires, in the form of a spiral without any support, are employed (Fig. 2).

Thermo-Electric Pyrometry.—Progress has here been made essentially in three directions, which will be discussed successively under *a*, *b* and *c*.

(a) **Increasing the Purity of the Metals Employed in the Platinum Thermocouple.**—Great steps have been made comparatively recently in the production of pure and homogeneous metals for standard thermocouples. The firms of Johnson, Matthey & Company, of London, and Heraeus, of Hanau, have sunk a considerable amount of capital in large ingots of pure platinum, platinum with 10 per cent rhodium, and platinum with 10 per cent iridium for this purpose. In the case of a sample supplied for thermocouples to the National Physical Laboratory by Johnson, Matthey & Company, the analysis of the platinum-rhodium alloy was as follows:

Platinum = 89.9
Rhodium = 8.98

both being directly determined and no trace of any other metal being discovered. (J. A. Harker, *Proc. Royal Soc.*, Vol. 76, A, page 243.) The ingots owned by Johnson, Matthey & Company have been found to be exceptionally homogeneous, wires obtained from any part of the ingot giving the same e.m.f.

Messrs. Heraeus have now been able to repeat their standard of purity for some time, as the following data supplied by them will show, giving the average e.m.f. at 1000° C. for the last six years:

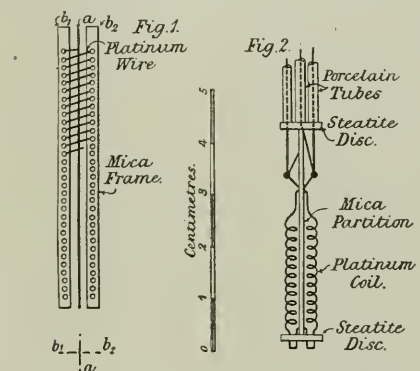
Year.	
1904 = 9.52	millivolts.
1905 = 9.53	"
1906 = 9.53	"
1907 = 9.57	"
1908 = 9.59	"
1909 = 9.55	"

As a result of the work of these two firms it is possible to obtain a second thermocouple similar to one previously obtained, which may be used without introducing any serious error on the same galvanometer or recorder.

It has been known for some time that platinum thermocouples deteriorate in an atmosphere of hydrogen, but it has not been possible to account for a large number of the failures. No very definite information was available on this subject until the paper published by White (*Phys. Review*, December, 1906). In this paper it was shown that the main source of error in thermoelements is non-homogeneity, by the removal of which, in most cases, far greater accuracy may be attained than has hitherto been realized. It was also shown that the iridium slowly sublimes from the platinum-iridium side of a couple and enters into combination with the platinum side, thus slowly reducing the e.m.f. given by the couple. This sublimation has been found to commence at 900° and to become serious at 1200° C.

The destruction of the platinum, platinum-rhodium thermocouples in the earlier stages of Day's later work could only be accounted for by the sublimation of the platinum-iridium heating coil employed in the furnace. This was replaced by a platinum coil and finally platinum-rhodium was substituted for the gas thermometer bulb itself because the iridium from the platinum-iridium bulb tended to destroy the couples. (*Am. Jour. of Science*, Vol. 29, February, 1910.)

Iron has undoubtedly a disastrous effect on platinum, and the



FIGS. 1 AND 2.—METHODS OF WINDING PYROMETER COILS.

destruction of large numbers of couples may be put down to this cause.

Annealing a low-reading thermocouple to a temperature of 1200° C. will frequently restore it to its original e.m.f.

When the use of contaminated elements is necessary, fair accuracy can still be obtained if the defective couples are frequently compared with standard ones.

(b) **Production of Cheaper, or Base Metal, Couples.**—A large number of wires have been suggested for making cheaper thermoelements. Onnes and Clay have employed gold and silver elements for very low temperatures with success, this couple showing itself at −253° C., as read on a hydrogen thermometer, to be twice as sensitive as constantan steel. (*Konink. Akad. Witensch.*, Amsterdam *Proc.*, 11, page 344, December, 1908.)

The most satisfactory base-metal couple for temperatures not exceeding 700° C. is undoubtedly copper-constantan (constantan = 60 per cent copper, 40 per cent nickel), and this couple has been used in several scientific researches. (E. H. Rayner, *Jour.*, Inst. Elec. Eng., Vol. 34, page 628; A. Gamgee, *Phil. Transact.*, B, Vol. 200, pages 219 to 240.)

Among other couples may be mentioned:

Iron-constantan.
Iron-nickel.
Iron-nickel chromium.
Copper-nickel.

A useful book by Prof. H. Pecheux has lately been published on the thermocouple. ("Le Pyromètre Thermo-Electrique," Paris: Gauthier-Villars.) In this he urges the adoption of the copper-nickel couple for use up to temperatures of 900° C., and he also shows that although it is impossible to make one parabolic formula fit the whole range of usefulness of the couple, yet with the aid of three formulas covering varying parts of the range the observed results agree with the calculated results to within 1° over the greater part of the scale.

Mr. Whipple's own experience shows that it is extremely difficult to obtain homogeneous wires, and, therefore, for accurate results it is necessary to calibrate every individual couple, and even then the calibration may be found to have seriously

altered, if, by any chance, the couple has been overheated or, perhaps, overannealed.

For accurate work it is customary to maintain the cold junction of a thermocouple at 0° C. (By using a Dewar flask to hold the ice a small quantity can be made to last for a considerable time.) But not infrequently, in what may be called semi-scientific work, it is not easy to control the cold junction temperature. Although it has been customary to have the difference between the temperature of the head of the thermometer and 0° C. and to add this quantity to the obtained hot junction temperature provided the cold junction correction does not exceed 50° C., yet this is incorrect, as the relation of the e.m.f. to temperature is not a straight line. R. Vogel (*Zeit. f. Anorg. Chemie*, 1905, 45) appears to have been the first to have calculated and applied correction factors for this cold-junction temperature. The matter has since been fully discussed by C. Offerhaus and Ernst H. Fischer (*Electrochem. and Met. Ind.*, Vol. 6, page 362, September, 1908), who show that the correction depends on both the hot and cold junction temperature and the particular couple. Briefly, they show that the true temperature of the hot junction is not, starting with the needle at zero (the cold junction having the temperature of the room), the temperature which corresponds to the millivolt reading, but is found by adding the number of millivolts which corresponds to the temperature of the cold junction to the millivolt reading.

(c) **Methods of Measurements with Thermocouples.**—For accurate work the conditions imposed upon a potentiometer for use with thermo-electric couples are very exacting and may be grouped as follows:

1. If possible, temperatures up to 1600° C. must be read to 0.1° C. with a platinum, platinum-rhodium element; 0.1° C. corresponding to about 1/1000000 of a volt.

2. The potentiometer coils must be unaffected by leakage from external circuits. (This is important where, as is now generally the case, the thermocouple is being used in a furnace which is electrically heated by heating coils, the potential difference in which may be 100 volts or more, any leakage to the potentiometer coils introducing serious errors.)

3. Manipulation must be simple in order that quickly altering temperatures may be followed.

To meet the above condition various potentiometers have been designed. Several recorders of various types have been introduced for use with thermocouples. In connection with one of these some experiments have lately been tried with a view to increasing the stability of zero. It has been found that a phosphor-bronze wire reduces the creep from one-half to one-third that which would be given by the same wire rolled as a strip. The instrument is arranged to deflect across the scale for an e.m.f. of 1 millivolt. The total range of the recorder can be adjusted by means of resistances in series with the couple.

Standardization.—Mr. Whipple remarks that it is interesting to note the tendency of recent workers to adopt the melting points of pure metals as the standards of reference, several of the melting points being known to a high degree of accuracy. Unfortunately the metals cannot be obtained commercially sufficiently pure to guarantee an accuracy of 1° C. at the higher temperatures, but if an accuracy of 1° to 3° is sufficient, then Kahlbaum's best materials will answer the requirements.

It is hoped that in the near future the (British) National Physical Laboratory or some similar institution will be able to supply small quantities of pure metals, the melting points of which have been carefully studied.

The following are given by Mr. Whipple as the most satisfactory standardization points, the values given being as far as possible the latest determinations:*

Freezing point of tin	231.9°	*Bulletin* of the Bureau of Standards, Vol. VI, No. 2.
" " " cadmium	321.01°	
" " " lead	327.4°	
" " " zinc	419.37°	
" " " antimony	630.7°	
" " " silver	960.8°	
" " " copper	1083.0°	
" " " nickel	1452.2 ± 2°	A. L. Day and B. B. Sosman, *Am. Jour. of Science*, Vol. XXIX, Feb., 1910.
" " " cobalt	1489.2 ± 2°	
" " " palladium	1549.2 ± 2°	
" " " platinum	1755.0 ± 5°	

Boiling point of sulphur, 444.70°.

Black-body temperature of the crater of the electric arc, 3700° C. (This latter point is constant, even although the current through the arc is varied between large limits.)

* * *

The Thermo-Element as a Precision Thermometer.—An interesting investigation of the use of the thermo-element as a precision thermometer has been carried out at the Geophysical Laboratory of the Carnegie Institution of Washington, the results being given in a paper by Walter P. White in the *Physical Review* of August. The investigation dealt with the construction, accuracy and convenience of the copper-constantan thermo-element. The data were derived from the study of four multiple thermo-elements which have been used for about three years as calorimetric thermometers at the Geophysical Laboratory, and also from a calibration where two of them were used side by side with a resistance thermometer to measure the same temperatures.

Constantan and copper were chosen as forming, on the whole, the best available combination for work at ordinary temperatures. Constantan and steel, its only serious rival at present, excels only in a slightly higher e.m.f. and has the disadvantages that (1) steel wire is not so readily obtainable insulated as the others, and (2) with constantan and steel we have two metals instead of one which may cause appreciable error from inhomogeneity—a defect which for most purposes evidently more than offsets the slight advantage in e.m.f.

The results are summed up as follows: The e.m.f. of a thermoelement, whether homogeneous or not, can be expressed as equal to $\int H dT$, where H = thermoelectric power and T temperature. It follows that the effect of each portion of a thermoelement is proportional to the magnitude of the temperature gradient in which it lies; hence the vital parts of a thermoelement are the parts along which the temperature varies; they mainly determine the calibration and the constancy; with them alone need the maker or user of the thermoelement be much concerned. In changing them (as by varying the depth of immersion in a furnace) the thermoelement itself is changed, if not homogeneous, while alterations in regions of uniform temperature distribution, as at the junction, have little effect on the temperature measurement.

Commercial constantan wire good enough to make thermoelements accurate to about 0.05 per cent is easily obtained and costs but little trouble in the selection, but cannot be relied upon without any test for possible inhomogeneity. Compensated thermoelements of higher accuracy, reading consistently to 0.0001°, for use at ordinary and low temperatures, can be made in a day or two each. Improvement in the accessory apparatus has shown that their constancy is about ten times that shown three years ago.

The accuracy of 0.0001° reached by the thermoelement can also be attained in its auxiliary apparatus (potentiometer, etc.). The resulting accuracy, both absolute and relative, and the sensitiveness are quite comparable with those of the best resistance thermometers yet used.

For the measurement of single temperatures the resistance thermometer is often more convenient. On account principally of the comprehensiveness of the potentiometer and its indifference to external contact resistances, the thermoelement is usu-

* Compare also the figures given by Dr. A. L. Day in the May issue of this journal, p. 260.—Editor.

ally preferable where several different measurements are to be made at once. The thermoelement is also usually superior for differential measurements, and its advantages in this respect can be made available for calorimetric work by the substitution of a suitable comparison body for the usual ice bath.

A very convenient wire tester, a simple bath for testing and comparing thermoelements, and a convenient form of tabulation for calibration curves are also described in the paper.

* * *

In connection with this investigation a paper by W. P. WHITE, H. C. DICKINSON and E. F. MUELLER should be mentioned, which is also published in the August issue of the *Physical Review*, on the calibration of copper-constantan thermoelements. A pair of copper-constantan thermoelements for use in calorimetric work was calibrated by comparison with resistance thermometers to an accuracy of 0.004°. It seems probable that the accuracy of the calibration could be considerably exceeded. A cubic equation probably represents the relation between temperature and e.m.f. in the interval 0° to 100° with an accuracy of 0.005° or better. New thermoelements may be very conveniently calibrated by direct differential comparison with those whose calibration is already known.

The Constitution of Cast Irons and Carbon Steels from the Practical Standpoint.[1]

BY DONALD M. LEVY.

The object of this paper is to offer a simplified view of the relations existing between iron and carbon over practically the whole range of cast irons and carbon steels.

It is especially desired to indicate how such a point of view helps toward a clear understanding as to the manner in which each particular constituent, such as the sulphur, the silicon, and the manganese, either singly or in conjunction, affects these relations in cast irons. It will thus, it is hoped, help toward an understanding of the direct relations which exist between the heat treatment, composition, and properties of these alloys. It may further, perhaps, indicate the value of very much of the scientific work which has too often been regarded as mainly of academic interest, but has, nevertheless, assisted toward a clearer understanding of the causes of many phenomena with which the practical man is greatly concerned.

For reasons which are discussed in an appendix the present equilibrium diagram cannot be regarded as altogether satisfactory as a full realization of the relations of iron and carbon in these alloys. It has tended to make the question so much a matter of the physical relations between these elements that the chemical side of the problem has not been sufficiently appreciated. It is hoped in this paper to indicate the primary importance of the meta-stable chemical compound iron-carbide, in connection with the constitution of iron and steel, and by noting the influence of the varying physical and chemical conditions upon it, to show how a clearer, simpler and more practical view is arrived at.

The object of this paper is therefore to classify cast irons and carbon steels as primarily a series of iron and iron-carbide alloys, and by means of what may be termed a constitutional diagram, to indicate what constituents tend to separate and the conditions under which these constituents separate in the alloys.

It is intended to show how (instead of the two systems hitherto considered necessary) the conception of a single series of alloys meets both the theoretical and practical requirements of the subject; affording a clearer idea of the real points of practical interest, and assisting in their application with some certainty to practical problems.

[1] A paper presented before the Iron and Steel Institute at the May meeting, 1910.

[2] D. M. Levy, "Iron, Carbon and Sulphur," *Journal of the Iron and Steel Institute*, 1908, No. II, p. 85.

The Iron-Carbide Series of Alloys.

In a Carnegie Research Memoir,[2] in which it was endeavored to find an explanation for the effects of sulphur in cast iron, conclusions were arrived at regarding the condition in which the carbon separated in cast irons, which offered a rational explanation for many long observed but hitherto unexplained phenomena. The part played by sulphur will be referred to later, but the observations concerning carbon may be quoted as follows:—

"The iron-carbon alloys can best be regarded as an iron: iron-carbide system, and when molten as solutions of iron-carbide in iron. The graphitic system is largely a secondary one, arising from decomposition of members of the first; graphite not separating as such from solution in iron, but occurring through the decomposition of previously separated or separating carbide (neither is the direct separation from solution of a graphite eutectic near 1130° C. probable). Carbon dissolves in iron as carbide, and separates from solution as such."

Goerens[3] has shown this to be so in an exhaustive and most admirable research.

There are many reasons which lead to this conclusion. In general, when a molten metal has placed in contact with it a non-metal, there is a tendency to combination between the two, and it is the compound which passes into solution in the metal. The case of iron illustrates this point very well. Melted iron exposed to air forms iron-oxide which is soluble in the liquid metal. Sulphur in contact with melted iron forms iron-sulphide, and this sulphide is dissolved; phosphorus forms the phosphide which dissolves; silicon forms the silicide, and the case of carbon is similar.

That is, carbon in contact with molten iron forms a carbide, and it is this carbide which, dissolving in the liquid iron, gives rise to cast irons and steels.

Conversely, on cooling, it is the carbide which first separates. Thus it will be seen that cast irons and carbon steel may well be regarded primarily as a series of alloys of iron and iron carbide, and that a constitutional diagram can be constructed to represent them.

This is shown in Fig. 1, and, to all intents and purposes, indicates what separations primarily occur in the course of solidifications and cooling; with the proviso always borne in mind, that such carbide is unstable, and tends to decompose to iron and graphite during and after separation.

If, as is surely always the case, one rarely thinks of carbide in cast irons without remembering the fact of its ready decomposition, such a diagram becomes at once capable of application intelligibly to the alloys of every-day practice.

It will be seen that this diagram corresponds in general tenor to the equilibrium diagram of iron-carbon alloys. This is but natural since separation of constituents is indicated in either; but the whole of the constitutional diagram of Fig. 1 has reference to the solubility of iron-carbide in iron, either solid or liquid, and indicates at what temperature it separates from solution. (See Appendix.)

The interpretation of this diagram is as follows:

Alloys in the molten state are to-day regarded as being largely analogous to ordinary liquid solutions, the more of a substance that is dissolved in the solvent the lower the freezing point of the solution becomes.

In the case of iron and carbon alloys, as the quantity of carbon increases, the melting or solidifying point becomes lower, so that, commencing with pure iron, which melts at 1505° C., as the quantity of carbon increases up to about 4.2 per cent carbon, the metal has a lower and lower melting point, until, with this 4.2 per cent carbon alloy, we have a melting point of about 1130° C. This is the alloy of the lowest melting point of the whole series, and it is known as the eutectic alloy. These facts are represented along line *AB* of Fig. 1.

[3] "On the Present State of our Knowledge of the Iron-Carbon Alloys," *Metallurgie*, 1906, vol. iii, p. 175; 1907, vol. iv, pp. 5, 173.

As to the case of a cooling salt solution, it is excess water (ice) which crystallizes out first while the salt becomes stronger and stronger in the residual brine, so in the case of high iron-carbon alloys excess iron solidifies out of the solution first, most of the carbon as carbide remaining and gradually concentrating in the residual mother liquor.

But when the temperature has fallen to about 1130° C. this mother liquor also solidifies. It has remained liquid as long as it was able, but since this temperature (about 1130° C.) is the solidifying point of the most fusible alloy (the eutectic) of the series, it too becomes solid at this temperature.

On comparing the high iron-carbon alloys with a salt solution we have to notice an important distinction. In the latter case, pure ice tends to solidify first, while the salt remains dissolved in the brine, but in the former case the iron which first solidifies is not pure iron. Iron has the power of keeping

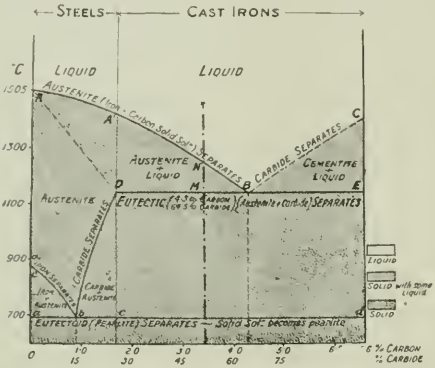


FIG. 1.—CONSTITUTIONAL DIAGRAM OF CAST IRONS AND STEELS.

some of the carbon dissolved in it, even when it is solid, to the extent of about 1.7 per cent carbon. Up to this percentage, therefore, the material solidifies as a homogeneous mass of crystals of iron containing the carbon in solid solutions, and no carbide is left behind to concentrate in a mother liquor, since it is all dissolved in the iron as it solidifies. To this solid solution the name "austenite" is given.

Its practical importance will be realized when it is seen (as may be observed from the diagram Fig. 1) that this range of alloys in which the carbon may be held wholly in solid solution constitutes steels. Beyond this proportion, where the solidifying iron cannot carry more than 1.7 per cent of carbon, some carbide has to remain in a mother liquid, and this separates later (along *DE*) as the eutectic of iron (that is, austenite) and residual carbide at about 1130° C. This range corresponds to the series of cast irons.

Steels.

It has just been seen that the range of carbon steels comprises those alloys in which the iron has solidified, with the carbide dissolved in it in solid solution (*AA'*, Fig. 1); that is, containing up to about 1.7 per cent of carbon.

If, now, this carbide can be retained dissolved in the solid steel a very hard material is obtained, and the hardening of carbon steels by quenching is simply the operation of cooling the metal so very quickly that the carbide does not get time or opportunity to separate from the solution as such, but remains dissolved in the solid metal. Quenching has to be carried out for reasons which can be understood on reference to the constitutional diagram (Fig. 13, and depends on the following data:

It is a fact well known, and practically universally recognized, that pure iron exists in a number of allotropic forms, each variety having certain properties which mark it as different from the others. It is beside the purpose of the present paper to discuss this matter in any great detail, but it will suffice to say that for all practical purposes the form of iron known as gamma (γ) iron, which exists above the temperature corresponding to the arrest, Ar_3, has the power of dissolving carbon, and that the form known as alpha (α) iron, into which it eventually changes on cooling, does not possess this dissolving power. On cooling a carbon steel, therefore, a certain temperature is arrived at, at which there are present carbide of iron and a form of iron which are mutually insoluble.

Therefore, just as ice or salt must separate from a solution which can no longer dissolve it, so on cooling beyond a certain stage must this α-iron (known as ferrite) or this carbide (cementite) separate from the original solid solution (termed austenite).

This will be more readily followed perhaps by considering what occurs while a mild steel or a high-carbon steel cools down slowly. Attention may first be drawn to the very close analogy between the solid solution of carbon in γ-iron and an ordinary liquid solution (say, of salt in water), bearing in mind that just as salt is insoluble in solid water (ice) so is iron-carbide insoluble in α-iron (including for the present purpose β-iron under this variety). This is shown in Fig. 2. The parallelism is marked if it be remembered that

Iron-carbide is soluble in γ-iron.
Iron-carbide is *not* soluble in α-iron.
γ-iron changes to α-iron at 900° C.
The change point is lowered down to 690° C. by iron-carbide.
Salt is soluble in water.
Salt is *not* soluble in ice.
Water changes to ice at 0° C.
The change point is lowered down to — 23° C. by salt. (Fig. 2.)

Mild Steel (say, 0.24 per cent carbon).—This solidifies at 1492° C. to 1448° C.[4] as a homogeneous mass of crystals—austenite—iron carrying the carbide dissolved in it.

On cooling down it remains in practically the same condition until about 890° C. is reached. Now the α-iron begins to separate out from the solution, as pure water-ice would from weak brine. As the temperature falls still further, more and more α-iron separates (as more and more ice would), while the carbon concentrates in the residual solid solution (salt doing the same in the residual briny mother liquor) until at 690° C. a temperature is reached where it is impossible to keep the α-iron and carbide in mutual solution, so that this residual solid solution splits up into its constituents of α-iron and carbide side by side, forming pearlite, in the same way as at the eutectic temperature (— 23° C.) the water and salt can no longer remain in mutual solution, but splits up into its constituents of ice and salt forming the eutectic.

In the steel, however, this separation has taken place from the solid solution instead of from the liquid solution, and to distinguish the two cases the constituent in the steels is termed the eutectoid. The name given to this constituent of steels is pearlite simply from its pearly appearance when seen under moderate magnifications, and is due to its consisting—as most eutectics do—of finely interstratified laminæ or plates. This pearlite is found by analysis to consist of 0.9 per cent carbon, and is a mixture of 13.5 per cent iron carbide and 86.5 per cent iron.

In the light of the above we can thus readily interpret the cooling and separation sequence in steels. The case of mild or low-carbon steels ("unsaturated steel" of Arnold) has just been stated.

With the high-carbon steel (hypereutectoid of Howe, or supersaturated of Arnold) carbide in excess of the eutectoid proportion separates out first (*bl*) of Fig. 1), the remaining mother solid solution becoming richer and richer in iron contents until the eutectoid proportion and temperature are again reached, when carbide and α-iron separate out alongside as pearlite

[4] Carpenter and Keeling, "Range of Solidification and Critical Ranges of Iron Carbon Alloys," *Journal of the Iron and Steel Institute*, 1904, No. I, p. 232.

The operation of quenching is to hurry the cooling materials so quickly past these critical ranges as to tend to keep them in the condition prevailing at the higher temperature—that is to say, to retain the carbide in the form of a solid solution, or something equivalent, by preventing the separation of the constituents which would otherwise occur. It is thus found that the microscopic character of such quenched materials corresponds somewhat to that revealed by etching at high temperatures.

It may be further explained here that the terms martensite, sorbite, troostite, etc., are the names by which those appearances are distinguished when instead of the drastic quenching just indicated, the methods of cooling is so modified as to allow of a partial separation from solid solution. Thus sorbite is an unsegregated pearlite, due to the quenching operation being carried out in such a manner as not to have permitted the eutectoid the opportunity of separating out in interstratified laminæ. By rather rapid cooling through the critical range the cementite of the eutectoid is seen in small granules instead. Troostite, again, is a solid colloid solution of cementite in iron—in other words, a pearlite having ultramicroscopic particles of cementite.[5]

It is thus possible to have a whole series from the austentite solid solution through the colloidal stages down to well-laminated pearlite, by modifying the heat treatment. By prolonged annealing, indeed, the carbide of the pearlite may coalesce into separate flocks in a matrix of ferrite.

It will, therefore, readily be understood how such modifications of heat treatment, which affect the manner and extent to which the carbide separates from solid solution, must in their turn modify the properties of the resulting products, giving us the range from harder and more brittle to softer and more ductile material.

Finally, it may be remarked that the explanations given above are simply a résumé of innumerable observations and results reduced to the lowest possible terms, so as to give a practical explanation to this very complex and intricate mechanism during cooling. There is much that might be added to amplify fully these views. The object of the paper is, however, to place such a construction upon the accumulated evidence that both scientific and practical men can, with a greater amount of ease, develop the explanation along the same lines to meet special cases or points of detail which are of particular interest.

Cast Irons.

It will be remembered that in the case of steels, whereas mutually insoluble carbide and α-iron represent the eventual stable conditions, the steel solidified first as a solid solution—a meta-stable form. Similarly in the case of cast irons, where as graphite and iron represent the eventual stable condition, the cast iron solidifies or tends to solidify as iron-carbide and iron—again the meta-stable form.

The science of carbon steels is the study of the conditions accelerating or retarding this change from meta-stable solid solution first obtained to the stable forms, in order to control the properties of the material and render it suitable for the particular purpose in view. In a similar way the science of cast irons is the study of conditions which accelerate or retard the change from the meta-stable iron: iron-carbide alloys tending first to be produced to the stable iron-graphite system—again in order to control the properties of the metal and to render it suitable for the special purpose for which it is to be employed.

In both cases, the properties of each end of the series are known; it is a matter of accumulated experience with equilibrium partially restored which enables the worker to know what constitution of metal is required for any particular case.

It is important, therefore, to consider generally what are the conditions which accelerate or retard this change from the primary meta-stable form on separation to the secondary stable form; and it will be found that in the main the general conditions are the same for cast irons as for steels.

Conditions tending to bring about or restore equilibrium—

1. *Slow Cooling* gives time and opportunity for changes from unstable to stable condition to occur. Examples in the case of the slowly cooled steels and cast irons are familiar.

Prolonged retaining of the materials at or a little below the critical range of temperature has the like effect.

2. *Reduction or Relieving of Pressure* tends to restore equilibrium.[6]

3. *Modification of Chemical Composition.*—It is possible by alteration of chemical composition to produce the stable forms from solution, or by decomposition.

Conditions tending to prevent equilibrium—that is, tending to retain materials in the unstable form—are, conversely

1. *Rapid Cooling or Quenching.*

2. *Increase of Pressure.*

3. *Modification of Chemical Composition* such as may tend to increase the solubility or to prevent decomposition of the chemical compounds originally existing in the meta-stable form.

Reverting now to the case of cast irons. They are regarded as alloys of iron and iron-carbide, the latter constituent being prone to decomposition during or after solidification, giving rise to graphite and a corresponding large amount of free iron.

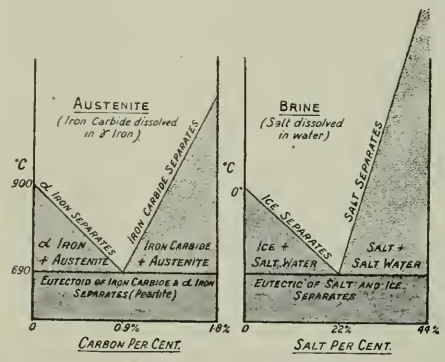


FIG. 2.—SHOWING ANALOGY BETWEEN SOLUBILITY OF IRON-CARBIDE IN α-IRON, AND OF SALT IN ICE.

The formation of gray irons by slow cooling and the formation of black-heart castings are readily understood, while the connection between heat treatment and constitution is apparent.

This conception necessitates a new and important way of regarding cast irons. Take, for example, a metal containing 3.5 per cent of total carbon. If this be regarded as primarily an iron:iron-carbide alloy it really consists of

$$3.5\times\frac{Fe_3C}{C}=3.5\times\frac{(3\times56+12)}{12}=3.5\times\frac{180}{12}=3.5\times15=52.5$$ per cent iron carbide.

before and at the instant of solidification.

That is, in cast irons the combined carbon should not be regarded, so far as its effects on the metal are concerned, as so much per cent combined carbon, but as 15 times that percentage of iron-carbide. This point of view will at once make clear the meaning of many observations and results, some of which have hitherto been more or less obscure. For instance, that 3.5 per cent of combined carbon in a white iron should be unstable does not appear to the imagination so forcibly as does a metal consisting of no less than 52.5 per cent of meta-stable iron-carbide.

Its ready decomposition by heat appears much more understandable, as well as the marked difference in properties accompanying even such seemingly small changes as, say, 0.1 per cent of combined carbon converted into graphite. Such an

[5] C. Benedicks, *Journal of the Iron and Steel Institute*, 1908, No. 11, page 255.

[6] C. Benedicks, *supra.*

alteration, however, really means the disappearance of as much as 1.5 per cent of the hard, brittle iron-carbide and the simultaneous setting free of 1.4 per cent of soft, ductile, free iron. Changes in carbon condition in reality are magnified no less than 15 times.

The Solidification Sequence of Cast Irons.

It will be seen from Fig. 1 that up to about 4.3 per cent carbon (64.5 per cent carbide) austenite separates first from the liquid metal.

Taking as an example a cast iron of the usual carbon contents of 3.5 per cent, the austenite separates at a temperature of about 1230° C. (*N*, Fig. 1) in the dendritic form usual in such cases. As the metal cools down these dendrites grow as more and more austenite separates, while the excess carbide remains behind, becoming more and more concentrated in the residual mother liquor. Eventually at 1136° C. (*M*, Fig. 1) this remaining liquid solidifies in between these dendrites in the form of an eutectic of austenite and iron-carbide, containing 64.5 per cent of the latter constituent (= 4.3 per cent carbon). The whole mass is now solid. On further cooling the austenite portion, containing about 13.5 per cent carbide (1.5 per cent carbon) in solid solution, gradually deposits its excess of cementite at its crystal borders, until just above Ar (690° C.) there is only 0.9 per cent carbon left in solid solution. At this temperature, as has been already seen, the pearlite change occurs, the solid solution finally depositing α-iron and carbide alongside each other in the form of the eutectoid pearlite. In such cast iron, therefore, the dendritic pearlite (surrounded with its border of cementite) marks the position of the primary austenite crystals, which first separated at *NM*.

A cast iron containing 4.3 per cent carbon (64.5 per cent carbide) tends to solidify wholly at 1130° C. as an austenite-cementite eutectic.

With cast irons containing more than 4.3 per cent carbon the first constituent to solidify would be the iron-carbide in excess of the eutectic proportion; then at 1136° C. the austenite-cementite eutectic separates as before, and the subsequent changes take place as already indicated. Now up to 4.8 per cent of carbon such results can be obtained with pure iron-carbon alloys on quenching.

Reference might be suggested to the numerous and excellent photographs of Benedicks, Goerens, Heyn, Howe and Tieman, Longmuir and McWilliam, Sauveur, Stansfield, Stead, Wüst, and many others.

Under ordinary cooling conditions, however, such results are not obtained, graphitic irons resulting, the reason being due to the ready decomposition of the carbide, particularly at high temperatures. The conditions which tend to accelerate such a change have already been pointed out, and are well known, but the exact mechanism of this change can now be brought to mind in a simpler manner than has hitherto been possible.

The tendency to decomposition of a metal containing something between 55 to 65 per cent. of unstable carbide during solidification will be apparent, especially bearing in mind the very high temperatures prevailing; and there is the further well-known fact that once the meta-stable condition is broken down, restoration to equilibrium proceeds much more rapidly and completely, so that in the present instance, once graphite has been produced in the metal, the production of further quantities rapidly follows.

With regard to the mechanism of this change, it is well known, and has often been recorded, that even in the solid condition, especially at high temperatures, the various constituents of alloys tend to migrate and coalesce into larger and larger masses[7] Even the cementite constituent of pearlite behaves in this way It has been further shown by Arnold[8] and McWilliam that eventually such large masses of cementite decompose, graphite being produced in the centre of the areas, whilst surrounding it is the iron, which is the other product of this decomposition of the carbide. It is this process which appears to go on in the formation of secondary graphite on slow cooling.

Further than this, the production of kish in high carbon irons, the formation of primary graphite, as well as the limit to the total carbon contents of ordinary cast iron, can be readily understood from this point of view.

During the actual solidification in such irons, the carbide which is in process of solidification tends to decompose. This iron-carbide has a density nearly similar to that of the liquid metal, so that, could it remain undecomposed, it would stay in the position in which it had been deposited. But graphite has a density of only about 2.0, so that being so much lighter than the liquid metal, it tends to rise to the surface, forming kish, and by this means the total carbon contents of the melt is decreased to this extent.

Since, however, such graphite is surrounded by a corresponding amount of iron from the Fe_3C, there may be some tendency for it to be held back in the material, this depending largely on the composition, temperature, fluidity and other conditions of the metal. An important consideration is that the dendrites of primary austenite which form first would help to entangle this graphite.

It will be readily seen why iron-carbide alloys containing more than, say, 4.5 per cent of carbon cannot be obtained under ordinary working conditions. Not only would such strong solutions of 68.0 per cent and upward of carbide in the liquid condition readily decompose at these high temperatures (much as strong sugar solutions, such as syrups, do), but on solidifying carbide would separate first, would instantly decompose, and the lighter graphite immediately rising to the surface of the liquid mass would lower the total carbon contents.

Below the eutectic proportion of 4.3 per cent carbon this does not occur so readily, but owing to this decomposition the tendency remains to lowering of total carbon contents by this means, though by the action of the primary austenite dendrites indicated at the left-hand side of the eutectic proportion such a tendency is checked mechanically.

The considerations dealt with above have had reference chiefly to the connection between the temperature and constitution of cast irons. The effect of pressure has not been investigated so fully, but such results as have been obtained bear out the views as stated previously.

The effect of chemical composition is most important from the practical point of view. The cast irons in practice contain considerable amounts of other elements, such as manganese, silicon, sulphur, and phosphorus; and these constituents, or at least the first three, in the quantities in which they are present in cast irons, exert a profound influence on the relations between carbon and iron.

One point deserves special attention. Sulphur, silicon, and phosphorus do not exist, as such, in cast irons, but the sulphur combines with any free iron, which may be present to form iron-sulphide, the silicon also combining with iron to form iron-silicide, Fe_2Si, in cast irons of ordinary composition, while the phosphorus similarly forms a phosphide. The first effect of these constituents (apart from reactions with manganese) is therefore to decrease the amount of uncombined iron which otherwise separates as primary austenite.

The Effect of Foreign Elements on the Condition of the Carbon.

Bearing in mind the fact that the carbon in the iron-carbon alloys tends to separate as carbide according to Fig. 1, and that this carbon tends to decompose into graphite and iron, the action of the common elements in commercial cast irons may, for the present purpose, be summarized as follows:

A. Tending to bring about Equilibrium.

To accelerate change from meta-stable carbide to stable graphite — Silicon.

[7] J. S. Stead, "Migratory Habits in Alloys," *Journal of the Society of Chemical Industry*, March 31, 1903.

[8] Arnold and McWilliam, "Thermal Transformations of Carbon Steels," *Journal of the Iron and Steel Institute*, 1905, No. II, p. 27

B. Tending to retard Equilibrium.

To prevent change from meta-stable carbide to stable graphite. Sulphur (by itself).
Manganese (by itself).

C. Neutral action.

Phosphorus
Manganese and Sulphur together
(when in such proportion as to form MnS).

Silicon.—Although the relations between silicon and the condition of the carbon have not yet been sufficiently studied chemically and physically to allow of reactions being stated with any pretension to exactness; generally speaking, silicon appears to act upon the combined carbon in cast irons in two ways, amongst others—

a. It lowers the solvent power of iron for iron-carbide (probably as Fe_2Si).

b. It probably tends to decompose carbide according to the following reaction—

$$Fe_3C + Si \quad Fe_2Si + Fe + C.$$

It will be seen, therefore, why the increase in silicon contents tends to lower the total carbon contents of cast irons, and leads also to the formation of graphite.

Taking as example a 4.2 per cent carbon cast iron. Melted with a moderate proportion of silicon, the carbide is thrown out of solution, is decomposed at the high temperatures prevailing, and a corresponding quantity of graphite is formed. On chilling the metal quickly, such graphite is retained *in situ,* and a "whitish" iron results, containing graphite according to the amount of silicon added. If such metal be kept melted for some time, and the graphite allowed to rise to the surface, and to thus separate as kish, then on chilling, a white iron results (if the silicon has not been too high). This iron is of lower carbon contents, however, because of the graphite kish separated (formerly in solution as carbide). If slowly cooled, such metal is gray, and also has a lower total carbon content.

It will be seen, even in this case, how largely the total carbon contents of cast irons depend on the stability of the iron carbide; for had this remained stable and unchanged on separating from solution apart from its decomposition by the silicon (see *b* above) the total carbon contents of the irons could not have been affected, because the separated carbide, by reason of its density, would have remained unchanged where it was, instead of separating out of the melt altogether as graphite, which rises on account of its lightness.

Many experimental facts[9] may be quoted in support of this view, which is here given in the most condensed form.

Sulphur and *manganese,* each considered separately, tend to retain the carbon in the combined form, preventing its conversion to graphite and free iron.

Sulphur as sulphide of iron acts mechanically and physically in such a way as to oppose the conditions which favor the decomposition of iron-carbide, from which graphite results; physically, by a lowering of melting points and by surface tension and mechanically, by forming sulphide films and envelopes around the cementite crystals. If these crystals, which separate as a constituent of the 1130°C eutectic, 4.3 per cent carbon (DE, Fig. 1) were free, they would coalesce and eventually decompose under favorable conditions. These minute sulphide films, however, having a high surface tension, tend to emulsify this cementite, thus retarding those necessary stages prior to its decomposition. It is, moreover, probable that these emulsifying envelopes of sulphide, acting in opposition to the expansion which accompanies the formation of graphite from carbide, still further retards the change.

The effect of manganese on the iron-carbon alloys is a subject to itself. For the present purpose it may be stated that in the quantities usually found in cast irons, manganese tends to the formation of carbides, which are very stable—as first readily shown by Arnold. In other words, while iron carbides are readily decomposed by heat into graphite and free iron, manganese carbides are not, but remain stable and unchanged; so that the carbon remains in the combined form.

This fact has important applications in practice in several ways. Owing to the stability of the manganese carbides in cast irons, such alloys are kept white and hard by additions of much manganese and, therefore, more difficult to soften by heat treatment alone—especially if the silicon contents be not high. Again, when high proportions of manganese are present, not only are the white, stable, and brittle alloys familiar as ferro-manganese and spiegeleisen obtained, but the carbon contents of such irons is much higher than in the ordinary metal which, as just seen, has a limiting value under ordinary conditions of 4.5 per cent carbon. This is because, even on heating to the melting point, the double carbides of iron and manganese are so stable as to be undecomposed; and on melting and slow cooling they again remain as carbides, white and unaltered.

Although the same principles as regard solution of the carbon as carbide exist as in the iron-carbon series, there is no secondary decomposition of carbide to consider. No matter, therefore, how high the carbon content be—up to about 5.5 to 6.0 per cent carbon in the liquid metal—since the carbide does not decompose on solidification, graphite is not formed, and hence does not separate from the metal as kish; so that the total carbon contents are not lowered at all by this cause.

It has been indicated recently that when manganese and sulphur are present wholly as manganese sulphide, this neutral body has but little action either way on the relations of the carbon and iron. Its tendency, if any, would be towards the retarding of equilibrium, in that some of the manganese sulphide tends to separate with the 1130° C. eutectic; so that some slight mechanical action might take place. Manganese added to a sulphurous iron or sulphur added to a manganese iron tends, therefore, towards neutralizing the retarding of graphite formation, caused by either acting alone. But either element in excess of the proportions required to produce MnS will naturally exert its own specific action—particularly sulphur in excess; because not only does this excess sulphur form iron-sulphide, which acts as already indicated, but this iron-sulphide carries some of the manganese sulphide with it to act together in this way, so that the effect of excess sulphur is much magnified.

The action of phosphorus may be followed from the valuable researches of Stead and Wüst.

The action of manganese and silicon together is not yet thoroughly investigated. Fletcher has, however, done much good work in this direction; and research is in progress bearing on the subject at the University of Birmingham at the present moment.

The study of the action of sulphur and silicon together is a stupendous piece of work, which has not been dealt with very fully so far, but which promises important practical results.

In conclusion, it will be recognized how, commencing with this conception of the primary condition of cast irons and carbon steels, it is possible to arrive at a systematic corelation of the effects of constituents and heat treatment in the industrial alloys. By standardization work, and by ascertaining the definite conditions which retard or accelerate the all-important changes from the meta-stable condition primarily existing to the final stable forms, it is possible to define the exact conditions necessary for the attaining of desired composition and of desired properties of metals for specified purposes.

Appendix.—The development of the iron-carbon equilibrium diagram.

Largely to the manipulative skill and the genius of Roberts-Austen, Standfield, and Le Chatelier, we owe the conception of metallic alloys as being closely analogous to ordinary saline solutions.

These workers were enabled from their freezing point

[9] Charpy and Grenet, *Metallographist,* 1902, Vol. V, p. 202; Wust and Peterson, Metallurgie, 1906, Vol. iii, p. 384; also Turner's papers and many others.

diagrams to place iron-carbon alloys also in this class, regarding irons and steels as solutions of iron and carbon.

Roozeboom's application of the phase rule introduced a general law governing these alloys regarded as an iron-carbon series, and especially valuable was his addition of the solidus curves, which assists in following closely the solidification sequence. This work resulted in the conception of a diagram representing the equilibrium between iron and carbon in these alloys. But since such a diagram can only represent components in their stable form, it dealt only with iron and carbon in the elemental condition; while in order to account for the occurrence of combined carbon in many cast irons it was assumed that at certain high temperatures iron and carbon were capable of combining and forming a stable compound iron-carbide.

This theory, contrary to all practical observations, was abandoned soon after. The admirable work of Heyn, Wüst, and Benedicks, chiefly brought out the idea that there were two systems of alloys in irons and steels—one a labile or stable system, the iron-graphite series; the other meta-stable or unstable, the iron-iron-carbide series. Both were represented on a single diagram with the usual liquids and eutectic lines. The explanation of this view is that although in the sense of the phase rule, iron and graphite are the stable phases of the series, the carbide, though unstable often shows a strong tendency to remain as such after solidifying in this condition.

The separation of the carbon in the combined form, as carbide, on solidification was considered as an abnormality, and as being due to its tendency to meta-stable equilibrium, and it was considered that, normally, carbon seperates from solution in iron in the graphitic form.

This view led to some vagueness and difficulty in giving definite reasons for the constitution of the cast irons met with in practice, and especially for the changes in the constitution brought about by alterations of chemical composition and by the rate of solidification and cooling; it being explained that slow solidification and presence of certain elements encouraged the solidification of carbon in the graphitic or stable form, rapid solidification and the presence of other elements, on the other hand, encouraging the stability of the meta-stable cementite on solidification.

In the case of cast irons and steels the rigid application of the phase rule is, however, barely possible and almost unwise, and the utmost caution is necessary in applying it in the present instance if it is desired to utilize it in connection with the problems of everyday practice. For the equilibrium diagram can only deal with phases in stable equilibrium, and it can consequently only give indications as to what results might be expected if graphite separated directly from cast irons, and if alloys of iron and graphite only had to be dealt with. This is, however, only the very ultimate condition of such alloys, and with this one has, generally speaking, little to do in practice.

The problems of practical working are those connected with the presence of cementite, and the conditions which tend to keep it in that form, or which change it from that form. If now we can consider cast irons and carbon steels as solidifying primarily as alloys of iron-carbide and iron, there is to hand at once a rational and definite method of considering what subsequent changes occur to this carbide which result in the alloys used industrially.

An equilibrium diagram is inapplicable to the case, because the bringing about of true equilibrium, which can only be effected by the decomposition of the unstable carbide to stable graphite, occurs over practically the entire range of temperatures covered by such a diagram. It is, moreover, enormously influenced by considerations of time, pressure, surrounding medium, etc., which effects can find no direct interpretation on such a diagram. On the other hand, if iron-carbide be regarded as the constituent which solidifies primarily from cast irons and steels, it is possible to consider in a positive and definite manner what conditions affect its decomposition, yielding the graphite and remaining carbide which are found in the commercial irons. It is the admirable experimental research of Goerens which has finally established this primary separation of iron-carbide as a fact, and therefore enables us, with confidence, to consider cast irons and carbon steels as being primarily alloys of iron and iron-carbide.

University of Birmingham.

Standardization of Metallurgical Terms.

The committee appointed by the council of the Chemical, Metallurgical and Mining Society of South Africa to consider the advisability of furthering the standardization of metallurgical terms and phrases, has made a preliminary report in which suggested changes are made. While many of the suggestions relate primarily to Rand practice, they are, nevertheless, of more universal application, and will tend to clarify technical writing.

1. Except when otherwise necessary, the following words shall be treated as collective nouns, and therefore used in the singular: Sand, tailing, fine, black sand, residue, ash, pyrite (FeS_2), concentrate, slag, sulphide, telluride, slime, chloride.

2. The following terms shall be used in the plural form to indicate the products: Shavings, skimmings, steamings, sweepings, drillings, reserves.

3. Preference should be given to the singular form in: Precipitate, bin, cost, profit, value.

4. Such abbreviations as the following shall not be printed in the plural: oz., lb., dwt., gr., cc., cm., kg., in., gal., m., gm., mm.

5. As far as practicable the following terms shall be avoided: Dirt, mullock or muck (broken ore or waste), kokopan (truck), cheesa stick (firing stick), lasher (shoveler), lashing (shoveling).

6. The following spelling shall be used consistently: By-product, shoot (the geological term), by-pass, hydraulicking, siphon, chute (passage or opening through which material is grid(d), kafir, gram, dike, rise.

7. Vat shall be used in place of tank; cyaniding in place of cyanidation; breaking and sorting in place of crushing and picking; crushing shall imply stamp-milling plus tube-milling; ore slime shall be used to distinguish from gold slime; extraction shall refer to metal extracted from ore; precipitation shall refer to metal precipitated from solution; recovery shall refer to metal won in a form suitable for market.

8. In referring to screening it should be stated whether number of holes refers to the square or linear inch, and the diameter of the aperture in decimals of an inch should be given; battery screenings should have the gage of wire in decimals of an inch and the percentage of discharge area mentioned, whenever possible. Temperatures shall be expressed in degrees Centigrade (with the equivalent in Fahrenheit in parenthesis). Cyanide and alkali strengths shall be expressed in terms of specified reagents, thus: 0.05 per cent KCN; 0.02 per cent NaOH; 0.02 per cent CaO.

9. A ton shall be 2000 lb. A fuid ton shall mean 32 cu. ft.

10. The following scheme illustrates the nomenclature of Rand reduction works:

- Reduction works
  - Breaking plant
  - Crushing plant.
    - Stamp mill — Sets of stamp batteries.
    - Tube mills or other secondary crushing machines
  - Cyanide plant
    - Sand plant
    - Slime plant

Zinc box precipitate may be refined with a solution of sulphurous acid as well as sulphuric. The action is a little slower, but it has the advantage of being inexpensive, and is adapted to isolated places.

Death of Oscar Guttmann.

It is with sincere regret that we have to record the death of Mr. Oscar Guttmann, the well-known expert on explosives, of 60 Mark Lane, London, E. C., which took place with painful suddenness at Brussels on Tuesday, Aug. 2. He was one of the British Jurors at the Brussels Exhibition and on Tuesday was being driven in a taxicab to the Gare du Nord, when his conveyance was struck by another automobile and he was thrown out and his skull fractured. He was conveyed in an insensible condition to the Hospital of St. Jean, but did not rally, and died within a few hours.

Mr. Guttmann was born in the year 1855, and at a comparatively early age showed his remarkable capacity for literary work of a technical character. His connection with the explosives industry commenced with the Hungarian Explosives Works of the Nobel Company, where he laid the foundation of the extensive knowledge which he afterwards showed in his professional duties. He was a recognized authority on all matters associated with the chemistry and manufacture of explosives, and the erection of factories and chemical works, and in connection with these subjects he was the author of a number of books, papers, pamphlets and lectures. In these matters he was considerably helped by his linguistic attainments, being able to speak and write in some six different languages.

OSCAR GUTTMANN.

From 1874 to 1885 he was technical adviser and constructor of plant of various large works in Austria-Hungary, including the St. Lambrecht Dynamite Works and the Budapest Gunpowder Works. He then undertook the management of an explosives works at Isleten (Switzerland), which he organized and brought to a state of prosperity, this factory supplying the dynamite used for the blasting of the St. Gothard tunnel. After a short time he was entrusted with the management of the Italian Dynamite Works at Avigliana, which he completely rebuilt after a disastrous fire, introducing many notable improvements in the construction of the buildings of the works.

In the year 1887 Mr. Guttmann came to England, and established himself as a consulting and chemical engineer in London, becoming naturalized in 1894. He soon acquired an extensive practice. His apparatus for the manufacture of nitric acid is well known, and there are many installations at work both in this country and abroad, the apparatus giving a quantitative yield of nitric acid of the highest concentration. Guttman's reaction tower for chemical works is equally well known. It is filled with hollow stoneware balls with cup-like indentations, or with hollow cellular bodies of rhombohedral shape, and ensures a most intimate contact of gases with liquids.

Mr. Guttmann designed and built the large works of the National Explosives Company at Hayle, Cornwall, for the manufacture of dynamite, blasting gelatine and cordite, the English Government's smokeless powder, which were subsequently managed by him for several years. He also designed and erected the Acetone Works at Waltham Abbey, Woolwich, Clapton and Manchester, the testing station for mining and safety explosives at Reichenstein, Germany; sulphuric acid works in Australia, India and Mexico, the latter also including concentrating and recovery plant, and many other chemical works of all kinds in the United Kingdom, Germany, Italy, Austria-Hungary, Belgium, France, Norway, Russia, Holland, Switzerland and the United States.

But he is best known by his classic works on the "Manufacture of Explosives" and on "Blasting," which have also been published in German. Recently he delivered before the Royal Society of Arts the "Cantor Lectures" on the "Manufacture of Explosives: Twenty Years' Progress." These have been published in German as a separate work, which is remarkable for its comprehensiveness and beautiful illustrations. As a member of the committee for explosives at the Seventh International Chemical Congress held in London last year he took a leading part in the preparation of a book entitled "The Rise and Progress of the British Explosives Industry," the idea of its preparation being due to him and the book being published on behalf of the committee. The articles on explosives in most standard works of reference, such as Muspratt's "Chemical Technology" and Lunge's "Technical Methods of Chemical Analysis" were written by him. Mr. Guttmann published a very large number of technical papers, most of which appeared in the *Journal* of the Society of Chemical Industry, in the *Proceedings* of the Institution of Civil Engineers, and in a large number of technical journals. His unique experience in explosives and chemical manufactures naturally caused him to be frequently called as an expert in law cases of a technical nature, some of these such as Maxim v. Anderson, St. Helen's Corporation v. the United Alkali Company being of far-reaching importance.

He devoted his spare time to the study of the history of explosives, and made frequent and extensive journeys with a view of examining rare manuscripts. In this branch of historical research he was an acknowledged authority. The result of these investigations appeared in technical journals from time to time as occasion offered, and they were recently collected by him in a magnificent folio volume entitled "Monumenta Pulveris Pyrii," and richly illustrated with prints from his private collection. His latest contribution to the history of explosives appeared only a few days ago in the *Chemiker Zeitung*, and in another paper now in the press he had described the safe construction of buildings in explosives works. It must be noted as characteristic of him that although he had patented this latter construction he allowed the free use of it to the public without demanding a royalty.

He was a member of the Institution of Civil Engineers, which awarded him the Telford premium for a paper on "The Blasting of the Iron Gates of the Danube," and highly commended a long paper elaborately illustrated with scale working drawings on "Machinery Used in the Manufacture of Smokeless Powders"; a fellow and vice-president of the Institute of Chemistry, and twice a member of council of the Society of Chemical Industry.

As a freemason he held a conspicuous position. He was W. M. of the Pilgrim Lodge, No. 238, and it was chiefly at his instigation, and afterwards due to his indefatigable exertions, that the combined meeting of the Anglo-Foreign Lodges took place with singular success last March. In April last the United Grand Lodge of England invested him as a past senior grand deacon, and though he highly appreciated the distinguished honor conferred upon him, and thereby upon the Anglo-Foreign Brethren, it is stated in the current issue of *The Freemason* that he specially requested that no particulars of his Masonic career leading up to this distinction should be given in its columns. In a brief summary of the united meeting, *The Freemason* writes: "There was a complete consensus of opinion that Brother Guttmann, as W. M. of the enter-

tanning lodge, Pilgrim, was the right man in the right place. Everything in word and action was so natural and unrestrained that it was a pleasure to witness the ease with which all was accomplished. Undoubtedly much of the success was due to his intuitive tact and ability."

His death will be mourned by a large circle of friends whom he was ever ready to assist, and by his younger colleagues, to whom he was ever ready to lend a helping hand or to give friendly advice, and many of whom now hold positions due to his influence. He was well known to many American chemists and manufacturers, several of whom are among his clients. He leaves a widow, two sons and one daughter. The latter has devoted herself to painting; his elder son, Dr. Leo F. Guttmann, was for three years instructor in chemistry at the College of the City of New York, and is now assistant professor of industrial chemistry and chemical engineering at Queen's University, Kingston, Canada: his younger son is a civil engineer. One may expect, therefore, that his sons will carry on their father's practice.

A large circle of friends attended the funeral, Freemasonry being represented by Sir Edward Letchworth, secretary of the Grand Lodge of England, and by delegates from the lodges Pilgrim, La France, Entente Cordiale, Deutschland, Italia, America and Prosperity, and from the International Masonic Clubs. The Society of Chemical Industry, Institute of Chemistry, Society of Public Analysts and Institution of Civil Engineers sent their presidents or vice-presidents, and nearly all members of council in town showed their personal loss by being present. Nobel's Explosive Company, Woolwich Arsenal and the British Board of Trade also sent representatives.

Among the numerous wreaths sent by the learned societies and lodges was one, which was particularly appreciated, bearing the words: "He was a perfect Mason, a loyal friend, earnest and sincere; a light has left us that will shine on in the great mansions above." It was sent by Mr. Van Duzer, P. M. Lodge America.

San Francisco Meeting of the American Chemical Society.

The San Francisco meeting of the American Chemical Society marks a milestone in the career of the society.

A plastic story of the trip of the Eastern members to San Francisco, from the pen of a distinguished chemical engineer and particularly good friend of this journal, will be found in the correspondence columns of this issue. An illustrated account of the only marring incident of the voyage—the wreck of the special train with the marvelous escape of all passengers—was already given on page 446 of our last issue. We herewith give a concise account of the San Francisco meeting itself.

On Wednesday morning, July 14, the forty-second general meeting of the American Chemical Society was called to order by President Bancroft in the St. Francis Hotel, San Francisco. After an address of welcome by Dr. Arthur Lachman for the California Section, responded to by President Bancroft, the following addresses were delivered in general session:

Dr. W. D. Bancroft: Positive photography. This lecture was illustrated with lantern slides.

Dr Edw. C Franklin: Liquid ammonia as a solvent and the ammonia system of acids, bases and salts.

Dr W. F. Hillebrand: Chemistry in the Bureau of Standards.

Dr H. E. Barnard: The use of sodium benzoate as a preservative of food

At the same time a ladies' reception was held in the parlors of the St. Francis Hotel.

After luncheon the members present and their guests enjoyed an excursion over the Ocean Shore Railroad to Half-Moon Bay and Tunitas Glen, returning in time for the smoker held in the Fairmont Hotel and the ladies' theater party. A hot supper had been promised at the smoker and all who partook of the tomales and heard the Chinese music were ready to acknowledge that the adjective was quite descriptive.

On Thursday morning the meetings of the divisions were held in the St. Francis Hotel, and many interesting papers were presented. The symposium on smelter smoke before the Industrial Division excited special interest.

After luncheon all attending the meeting enjoyed one of the pleasantest excursions of the trip, made by special train to the top of Mount Tamalpais and through the Muir Woods, and the members were entertained at a banquet during the evening on the top of the mountain. Several of the members remained all night, coming down the mountain on the following morning in gravity cars. This grove of redwoods, named after John Muir, is probably one of the most beautiful bits of scenery in the immediate vicinity of San Francisco, and the ride to the top of the mountain with the changing interest of foliage and panorama of hill, valley, bay and distant city of San Francisco was appreciated by all.

Friday morning was spent at the University of California, in Berkeley, mainly in an examination of the various buildings, laboratories and campus of this beautifully situated institution.

The party then took a special steamer as the guests of the Selby Smelting & Lead Company, being entertained at luncheon by the company and afterward conducted through their plant, where the various processes of lead smelting and the recovery of gold and silver therefrom were explained. One of the chief attractions of this plant was the opportunity given to view the new Cottrell precipitating apparatus installed for the purpose of removing sulphur trioxide and any other solids or liquids present in smelter smoke.

After returning to San Francisco the evening was spent in a visit to Chinatown, where at 10 o'clock all were entertained at a Chinese collation of tea and sweets served in a Chinese restaurant to music which the local committee characterized as sweet.

Saturday morning was devoted to divisional meetings at which the remaining papers on the program were read.

In the afternoon, at 2 o'clock, the members were treated to an automobile ride over Buena Vista Heights, through the Golden Gate Park to the ocean beach and the Cliff House, returning through the Presidio and the residential section of San Francisco.

In the evening the members all assembled for the main banquet of the week in the St. Francis Hotel, at which all of the ladies attending were guests. About 250 sat down to the banquet, which will long be remembered by all present.

On July 17 the party, as the guests of the Italian-Swiss Colony, took a special train to Asti, where an unsually pleasant day was enjoyed in examining the vineyards and wineries of this well-known section. The lunch was served outdoors in attractive pergolas. The party returned to San Francisco early in the evening and were given, almost for the first time, opportunity to sleep.

On the following morning, July 18, an excusion was taken by steamer up the Sacramento River to Sacramento through the wonderfully fertile fields of the Sacramento Valley, between levees so high that the party was obliged to view the country from the upper deck of the steamer. The general aspect was much like that of portions of Holland. Returning from Sacramento by train, the party reached San Francisco late in the evening, having been royally entertained. Many, however, took train at Sacramento for the north.

On Tuesday, July 19, a special steamer was provided for those who remained to visit and examine the various points of interest around San Francisco Bay.

Following the meeting in San Francisco, the members returned to their homes by various routes, but some 30 traveled northward by invitation of the Puget Sound Section to visit Seattle and obtain a view of the Northern Pacific Coast scenery. Arriving at Seattle on the morning of the 21st, the party was met by President Falkenburg, of the Puget Sound Section;

Mr. Horace G. Byers, councilor of the section; President Kane, of Washington University and others, who welcomed them as the guests of the Puget Sound Section.

On arrival the ladies were supplied with bouquets of dahlias and shortly afterward all started on an automobile trip which covered all parts of the city, both business and residential, and included the beautiful grounds of Washington University.

At the end of the drive the party were lunched at the Commercial Club, after which they immediately left on a chartered steamer for a trip around Puget Sound. The first stopping point was the plant of the Pacific Creosoting Company, where the party left the boat and inspected the largest creosoting plant in the world. Returning, a stop was made at the Navy Yard, where battleships and armored cruisers were examined, and then the party proceeded to Tacoma, where a delightful lunch was served under the enormous trees of the Tacoma City Park. By the courtesy of the Board of Park Commissioners the party was well supplied with roses and given permission to pick all the sweet peas they could carry away.

From Seattle some of the party came east via Vancouver and the Canadian Rockies, while others came over the Northern Pacific, visiting the Yellowstone National Park.

Unusual enthusiasm was shown throughout the whole meeting, and many new Western members were added to the society, which now has a membership of over 5000. Two hundred and ninety members and guests registered for the meeting.

One hundred and twenty papers were presented at the meeting, embodying new chemical research, many of them reporting important results.

Notes on Chemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

The Synthesis of Ammonia.

Up to the present time the liquor obtained in the manufacture of coal-gas or from the coke ovens has been the sole source of ammonia, and its synthetic production has not been regarded as possible of attainment commercially. Attempts are being made to prepare ammonium salts from cyanamide, but even if these should meet with some measure of success the process would probably be too costly to prove remunerative. The Badische synthetic method is independent of electric power, and now that both hydrogen and nitrogen are obtainable with facility at a low cost it is by no means unlikely that the electrochemical processes of manufacturing cyanamides and nitrates will find in it a strong competitor in the production of fertilizing material; but the available information hardly justifies a definite forecast, although there is apparently a fair prospect of success for the new process.

It is stated that the Badische Anilin & Soda Fabrik has solved the problem of the direct production of ammonia from its elementary constituents, and will probably soon effect the preparation of synthetic ammonia on the commercial scale. Professor Haber and M. R. le Rossignol, after much experimental work, found that at a high temperature and under a pressure approaching 200 atmospheres, the desired combination of hydrogen and nitrogen occurred and the action was increased by certain catalytic agents, particularly osmium, but, in consequence of the high price of this metal, uranium is used in practical working. The difficulties involved were great, but the Badische staff appears to have been successful in grappling with them. The mixed gases are compressed and brought into contact with the powdered uranium in a chamber where a pressure of 175 atmospheres is maintained at a high temperature, with the result that union of a large part of the mixture of gases takes place. The ammonia and the residual uncombined gases pass into a freezing apparatus whence the liquified ammonia is withdrawn, while the unchanged gases pass through and, with further quantities of hydrogen and nitrogen, are again treated in the pressure chamber. The heat evolved in the formation of the ammonia is utilized to assist in bringing the fresh mixture of gases to the requisite temperature. A satisfactory yield of ammonia is reported to have been obtained at the Badische works.

Calcium Cyanamide.

From the report of the Gesellschaft für Stickstoffdünger, of Knapsack, for last year, it appears that the condition of the cyanamide industry, at least in the district supplied by this plant, is not very flourishing. The Westeregeln plant was running nearly all the year working up the carbide supplied under previous contracts into cyanamide, but a fluctuating demand necessitated the stocking of a large proportion of the output at considerable expense. Working costs were much increased at Knapsack by the employment of only a small fraction of the capacity of the plant. Agriculturists were slow in extending the use of cyanamide, and the excessive establishment of new factories had resulted in a total output greatly exceeding the demand, although the total turnover recorded by the Sales Association had improved 35 per cent, and the demand during this spring had been greater. The production of ammonium sulphate from cyanamide had not yet progressed beyond the experimental stage. The capital was £180,000, and the gross loss amounted to £12,900.

Radium from Sweden.

The Swedish Radium Company has produced its first batch of radium bromide, weighing 5 centigrams, and is sending it to the Banque de Radium at Paris, with which bank the company has made an agreement for the reception of its entire output of radium. Gold tubes have been found deficient in resistance to the emanations, and the bromide is now secured in receptacles of platinum-iridium alloy. The "kolm" mined in Sweden has been found superior to radium-bearing material from Australia and America, both in respect of facility of manipulation and of content of radioactive substance.

Slags in Iron and Steel.

By experimenting with specially prepared oxides, silicates and sulphides of iron and manganese, M. Matweieff has devised a method of ascertaining the composition of slags enclosed in iron and steel. The silicates of iron and manganese are not affected by organic acids nor by heating in hydrogen or superheated steam. The oxides of iron and manganese are also not acted on by organic acids, but are reduced by hydrogen. If the reduced metal be tested with a dilute alcoholic solution of ferric chloride, manganese without iron gives no reaction; iron gives a feeble coloration, and iron with manganese gives a strong coloration. The sulphides of both metals are acted on by organic acids, but hydrogen and steam have no effect. Tartaric acid gives a strong coloration with sulphide of iron, but only exerts feeble action on manganese sulphide.

The Production of Rare Metals and Their Alloys.

M. P. Girod advocates the addition of a borate or of boric acid to the mixture of ore, reducing material and flux in the preparation of such metals as tungsten, tantalum, titanium, molybdenum and vanadium, claiming that a much more fluid mixture is thereby obtained from which the metal separates with greater facility, and that the metal is protected from oxidation by the coating of quite liquid slag. Valuable qualities are imported to the metal or alloy by the incorporation of a small proportion of boron, and by the addition of the requisite quantities of iron-titanium-boron alloy to ordinary steel special tool steels or constructional steels can be produced. This process is patented in France.

Electro-Deposition on Aluminium.

It is reported that a German firm has solved the problem of electroplating aluminium, and has been successful with copper, silver, gold and nickel, but details have not yet been published. The rapid oxidation of aluminium by alkaline substances has

restricted its field of utility, and if this defect can be removed —as it would be by copper plating—engineers will be enabled to avail themselves to a much greater extent than formerly of the valuable properties of this metal.

Regenerative Galvanizing.

The Cowper-Coles Galvanizing Syndicate has published particulars of the "regenerative" process, which consists in first depositing zinc by a special cold method and then applying a treatment which is claimed to produce a film of iron-zinc alloy which prevents rusting. The coating of zinc is said to be perfectly uniform and free from irregularities. In galvanizing by the hot process the threads of nuts and bolts are often filled up and the distribution of the zinc is almost always irregular. The new method is said to coat even fine-threaded screws without any re-tapping becoming necessary, and to permit the galvanizing of fine wire gauze without filling the meshes.

A New Conductivity Apparatus.

A portable apparatus for the rapid and accurate determination of the conductance of ordinary water and dilute saline liquids has been devised by Messrs. W. Pollard Digby and C. W. V. Biggs, who exhibited it and gave a demonstration at a meeting of the Faraday Society on May 31. It consists of a long-limbed glass U-tube furnished with platinum electrodes, and is so designed as to minimize polarization. The filling funnel and the outlet are attached to the lowest point of the bend, and above the electrodes are overflow pipes to permit of a constant flow of liquid. The tube is mounted on a strong wooden stand furnished with a thermometer. The "conductance meter" used with the tube is the familiar "megger" with a scale graduated in reciprocal megohms.

This apparatus, which can be used in the boiler house or in the open just as well as in the laboratory, provides the engineer and others with simple, rapid and exact means of control in many operations, such as checking the purity of boiler-feed water, the regulation of the softening of hard water, detection and estimation of condenser leakage and of priming, control of chemical oil eliminators, etc.

The addition of one part of sodium chloride in a million parts of distilled water raises the conductivity from 2.5 to 6.6 reciprocal megohms, and the same grade of distilled water has its conductivity raised to 3.5 reciprocal megohms by 0.1 per cent of tap water containing about 18.5 grains per gallon of total solids.

The Testing of Water by Electrical Methods.

The *Journal* of the Institution of Electrical Engineers has an original communication from Mr. W. Pollard Digby on the examination of water by electrical methods, with the apparatus mentioned in these notes above. The object of the author was to point out the value of simple tests taken *in situ* in safeguarding the owners of steam plants; in deciding when the chemical analyst should be consulted; ascertaining whether condensers are leaking or boilers priming, and controlling oil-eliminating and water-softening plants. These tests were not intended to supersede the analyst, but might serve to indicate when detailed analyses were necessary by detecting changes in the impurities in the water by determinations of its conductivity at any desired temperature and comparison of such determinations with the known value of analyzed samples. With a boiler free from priming a single measurement of the conductivity of the water passing from the condenser to the hot-well will indicate whether the condenser tubes are tight or not.

The author gave details of a considerable number of tests carried out at various power stations under widely varying conditions. The specific conductivity of the water in boilers was found to range from 111 to more than 90,000 reciprocal megohms, and consequently in each test for priming it was necessary to plot a curve of the conductivities of varying proportions of the water in the boiler under examination diluted with standard distilled water. A specific conductance of 11 reciprocal megohms indicates an exceptionally pure water containing only about 10 parts of saline matter per million, while the sample giving 90,000 was very highly charged with sodium carbonate. In most of the tests there was an almost entire absence of priming. An interesting curve showed the relation between hardness and conductance in the Clark process of softening water. As the milk of lime is added to the hard water the conductance steadily falls until just sufficient lime has been added and increases with any excess. The hardness of feed water is ascertained from the conductance with considerable accuracy.

Market Report.

JULY, 1910.

Copper has shown a downward tendency. Starting at £54 10 0 it dropped from the 8th till on the 12th it was under £53, but has since recovered and is slightly firmer at £54.

Tin has fluctuated considerably during the month. On the 1st it was £149 18 0, by the 11th it had dropped to £146 15 0, recovering on the 15th to £148 18 0. On the 18th it reached £149 5 0, and remains (26th) at £149 0 0.

English lead has been steady at £13, declining slightly to present price, £12 17 6.

Iron hematite opened at 64/8, rose on the 5th to 65 6, had dropped by the 7th to 64/4, and touched 63/11 on the 14th. Thence recovered to 64/3.

Scotch pig has been rather flat at 54/10, falling slowly after the 11th to present price of 54/7½.

Cleveland warrants have been between 48/10 and 48/8, the fall being steady from the 11th. Present price, 48/7½.

	£	s.	d.
Alum, lump, loose, per ton	5	12	6
Antimony, black sulphide powder	24	0	0
Borax, British refined crystal	10	0	0
Sulphate of ammonia, f.o.b. Liverpool, per ton	11	15	0
Sal-ammoniac, first, delivered U. K., per ton	42	0	0
Copper sulphate, per ton	17	10	0
Camphor, 1-oz. tablets	1	7	
Caustic soda, ash, 48 per cent, ordinary, per ton	5	10	0
Bleaching powder, 35 per cent, per ton	4	5	0
Shellac, standard, T. N. orange spots, per cwt	3	15	0
Sulphur, recovered, per ton	5	0	0
Carbolic acid, liquid, 97 to 99 per cent, per gal		11½	
Hydrochloric acid, per cwt	5	0	
Creosote, ordinary good liquid, per gal		2⅛	
Naphtha solvent, 90 per cent at 160° C., per gal	1	1	
Platinum, per ounce, nominal	6	10	0
Copper ore, 10 to 25 per cent, unit	8/9 to	9	9
Tin ore, 70 per cent, per ton	5	14	0
Mica, small slab, in original cases, per lb	6d to	1	0
Litharge, flake, per ton	15	10	0
Red lead, genuine, per ton	16	0	0

India rubber remained steady for the first half of the month at 10/0½, rising slightly to 10 1½ by the 13th. It then declined and stands at present at 9 6.

Of the chemicals it will be noticed that sulphate of ammonia is quoted slightly higher, shellac a little cheaper and platinum higher than last month.

London, July, 1910.

In making screen analyses care must be taken to insure the complete separation of the slime which, in the dry state, adheres to the larger particles of ore. A combination wet and dry method of making these analyses gives excellent results. A convenient weight of the sample is dried and then washed in the finest screen it is proposed to use. The residue on the screen is then dried and the operation repeated on the next coarser screen, proceeding thus until the grading is completed. This method gives great accuracy and allows no loss by dusting. Time is saved in spite of the repeated dryings and washings, as the end of each screening operation is quickly noted. The method is well worth trying.

Synopsis of Metallurgical and Chemical Literature

Elastic Breakdown of Certain Steels.—In a recent Iron and Steel Institute paper of C. A. M. SMITH the usual methods of tension tests are first discussed. The stress at elastic breakdown is generally obtained by dividing the total pull on the specimen in the testing machine by its area. This, however, does not give in general the maximum stress, but the mean stress. In order to obtain the maximum stress at elastic breakdown it is necessary to obtain the measurements of longitudinal strains in three places. Then if the material is elastic and obeys Hooke's law, it is possible to calculate the maximum stress upon the material during a tension or compression test. A great deal depends upon the method of gripping specimens. Detailed experiments along these lines showed that the strain distribution differs widely at different parts of the specimen and in different tests. When using V-grips it seems that the load should be taken up a few tons and then back to zero before carrying out the test proper. Although this unsatisfactory action of V-grips is generally recognized, it is frequently asserted that all the difficulties are overcome by the use of spherical seats. But by a detailed analysis of the stress distribution when using the ball seat, the author shows that friction plays an important part. The bearing stress on the spherical seat during a tension test becomes very great, and any lubricant used becomes squeezed out. The author then comments on the lack of uniformity of material under test; in case of a special steel he proves that it was not uniform. He then compares at some length the effect of time and increase of load, immediately after elastic breakdown, on the different steels tested. Finally, he remarks that until recently all of our knowledge of static testing was obtained from a consideration of one test; he then discusses the more interesting problem: what is the effect of a second stress? He made experiments with certain kinds of steel, a direct stress being obtained as a result of a tension or compression load, and a shear stress by means of a torsion load. The results are plotted in form of curves. They seem to suggest that the materials which most closely obey the shear stress law (or Guest's law) are those which show a decided elastic breakdown at a certain test. An appendix is added on the strength of pipes and cylinders.

Crystallography of the Iron-Carbon System.—A recent Iron and Steel Institute paper by ADOLPHE KROLL, JR., gives an extended discussion of this subject. The principal results are summed up as follows: "The polyhedral structure of beta and gamma ferrite has been shown for the first time by heat-tinting, the allotropic theory of Osmond being thereby confirmed. The iron-carbon stability diagram is considered from the legitimate standpoint of crystallographically uniform but mechanically dissectable phases, on the basis of the observation that a mixed crystal continues to grow in either of its components separately. It may be taken as established that, in the solid state, the processes are at first of a purely crystallographic character and independent of equilibria of diffusion (which have hitherto been regarded as the main criteria). The dissolving, or precipitating, phase is the seat of the crystalline force in all cases. The progressive growth of the martensitic austenite bodies in alpha and beta iron, accompanied by their modification into and deposition as gamma iron, was demonstrated by heat tinting. Similarly the continued growth of the same polyhedra of martensite into the cementite—accompanied by powerful transformation phenomena in the latter—was determined; and from this the existence of a solution-carbide, crystallizing according to the regular system, was deduced. Quenching experiments enabled pseudomorphs of this transformed and deposited cementite to be identified, these occurring as elongated cubes, in contrast to the hexagonal crystalline form of the cementite.

"Grounds were discovered for the assumption that the regular solution-carbide strives after saturation with gamma iron, thus forming the constituent troostite (the x constituent). Consequently, the existence of a system of partial isomorphism was recognized, to which belong the solution-carbide and the unstable troostite-saturation line, this system being established, in an intermediately free state, during tempering. It was recognized that the hexagonal-cementite system proceeds, in a partially cryptodimorphous manner, from the isomorphous system, and enables the crystallographically stable equilibrium to find expression. Consequently, a crystallographical double diagram had to be established as stability diagram, from the iron-carbon alloys, the said diagram allowing all the constituents of the isomorphous system to appear intermediately as unstable tempering—or deposition products previous to the complete establishment of the crystallographically stable equilibrium of the cryptodimorphous system.

The independence of the chemical processes with relation to the crystallographic stability diagram was recognized for certain cases, and, therefore, the contingency of primary or secondary establishment of a special purely chemical equilibrium. The existence, in iron-carbon alloys, of a purely chemical system differing in its nature and form from the two aforesaid purely crystallographical stability diagrams, was recognized. This system adheres formally to the lines of the former (Charpy-Heyn) diagram of graphite solubility, while following the carbide decomposition reaction of Wüst and Goerens. This special, chemically stable equilibrium is established in accordance with special, and always chemically homogeneous phases, the result being achieved in such a manner that the partially isomorphous system, is, for the most part, chemically unstable while the partially cryptodimorphous system is chemically metastable, and, therefore, the crystallographical double system may be entirely mutilated, chemically. (From advance sheets.)

Homogeneity of Metals; Rails.—A recent Iron and Steel Institute paper of GREGORY TAGAYEFS, of St. Petersburg, emphasizes the importance of studying the microstructure of metals for practical engineering purposes. There exist data showing that all values obtained from mechanical tests of different specimens of iron and steel are in harmony and that the curves of the different properties of the specimens, arranged in the series of any one value—for instance, hardness, tenacity, or carbon content—are parallel to each other. This circumstance often renders it possible to ascertain the character of the metal by one test; for instance, by simple tensile stressing. Such a method of procedure, however, only yields a reliable result under the condition that the treatment and the homogeneity of all the specimens shall be the same. Of these two conditions non-homogeneity is in practice the more dangerous, owing probably to the circumstance that in works the possibility of glaring irregularity of treatment is eliminated. The idea of homogeneity is introduced by the theory of elasticity in the considerations of isotronic bodies. But an identification of metals with bodies that are isotropic ignores that complication of structure which is revealed by metallography. The author discusses at some length the influence of non-homogeneity on fracture due to static and dynamic stresses. The influence of non-homogeneity is undoubtedly greater under impact stresses than under static stresses. It is especially important to notice this in connection with repeated light blows. In this case, intrusions of sulphide of manganese, under the influence of constant vibration, gradually develop into small cracks and promote fracture; whereas in statical fracture the presence of sulphide of manganese does not manifest itself in any way. The author's observations on rails gave him the opportunity of noticing this important action of sulphide permeations, consisting chiefly of

manganese sulphide, especially in those cases where it fills up cracks or is observed as irregular markings on the outside. The author finally refers to an exhaustive investigation of rails taken from Russian railways by the Rail Commission and to a supplementary report by himself, the general conclusions of which are as follows:

Data derived from the study of the chemical composition, in connection with the wear of rails, do not afford an explanation of the difference in their service. Observation of the service of rails and of other articles (shafts, axles, propeller screws, etc.) indicates that local defects in the metal have a predominant influence on their endurance. Flaws and defects observed in rails are particular evidences of non-homogeneity of their internal structure. In order to explain this property it is necessary to call in the aid of metallography which studies the structure of metal.

Structure of steel is determined: (a) by the particular arrangement of the different constituents of the alloy (phases) resulting from its physico-chemical equilibrium as represented graphically by melting curves; (b) by the external features of these structural elements (form, size and distribution) resulting from the so-called structural equilibrium, which depends upon the conditions of initial temperature, rate of cooling, cooling temperature, and external mechanical influences, the sum of which constitutes the mechanical and thermal treatment. A comparison of the results of fractures, with data from microscopical investigation, can be of great use. In particular, such a comparison confirms the existence of a constant definite correlation between the fracture and the microstructure of steel. The mechanical properties of steel are the resultant of the whole of the properties of the structural elements which the steel contains. The mechanical properties of a metal and its quality as measured thereby, are functions of the structure. For this reason the structure affords an indication of the strength and endurance of a manufactured steel article.

Casehardening.—A paper presented by Prof. Leon Guillet of Paris before the International Metallurgical Congress in Dusseldorf deals with "cementing considered from the practical and theoretical standpoint." Superficial carburizing is effected with the object of obtaining pieces of machinery with a very hard outer surface and a tough inner portion or core. The main factors which come into play are the steel to be treated, the cementing medium, the temperature and duration of the process, and finally, the thermic treatment of the pieces to be so dealt with. The steel suitable for cementing should contain, unless it be a special steel, not more than 0.120 per cent C, not more than 0.3 per cent Mn and not more than 0.3 per cent Si, and should have in its annealed state a breaking strain of not more than 38 kilograms per sq. mm. (24.13 tons per sq in.) with a 30 per cent elongation; and after hardening without a subsequent tempering down a breaking strain of 65 kilograms per sq. mm. (41.27 tons per sq. in.). The author gives in his paper the grounds for these conditions. As to the cementing medium he says that this has to be chemically determined and consists mainly of a mixture of charcoal and barium-carbonate. According to the latest experiments of Charpy, this mixture acts solely owing to the formation of carbon monoxide. Recent investigations have afforded a clear proof of the non-cementing property of carbon. The author lays special stress upon the two properties of a cementing medium, i. e., this should not have a too violent action since there would occur a too heavy formation of cementite needle crystals, and it should not be used up too easily. With respect to the question of duration and temperature, the author has collected in tabular form figures showing the influence of these factors. He then deals with the practical side of the question, in regard to the temperature at which cementing commences, and adds that varying views are entertained on the point. He then enters into the question of thermic treatment in detail, for it is in his opinion a fundamental one. The final result is largely dependent upon this treatment. He adds the two following rules which are now followed in modern installations:

1. Cementing at 850 deg. Cent. (1562 deg. Fahr.), quenching once at 750 deg. Cent. (1382 deg. Fahr.)

2. Cementing at 1000 deg. Cent. (1832 deg. Fahr), quenching twice, the first time at 1000 deg. Cent. and the second time at 750 deg. Cent.

After stating the above and giving prominence to the influence of the cementing temperature upon the brittleness of the core previous to the thermic treatment, he adds three important points, as follows:

1. The double quenching carries with it in any case an important advantage, since it causes the cementite which had taken a needle crystallization to disappear.

2. The double quenching leads to the formation in the case of a much more uniform texture, proceeding from a uniform martensite which has tempered down extensively.

3. The brittleness of the core after cementing is the same as when the metal has been subjected to a single tempering action.

Among the tests made with cemented pieces, the determination of hardness by the ball rebounding test is specially referred to. Annealing during five minutes at a temperature of 200 deg. Cent. (392 deg. Fahr.) after cementing appears very advisable for all pieces subject to blows. The author concludes by giving further applications of cementing. He reminds the readers that iron can absorb aluminum, that copper and brass absorb tin, by heating these metals just below their smelting point. He adds data upon the negative results he has obtained in his attempts to get very soft steel to absorb nickel. There is, however, still much to be done along these lines. (From advance sheets.)

A paper on "cementing in a vacuum" was presented at the same Congress by F. Weyl of Aachen. The author carried out a large number of experiments in order to ascertain whether elementary carbon had a cementing action in a vacuum. He took carefully purified natural graphite, also purified kish, separated from pig iron, sugar carbon and diamond. For the experiments he used a furnace in which a vacuum down to a few thousandths of a millimeter could be obtained. The small quantities of gas which developed in the furnace during the tests were immediately removed by the constant action of the air pump. The tests were carried out with the purest electro-ingot-iron obtainable, which, owing to complete freedom from slag, afforded the certainty that no further reactions intervened during heating. In two of the tests the arrangement was such that the smallest amount of gas given out by the cementing medium had to flow past the iron, so that it was possible to ascertain whether the attenuated gases had a carburizing effect or not. The experiments have afforded the proof that under the conditions followed elementary carbon does carburize. It was further ascertained that carburizing only takes place when the cementing agent is in close contact with the iron at a sufficiently high temperature. The experiments with powdered diamond are specially instructive in this connection. In those in which the iron specimens were laid upon the powdered diamond almost no cementing was found to take place at a temperature of 1000 deg. Cent. (1832 deg. Fahr.). This is explained by the fact that the iron specimens were supported solely by a few diamond points, contact with the cementing medium being thus a very imperfect one. When the powdered diamond was strewn upon the iron a heating during 30 hours to a temperature of 975 deg. Cent. (1787 deg. Fahr.) sufficed to produce distinct carburizing. In the latter case the surface of the diamond particles was in contact with the iron, contact between the cementing medium and the iron thus being much more complete. (From advance sheets.)

Gold and Silver.

Cyanide Practice at the Dolores Mines in Mexico is described in the *Bulletin* of the Colorado School of Mines (Vol. 5,

No. 3) by Mr. W. H. PAUL. The principal details of the practice are as follows: Twenty-five stamps crush the ore through 8-mesh battery screen, the product flowing to a 20-mesh King revolving screen. The King screen oversize is reground in Bryan mills fitted with 70-mesh screen. The King screen undersize and the Bryan mill product are united and prepared for table treatment by cone classifiers and hydraulic sizers. The table concentrate is dried and sacked and the tailing passes to Baylis classifiers where separation of sand and slime is effected. The slime passes to the cyanide plant and the sand is ground in tube mills. The latter product is then treated on tables and passes through the classifying system again.

The cyanide plant comprises seven tanks, each 22 ft. in diameter and 15 ft. deep, equipped with mechanical agitators and centrifugal pumps. Two tanks are used for filling, two for agitating and two for settling and decanting; the seventh tank is used for storage for the Butters filter. Two tanks are filled at the same time, requiring about 24 hours, and holding about 150 tons. After filling, the pulp is allowed to settle for about four hours and the mill solution is decanted. A cyanide solution is now added to the pulp, the strength being 0.04 to 0.05 per cent KCN, and the mixture agitated for 24 hours, after which it is discharged to another tank and allowed to settle, the solution being decanted as far as possible. Finally, the pulp is transferred to the storage tank for the Butters filter. The latter has 60 frames 10 ft. x 5 ft. The filter cake is formed in the usual manner and subjected to a wash with weak cyanide solution for an hour, after which a clear water wash is applied. The filtered solution containing the gold and silver is clarified and precipitated in zinc boxes.

The treatment of the zinc box precipitate at this plant is interesting, inasmuch as acid treatment is dispensed with. In the clean-up of the zinc boxes the precipitate flows into a tank equipped with a mechanical agitator; from this tank the precipitate is pumped into a Johnson press. Strong cyanide solution is run into the tank during agitation and also through the press, the overflow from the latter running into a row of zinc boxes so that nothing can be lost. In this way a high-grade bullion is obtained without resorting to acid treatment. The shipped bullion averages 960 fine in gold and silver. The short zinc is treated in an amalgamation pan into which strong solution flows, the outflow passing back to the zinc boxes. The average extraction on Dolores ore for several years past has been a little over 90 per cent of the gold and silver, the former showing higher extraction percentage than the latter. Experiments have been made on cyaniding the concentrate, and extractions of 96 to 98 per cent have been obtained by grinding in pans for 72 hours with very strong cyanide solution.

Crushing Machines for Cyanide Plants.—The recent growth of a sentiment among cyanide metallurgists against the use of gravity stamps for crushing preliminary to cyaniding dates from the perfection of the ribbed tube mill liner and of the tall, air-agitation tank. Mr. MARK R. LAMB reviews the steps in this change of opinion, and outlines several combinations of machines that can be used in crushing ore for cyanidation, in the July, 1910, *Bulletin* of the American Institute of Mining Engineers.

The beginning of the surrender of the supremacy of the gravity stamp was in the adoption of tube mills for fine grinding and the use of coarse battery screen. Rolls are usually used where a comparatively small tonnage is to be prepared for coarse concentration, but where no concentration is to be provided for, the combination of rolls and tube mills, without the complication of trommels and elevators, is gaining favor. The next step was the use of the Chilean mill between the stamps and tube mill, which again narrowed the field of the stamp. This combination offers advantages when fine concentration is necessary, but even concentration is not favored in all quarters, and in one instance the use of the concentration table is merely to separate the coarse product which requires further tube milling.

The next step was the abandonment of amalgamation of every product except the concentrate by the Goldfield Consolidated Company in treating a high-grade, free-milling gold ore. This places the stamps at that mill in the class of coarse crushers, and Mr. Lamb believes that the work could be done more economically by rolls if the plant were redesigned.

In considering the steam stamp, Mr. Lamb states that manufacturers are considering a steam stamp that will be intermediate between the Tremain stamp, having a capacity of 20 to 30 tons, and the stamp of Lake Superior district, which crushes 500 to 800 tons.

The success of the pebble mill in cyaniding offers encouragement to adopt another machine from the cement industry, viz.: the ball tube mill. The disadvantages of the ordinary ball mill, with its expensive liners and screens, are entirely avoided in the ball tube mill. Data are quoted from M. W. Von Bernewitz, showing that 120 tons crushed in a No. 5 Gates crusher are handled in three No. 5 ball mills, crushing to 27 mesh. This dry grinding costs 42 cents per ton, and the wear is 3.5 oz. of steel per ton of ore ground. The power required for each ball mill is 16 hp to 20 hp.

Thus there are three machines that can replace gravity stamps where neither concentration nor amalgamation are to be practised. A simple plant would consist of breaker, ball tube mill, classifier and pebble tube mill. Rolls could be substituted for the ball tube mill, and if power is cheap the plant could consist of breaker, steam stamp, classifier and pebble tube mill.

In conclusion, the author finds that: Large tonnages of hard ore are crushed advantageously to 3/16 in., or coarser, by steam stamps; small tonnages of hard rock are crushed advantageously with rolls, which are cheaper than gravity stamps if they are not expected to crush too fine, and are not complicated with screens and elevators, as they would not be for feeding tube mills. Large ball tube mills are available for crushing 3-in. ore to a size suitable for pebble tube mill feed in one operation with no screening. Finally, small steam stamps will probably be at the disposal of metallurgists in a short time, and while we in America have been wondering at the tendency of South African metallurgists to use heavier stamps than we could be persuaded to use, perhaps, after all, we have been fortunately slow.

Lead.

The heat of formation of slags in lead smelting has been investigated by Messrs. H. O. HOFMAN and C. Y. WEN, who have published their results in the July, 1910, *Bulletin* of the American Institute of Mining Engineers. The investigation is interesting inasmuch as heretofore assumptions have had to be made for the heat of formation of the slag in casting a thermal balance of the heat generated and absorbed in a blast furnace treating non-ferrous ores.

The materials used in the experiments were silica, prepared from pure quartz crystals; ferrous oxide, prepared from pure iron and pure ferric oxide; and calcium carbonate, being the precipitated salt sold as chemically pure. The usual method of determining the heat of formation of slag is to mix the slag-forming elements with sufficient finely divided charcoal to raise the temperature of the charge well above the formation temperature of the slag when the charcoal is burned in oxygen in a calorific bomb. The difference between the predetermined calorific power of the charcoal and that of the charge gives the quantity of heat generated or absorbed in the formation of the slag. This method was not used in the present investigation, however, owing to the possibility of reducing some of the ferrous oxide by the charcoal. A satisfactory method was found in fusing the charge in a platinum boat by electricity, the boat to be contained in a bomb and the latter submerged in a calorimeter. The difference between the quantity of heat generated by the electric energy alone and the quantity generated when fusing the charge gives the heat of formation of the mixture.

In every experiment the determination was preceded by duplicate blank tests. The conditions of each experiment were

uniform as to quantity of water (1200 gal.) and time of passing the current (1 min.).

Ferrous Singulo-Silicate (FeO, 70.8; SiO_2, 29.2 per cent.—The accepted values obtained were 111, 103 and 115 g-cal. evolved in the formation of 1 gal. of ferrous singulo-silicate; average, 109 g-cal.

Ferro-Calcic Singulo-Silicate (FeO, 57.58; CaO, 12.00; SiO_2, 30.42 per cent.—Calcium carbonate was used instead of calcium oxide, and the heat necessary to split $CaCO_3$ into CaO and CO_2 (96 g-cal.) was considered in the results. The average heat evolved in the formation of this slag was 140 g-cal.

Ferro-Calcic Singulo-Silicate (FeO, 40.3; CaO, 28.0; SiO_2, 31.7 per cent.)—The heat of formation of this slag was found to average 193 g-cal.

The amount of heat evolved is seen to increase with the greater percentage of lime. Experiments were conducted with varying percentages of lime between 0 and 12 per cent CaO, and between 12 and 28 per cent CaO, but many failures were encountered and the gaps were not filled.

Fine grinding tests to determine the relative value of the grinding pan and tube mill in regrinding jig tailing for further concentration are reported in the *Transactions* of the Australasian Institute of Mining Engineers, Vol. XIII (1909), by Messrs. W. E. Wainwright and W. J. McBride. The ore at the Broken Hill South mine assays about 15 per cent lead, 12 per cent zinc and 5 oz. silver. It is a hard ore containing quartz, rhodonite, garnet and feldspar in the gangue. The galena is very finely disseminated throughout the gangue, rendering fine grinding imperative for its liberation.

The system of concentration in use involves jigging, and it is with the jig tailing that the tests are concerned. At the time a grinding pan was introduced Krupp ball mills were used, but the power and maintenance charges were very high. Preliminary work with the pan showed such good results that the Krupp mills were taken out, and grinding pans installed throughout the mill. Later a Krupp tube mill was installed and run in parallel with the pans so that comparative tests could be made. The tube mill was first run with a lining of special toughened steel plates, but these soon wore through and were replaced with 1¼ in. hard cast-iron plates which gave excellent satisfaction.

In the test runs one section of the mill was used comprising rolls, trommels, jig, grinding pans or tube mill, Wilfley tables and Luhrig vanners. The jig concentrate removed 47.9 per cent of the lead in the crude ore, leaving 52.1 per cent of the lead to be dealt with. Detailed tables are given which show that the pan-ground product was slightly better suited for table concentration than the tube mill product, but the difference was very slight and might with equal justice be referred to inherent differences in the quality of the ore and conditions of treatment. But on investigation of the two machines as crushers, without regard to subsequent concentration, the units of efficiency were found to be in the ratio of 2:1 in favor of the tube mill. Plain liners for the tube mill give very poor crushing efficiency, but the use of bars and that of hard rhodonite lumps instead of small flint pebbles increases capacity.

Concentration.—The instances in which a large mining company has opportunity to plan ore dressing works based on a wide past experience with its ore are so few that the published data on such work are very valuable. In the *Mining Magazine* (London) for May, June and July, 1910, Gelasio Caetani presents some considerations of the theory and practice of concentrating lead-silver ore at the new mill of the Bunker Hill & Sullivan Mining & Concentrating Company. Experiments were conducted over a period of three years to ascertain the best method of treatment for the new mill.

The chief aim was to design a mill of great mechanical simplicity and high economical efficiency. The mill is divided into three departments mechanically independent, viz., the ore house for the preparation of mill ore, the section for coarse concentration, and that for table concentration. The ore progresses from one department to the other, undergoing at each step a complete treatment, so that none of the products is ever returned through the channels or over the machines through which it has already passed. This prevents idle circulation of products and makes the mill sensitive to changes in any part. This sensitiveness is taken as an advantage, for no defect in operation can remain long undetected. All of the products of one machine are usually retreated on other machines, the function of which is to rectify the work of the first.

The ore is prepared for milling by crushing and screening in 30-mm trommels. The undersize of the trommels is ready for treatment and is stored in bins of 1900 tons capacity. As the ore is delivered to the mill it passes under a 12-in., 500-volt, ½-amp Browning electromagnet which removes nails, bolts, nuts, etc., which would cause trouble in the mill machinery.

In the mill the ore is screened through 10-mm trommels, which split the ore into two products, 30 mm to 10 mm, and 10 mm to zero. These products are treated in coarse and fine jigs respectively. The jigs split the feed into 14 products, two of which are concentrates ready for shipment, two are tailings, and the other 10 are intermediate products which undergo further reduction.

The feed to the table concentrating department comes from the jigs and consists of jig slime and classifying-jig middling. All pulp passed onto the tables has been through Bunker Hill screens of 22 mesh. Card tables are used. Fine concentration is effected on Frue vanners which treat a thickened pulp of the fine material.

In the coarse concentration department a wider range of sizes is used than is customary in the district. To determine these sizes a 100-ton experimental plant was erected and operated, which showed that with specially constructed Harz jigs it was possible to handle all ore by dividing it into two products, a 30-mm to 10-mm and a 10-mm to zero feed. Both bull and classifying jigs are of the same design except that the latter has a slime separating device in the first compartment, and the bull jigs have one less compartment than the classifying. The gate and dam discharges are all placed at the center of the tailboard and give perfect satisfaction.

The bull jigs separate the following products: 1st, cup concentrate, 52 to 59 per cent lead; 2d, cup concentrate, 40 per cent lead; 3d and 4th, cup middling, 8 per cent lead; tailing, 1 to 1.4 per cent lead, and hitch discharge, 25 per cent lead.

The classifying jigs make the following products: 1st, cup concentrate, 65 to 73 per cent lead; 2d, cup concentrate, 40 per cent lead; 3d and 4th, cup middling, 10 per cent lead; 5th, cup middling, 5 per cent lead; slime discharge from 1st compartment, 14 per cent lead; 1st hutch discharge, 55 per cent lead; 2d hutch discharge, 30 per cent lead; 3d, 4th and 5th hutch discharges, 7 per cent lead; tailing, 1.4 to 1.8 per cent lead.

An interesting device used in connection with the table work is the pipe classifier which concentrates the first and second hutch discharge of the classifying jig, making a 60 to 77 per cent lead concentrate and sending an overflow of lower lead content to the tables. In the table work of the mill the following fundamental principles have been followed. (1) In grinding, the free galena will concentrate almost exclusively in the fine; therefore, the oversize can be screened out and sent to the fine grinding machines without first undergoing a process of concentration. (2) Not to require from any single machine a thoroughly clean work, but to have each operation of screening, classifying or concentrating repeated by other machines on each single product of the first machine.

The screens and classifiers used in the mill are of local design. In the classifier very little hydraulic water is used, and it will not choke or get out of order, even when not looked after for weeks at a time. Certain theoretical considerations and experimental results given by the author lead him to believe that hydraulic classification is preferable to screening, although a perfect classification can never be obtained in practice. Working with a perfectly classified pulp, a table would act as a perfect sizing machine.

Roasting.

Recent Progress in Blast Roasting is detailed rather fully in the June *Bulletin* of the American Institute of Mining Engineers by Prof. H. O. HOFMAN. Both up-draft and down-draft apparatus is considered and the various installations of each described, with details as to actual practice.

Of the up-draft apparatus the Huntington-Heberlein is the best known and most used. At Trail, B. C., this process is used on galena concentrates, the charge being made up to work well in the reverberatory, the blast roast and the blast furnace. It is compounded to contain, Pb, 40 to 44; Fe, 10 to 13; SiO_2, 8 to 11; CaO, 7 to 10; zinc, less than 10 per cent. An interesting development of blast roasting at this plant is the roasting of lead blast furnace matte, containing Pb up to 25 per cent, and Cu from 8 to 10 per cent.

At the different works of the American Smelting & Refining Co., where the H-H process is used, variations in practice have to be introduced to meet the conditions. As a rule a considerable amount of experimenting has to be undertaken before the process is worked satisfactorily. The process is of great aid to smelting, however, as is shown by the results obtained at the Murray plant of this company, where a blast furnace formerly treating 160 tons in 24 hours, now treats 200 tons with more ore on the charge than formerly. The blast roaster charge is made up of roasted and raw ore in proportions to keep the lead between 18 and 20, the zinc under 10 and the sulphur between 16 and 20 per cent.

The Savelsberg process is in use at the works of the St. Joseph Lead Co., Flat River, Mo. Eighteen pots, in two rows, are used for the roasting and agglomeration of non-argentiferous galena concentrate. The mixture roasted has the following composition; SiO_2, 13.12; FeO, 5.90; CaO, 6.60; MgO, 3.30; Pb, 47.40; Zn, 2.10; S, 11.20; H_2O, 6 per cent. In the finished product the Pb is 44.2 and the S, 2.30 per cent.

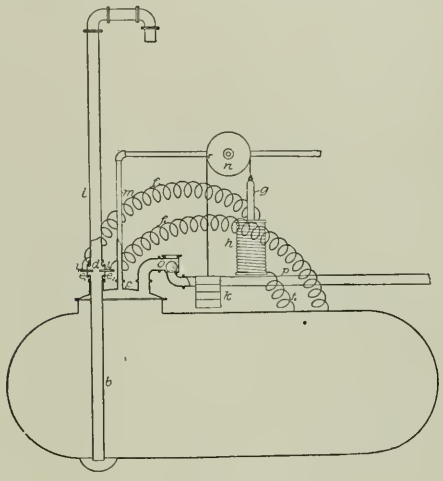
At the Midvale plant of the U. S. Smelting, Refining & Mining Co., blast roasting is carried out in 20 so-called roasting boxes. Each box has a hearth 6 ft. square and 4 ft. deep. When the charge is sintered it is rammed out by an electrically driven ram similar to that used in discharging coke ovens. The charge consists of 33 parts concentrates, 5 to 10 of flue dust and 62 to 57 of fine ore. A priming mixture is used on which the charge is dumped. One of the boxes is kept in commission on the priming mixture.

On copper and copper-nickel ores the blast roast has not been very successful. This has been tried at the Garfield plant of the A. S. & R. Co., and at the Detroit Copper Co., Morenci, Ariz. Better success was had at the Copper Cliff works of the Canadian Copper Co., but the process is not now in use owing to the lack of mechanical appliances.

Three different types of the down-draft apparatus are described in this article, viz., the drum, the straight line and the horizontal table. These embody the principle patented by Messrs. Dwight and Lloyd. The time of treatment in these continuous machines is about 1 minute for each per cent. of sulphur. The drum machine is in operation at Maurer, N. J., and at Baltimore, Md. The feed and suction box are at the top of the drum, and the discharge is effected by an upturned grizzly which scrapes the sinter from the hearth. The straight line machine is in operation at Salida, Col., on a charge carrying about 35 per cent silica and 17.7 per cent sulphur. This charge is made up of lead ore, sulphide concentrate, flue dust and silicious and oxide ore. The sinter carries about 4 per cent sulphur. The machine treats about 50 tons in 24 hours, at a cost of $.75 per ton. In Illinois the cost of similar treatment is $.50 per ton. The horizontal table is in use at Garfield, Utah. The sulphur on the charge is about 30 per cent and the sinter carries about 6 per cent. The annular grate has an area of 126 sq. ft., of which about 50 per cent is effective. The feed hopper and igniter are stationary and the sinter is removed by a scraper and deflecting apron.

Chemical Engineering.

Automatic Acid Egg.—Vol. II of the *Transactions* of the Amer. Inst. Chem. Eng., just published, contains a paper by Mr. RICHARD K. MEADE on an automatic acid egg. The ordinary cast-iron acid egg has proved a most satisfactory method of elevating acids, but its great objection is the constant attention which it requires. On the other hand, it does not get out of order and may always be relied upon to do the work. The author has devised a simple attachment which may be placed upon the ordinary cast-iron egg, and which is designed to operate the valves, thus doing away with an attendant and so making the egg automatic in its action. It is shown in the accompanying illustration. The acid enters the egg by means of the pipe *p* and leaves by means of the pipe *b*. Air is conducted in by means of the pipe *m*. To the pipe *b* is bolted an



AUTOMATIC ACID EGG.

ordinary cast-iron cross *d* and to the side openings of this are fastened two porcelain plates e_1 and e_2. Into these pipes are cemented two ordinary electric light carbons and connected respectively by wires f_1 and f_2 with the electromagnet *h*, this circuit containing a source of current.

The working of the egg is as follows: The acid flows into it through the pipe *p* and ball valve *o* until the egg is full and it rises to a point above the carbon pencils. As soon as it does this, electrical connection is established and the electromagnet *h* pulls down a core *g* of soft iron. This latter opens a valve *n* and allows the air to enter the egg. The check valve *o* closes, of course, as soon as air enters the egg and causes back pressure in the pipe *p*, and the acid is forced up the pipe *b*. As long as any acid is passing through this pipe, the electrical circuit is, of course, closed and the magnet excited and the valve *n* is held open. As soon as the current stops the connection is broken and the valve is closed by means of the weight *k*. The valve *n* is made by simply taking off the wheel from an ordinary glove valve and replacing it by a wooden disk with a groove in the edge in which the cord works. The cord is fastened to the disk at one point *r* only. In place of the magnet, the connection may be made to start a motor-driven air compressor. The electrical current used for operating the egg may be either from the electric lighting current or from batteries, and the magnet may be located at any convenient point where its workings may be observed. A counter may also be attached to the valve, and if the volume of the egg is known the liquid is so measured. The author has used this device, the feature of which is the simple electrical attachment, for the lifting of sulphuric acid in practice.

Recent Metallurgical and Chemical Patents

Iron and Steel.

Electric Steel Refining.—In view of Mr. Wm. R. Walker's position as the technical head of the United States Steel Corporation, his latest patent on electric steel refining is of particular interest. In this a prominent place is given to the mixer in a Bessemer converter-electric furnace combination process. The pig iron is treated in an acid-lined Bessemer converter or open-hearth so as to desiliconize it and in part decarburize it; from 0.04 per cent to 0.3 per cent of carbon are left in the metal to prevent oxidation of the metal and maintain it fluid. The metal is then run into a suitably heated basic-lined mixer. The metal in its passage to the mixer passes through a spout or runner, which is provided with a skimmer, such as used in the casting of blast furnaces, and by this skimmer, the floating slag is removed from the metal. The metal is thus delivered to the mixer in as nearly slagless condition as possible, since otherwise the acid slag derived from the furnace or converter would prove destructive to the basic lining of the mixer. In this mixer a basic slag is maintained for the purpose of aiding the dephosphorizing of the metal or to neutralize the acid slag which may be mixed with the metal. From the mixer charges are withdrawn from time to time and introduced into basic-lined electric furnaces, where the metal is treated with additions of oxide of iron, together with burnt lime or limestone. In the electric furnace the metal is dephosphorized and desulphurized. This process not only permits to improve the product of the Bessemer converter in such manner as to obtain steel of the quality produced from the open-hearth furnace or by the crucible process, but it also permits to employ for the Bessemer converter iron containing more phosphorus than is permissible in so-called Bessemer iron. The use of the mixer between converters and electric furnaces permits to deliver the metal to the latter in a uniform condition, and to deliver it as required. The application for this patent was filed April 21, 1906. (967,590, Aug. 16, 1910.)

Nickel-Copper Steel Direct from Ore.—In connection with the article of C. F. Burgess and J. Aston in our last issue on alloys of monel metal and iron and the article of G. H. Clamer in our present issue on the same subject, a patent of G. H. Clamer, of Philadelphia, is interesting, which proposes to produce nickel-copper-iron alloys directly from the Sudbury sulphide ores. The alloy can be made within the following range of proportions by weight: Nickel, 25 per cent to 50 per cent; copper, 5 per cent to 20 per cent; iron, 30 per cent to 70 per cent. An alloy containing 65 per cent iron, 25 per cent nickel, 10 per cent copper and 0.2 per cent carbon gave the following test results: On a test bar, ½ in. in diameter and 2 in. between shoulders, strain per square inch in pounds, 96,100; limit of elasticity per square inch in pounds, 51,750; elongation per cent of original length, 42 per cent; reduction of area per cent of original section, 53.7 To make the alloy directly from ore containing nickel, copper and iron in the form of sulphides, the ore is first melted with fluxes to form a slag with the gangue and the production at the same time of an iron-copper-nickel matte. This matte may be of varying composition and, if the iron in the matte is higher than desirable, the matte can be bessemerized, so that approximately the desired amount of iron is removed by the resulting oxidation, and the amount of iron desired for the finished alloy remains. This matte is then calcined to remove sulphur and convert the nickel-copper and iron into oxides. These oxides are reduced with carbon, whereby there is produced a homogeneous alloy of the three metals in the same relative proportions in which they existed in the matte before calcining, the carbon should not be present in amounts producing hard metallic nodules. (965,871, Aug. 2, 1910).

Electric Steel Furnace.—The electric furnace of Hans Nathusius, of Friedenshütte, near Morgenroth, Germany, is interesting, since it is known that electric steel refining has been carried out with it at Friedenshütte for over a year. The construction for operation with three-phase currents is shown in Fig. 1. The furnace may be said to combine the Héroult and Girod principles. *abc* are three upper electrodes, as in the Héroult furnace. *def* are water-cooled steel electrodes, embedded in the hearth. These three bottom electrodes are connected to the neutral wire *t* of the three-phase system, while each of the three upper electrodes is connected to one of the three phases. If the switches 5, 6, 7, 8 are open current is

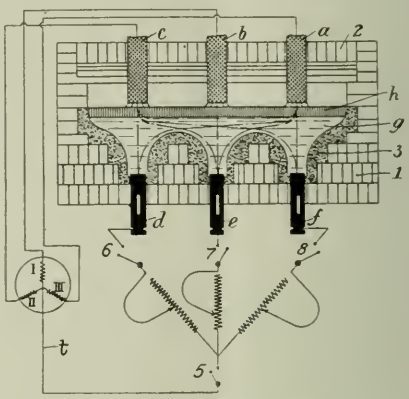


FIG. 1.—STEEL REFINING FURNACE.

supplied to the upper electrodes only and the furnace operates like a Héroult furnace. When 5, 6, 7, 8 are closed the current will also pass from the lower to the upper electrodes as in the Girod furnace. The latter current may be varied by means of the adjustable resistances shown. (958,757, May 24, 1910.)

Combined Induction and Resistance Furnace.—J. H. Reid, of Newark, N. J., patents an electric furnace in which the resistance heating of the charge by means of a current sent into the charge through electrodes or terminals is supplemented by "induction heating." For this purpose an induction coil is provided which may be suspended more or less into the charge to produce greater or smaller heating effect. It contains a central, annular, current-conducting coil, surrounded by an annulus of soft laminated iron rings, which are in turn surrounded by a cover of refractory material. (956,544, May 3, 1910.)

Calcium Silicide.—If calcium silicide is produced from lime, silica and carbon in an electric furnace according to the reaction $CaO + 2\ SiO_2 + 5\ C = Ca\ Si_2 + 5\ CO$, the difficulty is experienced that the silica and lime have a tendency to unite directly and form silicates of lime. To prevent this, Georges Strauss, of Paris, France, employs calcium carbide instead of lime and metallic silicon instead of silica, the reactions being as follows:

$$CaC_2 + 2\ SiO_2 + 2\ C = CaSi_2 + 4\ CO.$$
$$2\ Si + CaO + C = CaSi_2 + CO.$$

"Experiments made with calcium silicide have shown that this metallic silicide is capable of replacing with advantage the aluminium usually employed in steel works for the purpose of preventing the formation of blow holes in steel ingots. The advantages ensuing from such application are, in addition to the economy secured by the very low price of calcium silicide, very great rapidity of reaction and the suppression of the undesirable

'shortness' which aluminium communicates to ingots of steel." (964,459, July 12, 1910. Assigned to Soc. An. Co. Generale d'Electrochemie de Bozel.)

Blast Furnace Charging.—In charging blast furnaces, the ore, coke, etc., is drawn up by a skip and discharged usually from one side into a hopper. By reason of this side discharge the material is unevenly distributed on the bell of the main hopper and consequently unevenly distributed in the furnace. Mr. WALTER KENNEDY, of Pittsburgh, interposes an intermediate receptacle between the receiving and main hopper, provided with a movable closure in combination with means for rotating the receptacle any desired distance so that the material in the receptacle may be discharged at any desired point upon the bell of the main hopper. (955,885, April 26, 1910.)

Briquetting Iron Ore.—W. J. SHAW, of Toronto, Ont., Canada, mixes 100 lb. of magnetic iron oxide sand or other pulverized iron ore with 10 lb. or 20 lb. powdered charcoal, sufficient for combination with the oxygen of the ore, 6 lb. to 10 lb. of silica sand, and 10 to 20 lb. carbonate of sodium. These are mixed dry and then saturated with an adhesive vegetable paste, like a solution of potato starch, in hot water. Such a composition when mixed together in about an hour forms a solid briquette which incloses the particles of the pulverized ore within a pure carbon protecting it from the oxygen of the blast and from phosphorous fumes when fuel containing these impurities is used. (963,400, July 5, 1910; assigned to the International Tool Steel Company.)

Gold and Silver.

The Adair-Usher process of lixiviating sand with cyanide solution was developed in South Africa and is known to most metallurgists. Briefly, it consists of circulating the solution by introducing it into a central chamber in the bottom of the vat, whence it travels through radiating pipes and issues through apertures in the pipes and thus rises through the mass of sand. Mr. C. E. D. USHER, of Johannesburg, Transvaal, has recently received letters patent on the distributing pipe which is used in the process. It comprises an outer pipe which is connected with the chamber at the center of the vat and extends through the wall of the vat. This pipe is perforated and is placed preferably with the perforations on the under side. Within this outer pipe is another closely fitting pipe which may be turned by means of a handle at the end outside of the vat. Both ends of the inner pipe are plugged, but it has a large aperture in its side near the end which extends into the central chamber, and also has other apertures along its length to correspond with the apertures in the outer pipe. When the vat is filled with the pulp to be treated, the solution is pumped through the central chamber into the pipes radiating from it, the openings in the latter being adjusted by turning the inner pipe within the outer. When the lixiviating process is completed the openings may be closed and thus the pulp is prevented from clogging the pipes and preventing the even distribution of solution to every part of the vat. (965,767, July 26, 1910.)

Slime Filter.—A novel idea in centrifugal slime filters has been patented by H. V. HOLMAN, of Colorado Springs. Col. It consists of a rotor, which is made of heavy wire screen covered with canvas or other filtering medium, into which the pulp to be filtered is conducted. The rapid revolution of the rotor produces filtration by centrifugal action. The accumulated sand on the filter bed is continuously loosened and removed by a plurality of disks which are supported within the rotor, and which may be moved to or from the filtering surface in order to remove more or less of the accumulation. The disks are not fixed in their position and therefore do not offer great resistance to the rotation of the screen and are not subject to excessive wear. The inventor claims superiority over fixed screw types of excavators in centrifugal filters in that the excavation is more complete, and, therefore, the capacity of the filter greater; the disks are not likely to become stuck if the machine is fed too heavily or run too fast; varying thicknesses of filter bed can be formed owing to the adjustment of the position of the disks. (966,918, Aug. 9, 1910.)

Roasting.

Multiple-Hearth Roaster.—Improvements in the type of roaster known as the McDougall, are proposed by W. M. BARKER, of McGill, Nevada, who has assigned his interest to the Allis-Chalmers Co., of Milwaukee, Wis. It is proposed to cool the rabble arms by means of air introduced at the bottom of the hollow vertical shaft, which carries the rabble arms on each hearth of the furnace. At the ends of the rabble arms are holes for the exit of the heated air into the furnace. An improvement in the rabbles consists in the use of a rabble holder suspended from the rabble arms by flanges. Each rabble holder has two wedge-shaped slots passing through it, the slots intersecting near their centers. The rabbles have dovetail shaped ends, which fit into either of these slots depending on the direction of rotation of the rabble arms on which they are placed. The pitch or inclination of the rabbles in the lower hearths is greater than that of the rabble in the upper hearth; and the rabbles of the upper hearth are wider than the rabbles of the lower. This arrangement will tend to more readily break up any wet or sticky ore fed onto the upper hearth. The advantage of having the slotted holders is that any holder can be used on any hearth and can be made either right or left handed by changing the rabble from one slot to the other. (961,081, June 7, 1910.)

Tin.

Detinning Tin Scrap.—For detinning purposes C. J. REED proposes to make use of the fact that metallic tin is rendered weak, friable and easily pulverized by heating to about the melting point of tin. Tin scrap is mixed with an equal volume of finely divided quartz, sand, etc., and the mixture charged into a rumbler, wherein it is subjected to heat and agitation until the tin is entirely removed from the iron. The product is then thrown upon a screen which separates the pieces of detinned iron from the powdered tin and sand. The tin and sand powder mixture is placed into a closed vessel into which dry chlorine gas is passed. Gaseous stannic chloride passes out and is recovered in a condenser. (958,177, May 17, 1910.)

Detinning.—E. C. HIGGINS, of Bayonne, N. J., patents mechanical details of an apparatus in which tin scrap is detinned by placing it in a heating basket which is rapidly revolved. The tin melts and is thrown out of the basket through wire mesh by centrifugal force. (965,580, July 26, 1910.)

Aluminium.

Aluminium from Sulphate.—A new aluminium process is the object of a patent of H. F. D. SCHWAHN, of Belleville, Ill. He first produces by an older process aluminium sulphate, which he mixes with pitch, the mixture being heated in a closed container to 900° C. This results in the formation of $Al_2(SO_4)_3.C_{15}$, which is an "electric resistance material or electrolyte" of coke-like appearance. This is treated in an electric furnace with a flux of fluorides of the alkalies, or alkaline earth metals, in a reducing atmosphere of producer gas, with a "gaseous flux," as fluorine in form of hydrofluoric acid vapors. The result is stated to be aluminium free from carbon or aluminium carbide. (964,566, July 19, 1910.)

Alloys.

Magnesium Alloys.—G. PISTOR and P. RAKOWICS of the Chemische Fabrik Griesheim Elektron, at Frankfort-on-the-Main, Germany, proposes the use of high-percentage magnesium alloys, containing from 80 per cent to 99.5 per cent magnesium and from 0.5 per cent to 20 per cent of other metals, for any mechanical purposes, especially automobiles, airships, instruments and for military and sporting purposes. These alloys can be readily machined, soldered, welded, forged and cast; they have a high tenacity and elasticity and low specific gravity. An alloy of 92 per cent magnesium and 8 per cent aluminium has a specific gravity of only 1.75 and "strength and

extensibility equal to the best gun metal." Alloys of magnesium and zinc or of magnesium and copper are "greatly superior in their physical properties to the usual aluminium or its alloys." The well-known alloy "magnalium," which is an alloy of aluminium and magnesium, containing at most 25 per cent magnesium, is stated to be very different from these high-percentage alloys. (965,485, July 26, 1910.)

Electric Furnaces.

Calcium Carbide and Cyanamide.—A process has been devised by E. H. Meyer and J. M. A. Stillesen, of Niagara Falls, Ontario, Canada, for the object of producing calcium carbide from the raw materials in a single operation and utilizing the gaseous products from the reduction of coal and limestone to preheat the charge and to obtain (from the air used thereby) nitrogen for the manufacture of calcium cyanamide. The arrangement of the plant is shown in Fig. 2. The finely crushed raw materials, limestone and coal, are fed through 4 into the upper half of the stack 1. This upper half forms a retort for conversion of the limestone into lime and of the coal into carbon. The reduced lime and carbon is then dropped down into the receiver 3, from which it is fed to the electric furnaces 5 by means of the pumps 6. These furnaces are arranged around the bottom of the receiver; only one of the furnaces 5 is shown in the illustration. It is inclined to the vertical, as shown, for the purpose of getting better efficiency and to enable the charge to be supplied at the bottom of the furnace at the hottest part. 8 is a movable electrode (controlled by the motor 10) and 9 a fixed bowl-shaped electrode. The reaction in the electric furnace is, as is well known, $CaO + 3C = CaC_2 + CO$. The carbon monoxide evolved in this reaction is utilized as follows:

From the furnace 5 a gas vent pipe 12 connects with rings 13, from which the gas is supplied to the conical burner 14 in the retort 2, after being mixed with air, and burned therein to reduce the coal and limestone. A sufficient supply of air is admitted with the gas to burn all the carbon monoxide and hydrogen in the retort 2 to carbon dioxide and water, so that

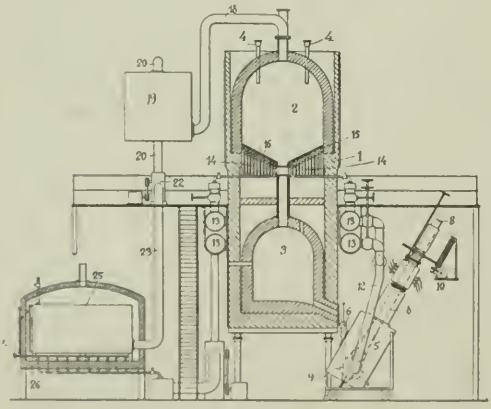


FIG. 2.—CARBIDE AND CYANAMIDE PLANT

the gases passing out at the top through pipe 18 will consist principally of carbon dioxide, water and nitrogen. A valve controls the admission of air. Pipe 18 leads through a cooler (not shown), thence to a tank 19 containing calcium hydrate, which absorbs the water and carbon dioxide, but not the nitrogen which was introduced as a part of the air. From tank 19 the nitrogen passes by pipe 20 through a fan 22 to a dryer not shown, thence through a pipe 23 to the cyanamide furnace 25. This latter furnace consists of an air-tight drum on a foundation 26, and is heated by carbon monoxide gas from the carbide furnace, if desired, ignited by acetylene or other means.

In carrying out the process the coal and limestone charged into retort 2 is admitted to reservoir 3 and thence fed, while still hot, into the furnace 5, and thence the melted carbide, when formed, is tapped out from the bottom of the furnace 5 and is ready for use after being cooled and crushed, in making calcium cyanamide, or for other purposes. Since the charge is hot when introduced into the carbide furnace and is fed at the bottom at the hottest zone, the electrical energy required is relatively small. (965,393, July 26, 1910.)

Carborundum.—A continuous process for the manufacture of carborundum has been devised by Frank J. Tone, of the Carborundum Company, of Niagara Falls, N. Y. A ver-

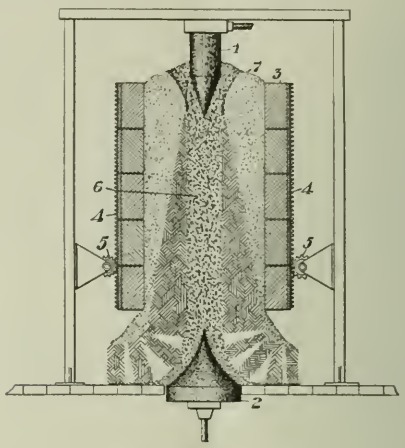


FIG. 3.—CARBORUNDUM FURNACE.

tical furnace for this purpose is shown in Fig. 3. The side walls are formed of transversely divided refractory sections 3 secured to rack sections 4, which engage with pinions 5. A continuous downward movement of the walls with the charge inside is then produced. In the center of the charge is the resistor consisting of granular core material which is supplied at and around the upper electrode. This granular resistor core is surrounded by the charge mixture of carbon and siliceous material 7, which is also supplied at the top. The lower electrode is pointed or conical so that the core material and finished product are deflected outward so as to be easily removed. A horizontal furnace embodying the same fundamental idea is also described. (965,142, July 19, 1910.)

Metallic Electrode.—In the production of metals or alloys which are intended to be free from carbon, Fred. M. Becket, of the Electro-Metallurgical Company, of Niagara Falls, employs metallic instead of carbon electrodes. A suitable form is a hollow block of cast iron with connections for the circulation of cooling water. Wrought iron may be employed instead of cast iron. Where particularly high temperatures are used, electrodes of a metal or alloy of high melting point, like ferrotitanium, may be used; in this case the cooling medium may sometimes be dispensed with. (967,150, Aug. 16, 1910.)

Electrolytic Processes.

Bicarbonate of Soda.—A modification of the Solvay process is patented by R. H. F. Finlay, of Belfast, Ireland. He follows the earlier steps of the well-known ammonia soda process up to the point at which the mother liquor has to be dealt with. That is to say, he treats the brine with ammonia and carbonic anhydride so as to obtain a precipitate of bicarbonate of soda, ammonium chloride being left in solution as well as some undecomposed sodium chloride and some ammonia and carbonic anhydride, chiefly in the form of bicarbonate of ammonia. The precipitated bicarbonate of soda being removed and treated in the usual manner the mother

liquor is passed directly or after treatment with carbonic anhydride into the cathode chamber of an electrolytic cell. The ammonium chloride in the mother liquor is here electrolyzed and ammonia is set free at the cathode and combines with carbonic anhydride, if present, chlorine being set free at the anode. In some cases he adopts the well-known expedient of introducing carbonic acid to the cathode chambers by a pipe. After electrolysis, the liquid which now contains some common salt and the ammonium chloride which was not decomposed by electrolysis may be used instead of brine in the first reaction of the process, in which, meeting with the ammonia liberated by electrolysis and carbonic anhydride, bicarbonate of soda is

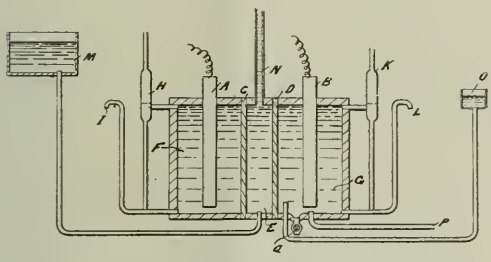


FIG. 4.—ELECTROLYTIC CELL FOR BICARBONATE OF SODA.

again precipitated and the mother liquor again treated as described. If salt be added to the mother liquor before, and carbonic anhydride be injected during electrolysis, bicarbonate of soda may be formed in the cathode chamber of the cell. In this case the chamber should be made larger than would otherwise be necessary and be furnished with suitable means for removing the precipitate. The cell is shown in Fig. 4. *A* is the anode, *B* the cathode and the anode chamber *F* is separated from the cathode chamber *G* by the intermediate chamber *E*. The gases from the anode and cathode chambers are collected in *H* and *K*, respectively. *M* is a vessel containing brine and *N* a column of liquid at the same level as in *M*, producing pressure on the electrolyte chamber. On operation *F* and *E* are filled with brine. *G* is filled from *O* with liquor containing ammonium chloride. Chlorine is evolved at *A* in small bubbles which pass over with liquid into *H*, where the separation of the chlorine gas from the liquid takes place. The solution deprived of portion of its salt by electrolysis flows off at *I*. In like manner ammonia and hydrogen are evolved at *B*. When carbonic anhydride is passed through *P* into *G* it combines with the ammonia. If the ammonium chloride liquor in this case is saturated with common salt before entering *O* bicarbonate of soda is precipitated in *G* and may be removed. (961,945, June 21, 1910.)

Electroplating.

Zinc Plating.—In a patent of Dr. E. F. KERN, of Columbia University, the following electrolyte is recommended for zinc plating: 100 parts by weight of water, 12 or more of zinc fluosilicate, 10 or more of aluminium fluosilicate, 10 or more of grape sugar, and 5 or more of ammonium fluoride. A zinc anode is employed. In certain cases, in order to restrain the separation of silica, it is advisable to add ammonium fluoride from time to time, this material being added to the bath at the outset in the proportion of one part of ammonium fluoride to one or more parts of zinc. (967,200, Aug. 16, 1910.)

Electroplating Mirrors.—A patent of EMILE HOORICKX, of Brussels, Belgium, relates to the deposition on silvered mirrors of a protecting layer of copper by electrolysis. Details of a mechanism for immersing the mirror in the electrolytic bath are described. By a single manipulation the frame carrying the mirrors and that carrying the anode are immersed in the bath in such a manner that the mirrors travel obliquely into the bath and are completely immersed in it before the frame carrying the anode has entered the bath. The oblique passage of the mirrors into the bath avoids the disturbance of the liquid which occurs when the mirrors enter the liquid normally to the surface. On the other hand, the fact that the mirrors are completely immersed before the frame carrying the anode enters the bath has the important result that there can be no localization of the current on a small portion of the silvering of the mirror which would result in its burning. (966,097, Aug. 2, 1910.)

Batteries.

Manufacture of Storage Batteries.—According to universal present practice, the spongy lead of the negative plate of the lead accumulator is produced or formed *in situ* either by repeated oxidations and reductions of the support itself (Plante process) or by reduction of a compound of lead mechanically applied to the grid (Faure-Brush process). PEDRO G. SALOM, of Philadelphia, proposes a radical change of this practice by producing the active material in bulk outside and applying it mechanically to the grid, whereby the plate is finished. A mass of spongy lead is produced by electrolytic deposition or electrolytic reduction, thoroughly washed with cold water, passed under stirring or rubbing through a sieve of 40 mesh, and is then exposed to the atmosphere and stirred to permit evaporation of the water. While the evaporation proceeds, a thin superficial coating of suboxide of lead is formed throughout the interstices of the spongy mass. For the success of the process it is of importance that this be stopped at the proper moment. The feel of the damp mass is a suitable indication which when squeezed between the fingers should have the "temper" of molding sand; it then contains still moisture, about 5 per cent in bulk of the total mass. In this state the superficially suboxidized spongy lead is applied to the grid and compressed within the compartments by means of heavy pressure (1500 lb. per square inch). The finished plate presents smooth faces, flush with the thickness of the frame and main bars. The chief advantages claimed for this process over present practice are that the new plate, while remarkably homogeneous, carries a greater quantity of spongy lead and that the cost of producing the spongy lead electrolytically in bulk and applying it mechanically to the grid is less than the cost of an equivalent amount of litharge, as employed in actual practice. (960,115, May 31, 1910.)

A New Dry Ore Concentrator.

Dry concentrating machines have attracted the attention of inventors for several reasons, but chiefly because material carrying gold or valuable mineral is found in arid places where water is not obtainable in sufficient quantity to operate wet concentrators, or because some material possesses properties which prevent successful treatment by ordinary wet concentration.

The McKesson Concentrator Company, of Colorado Springs, Col., has developed a dry concentrator which is intended to treat ore and placer gravel. As shown in the accompanying illustration, the machine is of steel construction throughout, with the exception of the belt, which is of some suitable cloth fabric. The machine is 5 ft. long, 4 ft. wide and 5 ft. high, and requires a floor space of 6 ft. x 6 ft. Supported on the steel frame is a fan and a triangular box into which an air current is forced. The triangular air chamber is closed on all sides with the exception of the top, over which the porous apron or belt is caused to move around three driving pulleys fitted with sprocket wheels and connected with chains. The two top pulleys are fixed in position, while the bottom or tension pulley is adjustable to keep the traveling belt in proper tension. The pitch of the table from feed to discharge end is adjusted by hand by means of a pulley and wire cable at the discharge end of the machine, the feed or head end being fixed, while the discharge or tailing end can be lowered to any desired point.

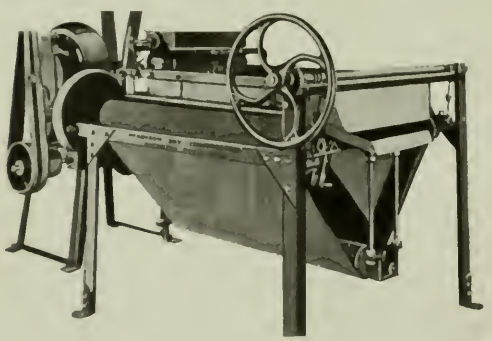
The area of the concentrating surface is 3 ft. by 4 ft., and consists of that portion of the belt which is traveling horizontally, or nearly so, across the two upper driving pulleys. The belt travels at the rate of 1½ r.p.m. to 2½ r.p.m., and at right angles to the flow of the material being treated, carrying the concentrates with it to the side of the machine, and allowing the tailing to pass down and over the edges of the belt.

Ore is fed through an adjustable automatic feeding device at the upper edge of the belt, and forms a thin bed on it. The fan, revolving at the rate of 2100 r.p.m., delivers a current of air into the air chamber beneath the concentrating surface. The air is forced through the pores of the belt in innumerable jets, and produces a stratification of the particles forming the bed of ore, forcing the lightest to the top and leaving the heaviest at the bottom. The current of air delivered by the fan is properly regulated by means of a gate between the fan and air chamber.

Supported immediately above the belt is a series of fixed iron bars, conveniently called launders, placed 6 in. apart and extending from feed to discharge end of the machine. These launders serve to direct the flow of the lighter particles down the table to the tailing discharge. Adjustable cross-riffles, so-called, made of wires placed 2 in. apart and extending in the direction of the travel of the belt, are supported above the belt between the launders.

When the machine is in operation it will be seen that the belt will be forced firmly against these launders and cross-riffles, which give the appearance of a grid-iron to the bed of ore on the belt. By lowering these cross-riffles below the launders at any point, the heaviest material on the belt settles into the depression formed and is allowed to pass under the launders with the belt, and is delivered at the side of the machine. The cross-riffles are arranged in three sections, each end section having six riffles and the middle section seven.

To recapitulate, the launders direct the flow of the upper lighter strata of the bed of ore toward the tailing discharge; the cross-riffles may be depressed at points below the launders, pressing the belt away from the launders and allowing the settled concentrate to pass with the belt to the concentrate delivery. By depressing these cross-riffles at the proper points, different minerals, such as galena and pyrite, can be withdrawn



DRY CONCENTRATOR

as separate concentrates; the minerals of highest specific gravity will be drawn off first, those of lower specific gravities next, and the lightest material will discharge as tailing. Finally, just as the concentrate is about to be delivered, provision is made to clean it with air coming up at the edge of the air chamber.

Various materials are used for the belt. For dry slime, a satin belt has been found best, for fine ore, a sateen belt, and for coarse ore, a twill or light canvas. These materials are inexpensive and durable, one sateen belt sufficing for the treatment of about 200 tons of ore. One man can make the change of belts in 20 minutes. Provision is made for cleaning the belt after it has delivered its concentrates by means of a device which periodically strikes the belt a sharp blow.

The machine works best with closely sized ore and has a capacity of 7 to 50 tons of ore, or 200 to 500 cu. yd. of placer dirt in 24 hours. There is absolutely no vibration of the machine during operation, which is quite a factor in determining its durability.

The weight of the concentrator is 850 lb., the heaviest piece weighing 110 lb.

Following are the results of recent tests on a Leadville zinc-lead-iron sulphide ore, which is one of the latest to be treated successfully.

The products were a lead concentrate, an iron middling carrying some lead, and a zinc tailing carrying all of the silicious gangue.

The test was made principally to determine the possibility of obtaining a clean lead concentrate from an ore known to carry a small percentage of that metal, with zinc and iron in larger quantities.

TEST NO. 1.

Capacity, 7½ tons per 24 hours — 29 + 39-mesh material.

	Lead per cent.	Zinc per cent.
Original ore	6.7	23.2
Lead concentrate	69.3	2.6
Iron middling	8.2	...
Zinc tailing	Trace	...

TEST NO. 2.

Capacity, 7 tons per 24 hours, of — 39 + 49-mesh material.

	Lead per cent.	Zinc per cent.
Original ore	7.8	24.3
Lead concentrate	63.0	3.4
Iron middling	10.6	...
Zinc tailing	Trace	...

TEST NO. 3.

Capacity, 7½ tons per 24 hours, of — 49 + 60-mesh material.

	Lead per cent.	Zinc per cent.
Original ore	5.4	24.8
Lead concentrate	54.7	4.5
Iron middling	7.6	..
Zinc tailing	0.9	..

TEST NO. 4.

Capacity, 36 tons per 24 hours, of — 60 + 72-mesh material

	Lead per cent.	Zinc per cent.
Original ore	6.6	24.5
Lead concentrate	70.3	2.4
Iron middling	7.6	..
Zinc tailing	Trace	.

TEST NO. 5.

Capacity, 15 tons per 24 hours, of — 72 + 97-mesh material.

	Lead per cent.	Zinc per cent.
Original ore	7.0	25.0
Lead concentrate	68.0	2.0
Iron middling	10.6	..
Zinc tailing	Trace	.

TEST NO. 6.

Capacity, 11 tons per hour, of — 97 + 109-mesh material

	Lead per cent.	Zinc per cent.
Original ore	7.3	23.0
Lead concentrate	59.5	7.5
Iron middling	6.7	..
Zinc tailing	0.4	..

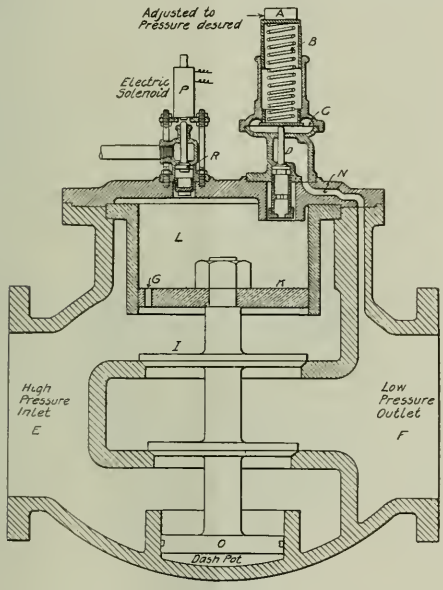
AVERAGE.

	Lead per cent.	Zinc per cent.
Original ore	6.8	24.3
Lead concentrate	64.1	3.6
Iron middlings	8.6	.
Zinc tailing	0.4	
Ratio of concentration		9.5:1
Percentage of lead recovery		94.8

Water Pressure Reducing Valve.

The following note describes a new automatic valve for reducing a high water pressure to low pressure and maintaining the reduced pressure constant; it is fitted with a quick-opening electrically operated attachment which is used whenever the full pressure is desired.

The valve as shown in the sectional view is in a closed position. The spring *B* is adjusted by the bronze sleeve *A* (in which it is encased) to the desired pressure. This spring acts on diaphragm *C*, moving auxiliary valve *D*, unseating same. The pressure or inlet side of the valve is at *E*. The low-pressure or outlet side is at *F*. The water on the high-pressure side fills the inlet chamber, exerting a pressure on upper valve *I* and lower valve *J* and large piston *K*. The pressure also passes through the port *G* into chamber *L*, and as the spring *B* is holding the auxiliary valve *D* open, the water passes on through towards port *N* to the pressure at which the valve or spring *B* is set. The pressure, still increasing, moves the



WATER-PRESSURE REDUCING VALVE.

diaphragm *C* upwards, compressing spring *B* and allowing the auxiliary valve *D* to close. The main valves *I* and *J* are forced to their seats by the initial pressure, shutting off water from the system. The pistons *O* and *K* prevent chattering and act as a perfect cushion in the opening and closing of the valve.

The electric solenoid which controls the auxiliary valve is wired up to the switchboard at the pumping station, and in case the full pressure is desired (for instance, in case of fire) the switch is closed, which causes the electric solenoid to open the small auxiliary valve and exhaust the water pressure from above the large piston *K*, which allows the main valve to instantly open to the full area. When the full pressure of water is no longer required, the switch is opened. This allows the small auxiliary valve to instantly close. The main valve then automatically returns to the regulating position. The brass ball covering the port below valve *D* prevents any water returning through the port *N* from the low-pressure side.

This valve is one of the latest designs of automatic control valves gotten out by the Golden-Anderson Specialty Company, of Pittsburgh. They are prepared to build it in sizes from 3 in. to 20 in. and for any required pressure up to about 300 lb. per square inch. It will operate satisfactorily with as big a reduction in water pressure as 3 to 1 ratio and is now in service in gravity water systems for mines and other industrial works and for municipal supply installations.

Condensers.

By Dr. Oskar Nagel.

The application of vacuum has lately been found advantageous in a good many industries for carrying out various operations, such as evaporating, drying, distilling, etc. Three different kinds of apparatus enter into this application, namely: (1) the dryer or distiller; (2) the vacuum pump; (3) the condenser.

Dryers and distillers have already been described at length in the columns of this paper. The vacuum pump is too well known, as regards construction and operation, to require further description. We will, therefore, in this article confine ourselves to a discussion of condensers.

Condensers may be divided into two groups: (1) surface condensers and (2) jet condensers. In both types condensation is effected by means of water. While, however, in the surface condenser the products of condensation do not come in contact with the water (both being separated from each other by the wall of a pipe), an intimate mixture of water and the products of condensation is effected and obtained in the jet condenser.

This also explains the field of application of each type.

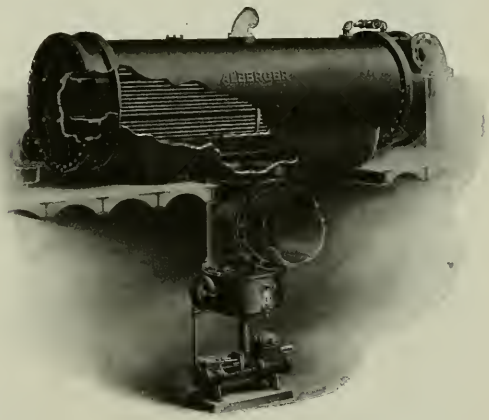


FIG. 1.—SURFACE CONDENSER.

Where the condenser products are to be kept separate from the water, the surface type should be used. Where a mixture of condensed product and water is admissible or not objectionable and where the products of condensation are simply to be removed and vacuum is the only desideratum, the jet condenser is the proper and only apparatus to use.

Fig. 1 shows an Alberger surface condenser. Its construction is readily understood from the illustration.

Fig. 2 shows a Koerting jet condenser, which operates upon the well-known jet principle. A jet of pressure water *C* passing from a smaller into a wider nozzle of a peculiar construction creates a vacuum, draws in the products to be condensed from *E*, condenses them, and discharges them together with the water. We see immediately that this type, wherever applicable, is vastly superior to the surface condenser wherever fresh water is to be had, since the jet combines the action of the vacuum pump with the operation of the condenser. Pressure water is the only requirement for starting and operating the jet condenser.

The jet condenser, Fig. 2, goes by the name of single-jet eductor condenser, and is satisfactory for medium-sized plants,

while for use in very large plants the multi-jet eductor condenser Fig. 3, is more efficient and is, therefore, in these instances to be preferred.

With the type shown in Fig. 2 a single condensing jet is used and this passes centrally through a long cylindrical tube provided with a series of rings of holes for the passage of the exhaust steam from the condenser chamber to the jet. The holes are drilled obliquely, directing the steam at suitable angle, so that it impinges on the condensing jet in the direction of flow. The steam, which strikes the condensing jet at high velocity, is condensed, and the particles of water into which it is converted having the kinetic energy due to the steam velocity cut into the jet and contribute to the momentum needed to discharge this together with the entrained air and non-condensible gases against the resistance of the atmosphere.

FIG. 2.—JET CONDENSER

The Koerting multi-jet eductor condenser, shown in Fig. 3, works on the same principle, but has instead of one central condensing jet a number of converging jets meeting and forming a single jet in the lower part of the condensing tube. This tube is cast in one piece, and consists of a series of concentric nozzles of gradually diminishing bore. The water enters at *B* and the steam coming from *A* flows through the annular passages between the nozzles, which guide it so that it impinges at suitable angle on the condensing jets.

The multi-jet condensers are considerably shorter than the single-jet apparatus of equal capacity, but in spite of this the area of contact between the steam or gas and the water is greater. A further advantage is gained by the form of the condensing tube, which in vertical section is an inverted cone. In the upper part of the tube the steam is in contact with the coldest water and there the condensation is keenest, so that a greater weight of steam is condensed per unit of area of contact than in the lower part of the tube where the water is hotter.

NOTES

Patent Lawsuit.—An interesting decision has recently been rendered in a lawsuit involving electrical panelboards. The suit against B. Altman & Company, a dry-goods house of New York City, was brought on patent No. 705,850 of July 29, 1902, granted to Hubert Krantz and later assigned to the Krantz Manufacturing Company. The defendants raised a novel point, claiming that the patent was incongruous for the reason that the claim specified an alleged invention different from that which was described in the specification and drawing. Judge Hand, sitting in the United States Circuit Court of the Southern District of New York, accepted the point and held the patent invalid because of the difference between the claim and specification and dismissed the bill with costs. Mr. Warren Wright argued for the complainant and Mr. C. P. Goepel, of Goepel & Goepel, appeared for the defendants.

Mechanical Gas Producer.—The Forter Miller Engineering Company, Pittsburgh, are issuing an attractive booklet on gas producers giving a description of the new Forter-Trump mechanical producer. This is an adaptation of the well-known Forter water-sealed producer, the Trump automatic feed and ash extractor having been added. The latter device consists of a slowly revolving wedge-shaped knife which forces out the ashes and causes the fuel bed to regularly drop in pie-shaped segments. A full description with illustrations appeared in the June issue of this journal. With air fed from both the outside and the center of the bosh a large grate area is obtained and figures on a 10-ft. producer of the Forter-Trump design are: Coal gasified, bituminous nut size, 30 tons per day, or nut and pea, 25 tons; calorific efficiency, gas used hot, 85 per cent; calorific efficiency, gas cooled to 80° F., 80 per cent; carbon in ash, 3.5 per cent; power required to run knife, 1 hp. The data on the new producer is preceded by a brief discussion of the advantages of producer gas as a fuel, and followed by illustrated descriptions of the Forter water-sealed gas producer, bell and hopper-feeding devices, the Waldburger patented feeding device, the Eynon-Evans blower, the Forter water-controlled reversing valve and the Forter water-sealed reversing valve. The water-controlled valve is a device for reversing regenerative furnaces and its unique feature is that the valve is stationary and the reversal of the circulation is accomplished through raising and lowering of bodies of water sealing or opening the passages, as desired, corresponding to the operation of the metal-sealed movable valve. The catalog will be of interest to men connected with the producer end of iron and steel industries and of glass houses and other works where producer gas for fuel is required.

Motor-Driven Pumps is the title of circular No. 1512 recently issued by the Westinghouse Electric & Manufacturing

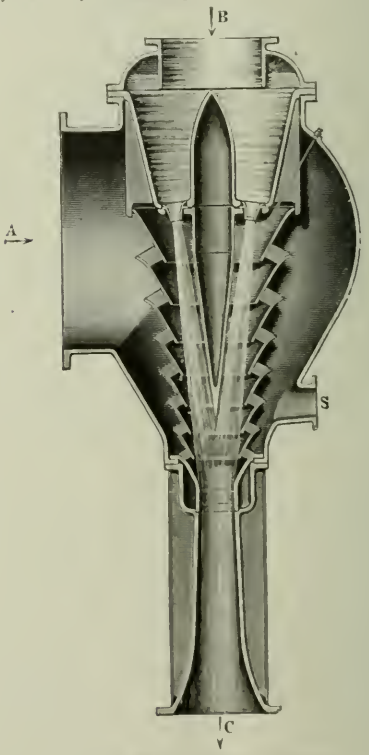


FIG. 3.—MULTI-JET EDUCTOR CONDENSER

Company, of Pittsburgh. It is pointed out that for nearly all purposes except in certain high-duty pumping installations electric drive can be recommended on account of its economy, convenience of location, cleanliness, and comparative noiselessness, small floor space and ease of control, the last feature adapting itself to either hand, remote electric or automatic

operation. Several photographs of typical motor-driven units with pumps of both reciprocating and centrifugal design are shown. For mine pumping the electric drive has still additional advantages and portable mine pumps mounted on trucks, together with the motor, are in use throughout the country.

Corrosion-Resisting Paint.—The Sterling Varnish Company, Pittsburgh, have recently sent out a booklet giving notes on their products: Sterling raw refined linseed oil, Sterling clear iron enamel, and Sterling black iron enamel (carbon base and asphalt base). With this booklet has been included a sample of thin gage sheet iron enameled and lettered. The lettering shows quite readily through the clear iron enamel, which is designed to give to structural steel a preliminary protective coating which can be applied all over without obliterating the necessary markings or preventing the detection of flaws, scale or rust. The black iron enamel is intended for use on exposed steel work, bridges, buildings, roofs, etc., and it has a good record for withstanding severe service. Sterling raw refined linseed oil is the base of the above products, and is also put upon the market separately. The purity of the oil in any paint is of prime importance as ordinary oil and much of the so-called refined oil contains impurities which are dissolved by rain or dampness, leaving the paint coating full of little flaws which permit rapid disintegration. By the "Todd processes" practically all of these soluble impurities are removed and the irregularities which ordinarily give trouble are absent. The "ten commandments," one might say, to which the Sterling Varnish Company subscribes are summed up in the following: "If you keep a coat of paint free from any opening through which outside influences can enter and have its coefficient of expansion properly adjusted, you attain the maximum of durability."

The Stearns-Roger Manufacturing Company, Denver, has issued an illustrated catalog describing the Edwards mechanical roasting and chloridizing furnaces, and the Edwards dryers and coolers. A number of installations of both simplex and duplex furnaces, of the tilting and fixed hearth types, are shown. The roasters are widely used in Australia, as well as in this country.

The Isbell Vanner is to be installed in the new mills of the Chino and Ray Consolidated Copper Companies. Over 1500 of these machines will be used, being the largest number of one make of vanners ever ordered at one time. The Galigher Machinery Company, of Salt Lake City, received the contract, and will have the machines made by the Allis Chalmers Company.

PERSONAL

Mr. **H. C. Beeler,** mining engineer of Denver, recently returned from Wyoming, where he is engaged in the development of asbestos properties.

Mr. **A. C. Dart,** who has held the chair of mining engineering in the University of Wyoming, has resigned that position to accept the general managership of the Rambler Copper & Platinum Company, at Holmes, Wyo.

Mr. **Courteney DeKalb** has resigned his position as one of the editors of the *Mining and Scientific Press,* to become general manager of the Pacific Smelting & Mining Company, Sonora, Mexico.

Mr. **Carl F. Dietz,** of the firm of Dietz & Keedy, Boston, sailed for Europe last month, and will attend the joint meeting of the Mechanical Engineers at Birmingham and London.

Mr. **F. W. Draper** is now with the Arizona Smelting Company, Humboldt, Ariz., having recently returned to the United States after fulfilling a two years' engagement in the Urals, Russia.

Mr. **J. E. Edwards,** representing the Edwards furnace, has removed from Colorado Springs to Denver and has an office with the Stearns-Roger Manufacturing Company.

Mr. **Ivan E. Goodner,** formerly with the Bogardus Testing Laboratories, of Seattle, Wash., is now in the new concentrates cyaniding plant of the Alaska Treadwell Gold Mining Company and will be located at Treadwell, Alaska.

Mr. **John A. Hunter,** formerly in the Southwestern States and Mexico, has established a metallurgical laboratory in Los Angeles.

Mr. **D. C. Jackling,** general manager of the Utah Copper Company, recently made a trip of inspection through Nevada and Arizona. He was accompanied by other officials of the Guggenheim Exploration Company and American Smelting & Refining Company.

Mr. **C. Colcock Jones** has returned to Los Angeles from an extended professional trip in Esmerelda County, Nevada.

Mr. **Jules Labarthe,** who has been superintendent of the Canadian Smelting Works, at Trail, B. C., for the past 10 years, has taken charge of the construction and operation of the new smelter of the Mason Valley Mining Company, at Yerington, Nev.

Mr. **J. R. Masson,** of Australia, is the United States representative of the A-Z agitator, and has offices at 2 Ferguson Building, Denver.

Dr. **Jos. W. Richards,** of Lehigh University, has sailed for Europe to attend the International Geological Congress in Stockholm.

Prof. **Robert H. Richards,** of Boston, is spending a part of the summer in Denver.

Mr. **Guy M. Vail,** of Portland, Ore., has recently been in Nevada on professional business.

Mr. **Pope Yeatman** was recently in Leadville, Colo., as an arbiter in differences between the Empire Zinc Company and the Western Mining Company. He was called upon to decide the question of ownership of ore in adjoining properties on the same ore body.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ELECTRIC FURNACES (*Continued*).

No. 675,576, June 18, 1901, Alfred H. Cowles, Cleveland, Ohio.

The furnace has a crucible, whose walls are permeable to gases, surrounded by double walls to form two chambers, the inner one for permitting gases to be drawn off, the outer one having suitable connections to form a water jacket. An electrode depends into the crucible through a cover of compacted carbon incased in an iron shell, insulating material being placed between the bottom of the cover and the crucible. The electrode is surrounded by a pipe, connected to but insulated from the iron shell of the top, this pipe having a chamber at its top into which is introduced a small quantity of hydrocarbon, neutral or reducing gas or gas-forming liquid to prevent any vapor passing up around the electrode. The electrode terminal passes through a stuffing-box into the chamber, current passing from one terminal to the electrode through the charge to the crucible and surrounding casings or jackets to the other terminal.

No. 676,985, June 25, 1901, Hugo Koller, Nuremberg, Germany.

Resistance type especially for calcium carbide. End electrodes project into horizontal chamber. Between and in line with these electrodes are a series of disconnected electrodes supported above the bottom of the furnace. A second series of disconnected electrodes are located on the bottom of the furnace between the ends of the raised electrodes. The lower electrodes may have troughs formed in their tops to receive molten carbide. The charge covers all the electrodes. The furnace is started by inserting a small resistance such as short carbons

between adjacent ends of electrode blocks. The object of the invention is to have as few feed conductors as possible and to increase the limits of voltage between which furnace works advantageously.

No. 677,070, June 25, 1901, Arthur H. Eddy, of Windsor, Connecticut.

Resistance type, especially for fusing enamel upon clay or metal. A longitudinal furnace is shown having transverse resistors, consisting of groups of parallel carbon rods enclosed in larger porcelain tubes, both electrically connected at each end to water-cooled clamps. After being heated to a high degree, the porcelain tubes become fair conductors and carbons may then be removed. The resistors may be arranged to provide for gradually increasing and then decreasing temperatures, as articles are fed through the furnace. The porcelain tubes protect the fused enamel from disintegrated carbon particles and from air jackets which assist in equalizing temperature.

No. 677,439, July 2, 1901, Ramon Chavarria Contardo, of Sevres, France.

Arc type. Prevents electrodes from getting hot outside of the furnace by means of a water-cooled hollow metal sleeve, through which electrode slides, embedded in the refractory wall of the furnace and carefully insulated from metal fittings. This sleeve may be made in two pieces, copper plates being wound spirally around the electrode, and the half-sleeves tightened in place by spring rings. Electrical connection to the electrode is made through bronze segments (which may be water-cooled (surrounding and pressed against the electrodes by concentric rings, each segment being connected to a strand of the cable. Successive carbons are attached, the rear ends having conical holes into which project correspondingly shaped front ends, the mechanical connection being made by a carbon cotter and the electrical connection by varnish containing graphite and sugar or tar. A single sleeve composed of hollow bronze water-cooled segments may serve both for cooling and electrical connection.

BOOK REVIEWS.

Die Wärmetechnik des Siemens-Martinofens. By. Dr.-Ing. **F. Mayer,** Professor of Metallurgical Engineering at the Institute of Technology in Aachen. 123 pages, 29 diagrams. Halle: Wilhelm Knapp, 1909.

Dr. Mayer investigated most thoroughly a 30-ton basic open-hearth furnace at the Rothe Erde works near Aachen, working on a charge of 72-73 per cent scrap, 15-20 per cent pig iron, ored down by hematite and deoxidized by spiegeleisen and ferro-manganese. The yield was 34-35 tons, time averaging six hours, coal used in producers 250 kg per ton of steel, surface of metal 0.6 sq. m per ton of steel.

Measurements were made for a month of all the details of operation, including particularly analyses of the producer gas and chimney gas, and temperatures at the upper outlet of all four regenerators, at the gas and air inlets, and at the chimney flue, using Le Chatelier thermo-elements. The temperature curves through the regenerators, also the change in composition of the gas as it was being heated, were followed. All the original data are given in extensive tables, so that any one can use them to draw his own conclusions. The principal conclusions drawn by Dr. Mayer on this particular furnace (but probably largely true of all open-hearth furnaces) are interesting and have an important bearing on open-hearth practice.

The air should be heated to 1,450° to 1500°, the gas to 1300° to 1,350°. Reversing every 20 minutes, the temperatures at the outlet of the air regenerator should not fall over 100° to 120°, of the gas regenerator not over 180° to 200°

The temperature difference between the hot air and the hottest part of the air regenerator is not over 20°.

The depth to which the heat enters the regenerator bricks varies directly with the time between reversals; the effective weight of regenerator material is therefore dependent on the same.

With producer gas at 600°, the gas regenerator returns only 60 to 70 per cent as much heat to the furnace as the air regenerator.

The producer gas could be cooled in order to remove its moisture and sulphur, and yet be heated to the same temperature if the gas regenerator were made 10 per cent larger.

In regular running, about one-sixth of the gas is burned in the regenerators after leaving the furnace. This is because of the great velocity with which the gases traverse the hearth.

Hydrogen gas is consumed proportionately quite as completely in the furnace as CO gas. Both CO and H^2 burn in the regenerators.

This heat developed in the regenerators is practically lost to the furnace.

No heat in the chimney gases below 600° can be utilized.

Chimney gases at 600° give a much better draft than at 300°, which is useful for running the furnace, as it enables it to draw in more air and gas and do faster smelting.

Hydrogen is not advantageous in the producer gas, because it makes it lighter and more inclined to rise and burn along the roof of the furnace, which it thus injures.

Only half the steam blown into the producers was decomposed therein.

In the regenerators, each cubic meter of checker work contained 790 kg of bricks, and had 16 sq. m of surface.

The air chamber had 20 sq. m and the gas chamber 15.5 sq. m of checker surface, per kilogram of coal burned per minute. Each square meter of checker surface yielded 7400 calories per hour to the air, and 4400 calories per hour to the gas.

The regenerators lost by radiation and conduction about 17 per cent of the heat intercepted by them (including that developed in them by combustion).

The body or laboratory of the furnace utilized for heating charge and supplying radiation and conduction, 43 per cent of the calorific power of the coal used in the producers.

Many of the above conclusions evidently apply only to the particular furnace investigated by Dr. Mayer, but some are of great value to open-hearth practice in general. We recommend the close study of this book to all interested in the theory or practice of the regenerative gas furnace, and take this opportunity to congratulate the author on his splendid work.

NEW BOOKS.

Allen, Alfred H. Commercial organic analysis. Fourth edition. Vol. 3. Hydrocarbons, asphalt, phenols, aromatic acids, modern explosives. 645 pages, illustrated. Price $5. Philadelphia: P. Blackiston's Sons & Co.

Benson, Harrison T. Compendium on mines, mining, minerals, ores, rocks; weights of metals and rocks; effect of heat on various substances; water measure, tanks and piping; atomic weights explained, chemical symbols, etc. Illustrated. Price $2. Denver, Colo.: Hall and Williams

Clarke, Fk. Wigglesworth. Recalculation of the atomic weights. Third edition, revised and enlarged. 552 pages. Smith Inst.

Coles-Finch, W. Water, its origin and use. Illustrated. Price $5 net. New York: D. Van Nostrand & Co.

Crane, Wa. R. Ore mining methods. Illustrated. Price, $3 net. New York. John Wiley & Sons.

Hegler, Edward C. In memory of Edward C. Hegler. No paging. Paper gratis. Chicago: Open Court.

Osborn, S. Stafford. Prospector's field book and guide in the search for and the easy determination of ores and other useful minerals. Eighth edition. Price, $1.50. Baird.

Treadwell, F. Pearson. Analytical chemistry. Translated from the German by W. T. Hall. Vol. 2. Quantitative analysis. Second edition, thoroughly revised and enlarged. Price $4. New York: John Wiley & Sons.

Metallurgical and Chemical Engineering

A Consolidation of Electrochemical and Metallurgical Industry and Iron and Steel Magazine

Vol. VIII. New York, October, 1910 No. 10

PUBLISHED MONTHLY BY THE

Electrochemical Publishing Company

239 West 39th Street, New York.

Pittsburgh Office, 201 Oliver Building.
Western Office, 1510 Court Place, Denver, Colo.
European Office, Hastings House, Norfolk St., Strand, London, Eng.

J. W. Richards, Ph.D., President.
Curtis E. Whittlesey, Treasurer. E. F. Roeber, Secretary.
Telephone Call: 4700 Bryant.

E. F. Roeber, Ph.D., Editor.
H. C. Parmelee, Western Editor.

Yearly subscription price for United States, Mexico and United States dependencies, $2.00; for all other countries, $2.50 (European exchange, 10 shillings, 10 marks, 12.50 francs).

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Entered as Second-Class Matter at the Post Office at New York, N. Y., under the Act of Congress, March 3, 1879

New York, October, 1910.

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Chicago Meeting of the American Electrochemical Society.

The program of the Chicago meeting of the American Electrochemical Society, printed elsewhere in this issue, is so attractive in all its features that a great many members from the East may be expected to attend it. The variety of excursions is certainly alluring. And it is a pleasure to note the many contributions from Western members, especially from Chicago and the University of Wisconsin, to the program of papers. We are sure that the meeting will be not only highly enjoyable, but that this first excursion to the West will have a far-reaching effect on the further development of the society.

Fixation of Atmospheric Nitrogen in the United States.

It is only during the last few years that the fascinating problem of combining by electrical means the nitrogen and oxygen in atmospheric air for the production of nitrates and nitric acid has been sufficiently worked out in detail to pass out of the hands of the research chemist into those of the works manager. This development in all its technical phases has been recorded in great detail in former issues and volumes of this journal. We shall say here only a few words on the geographical aspect of the commercial problem. The first commercial plants were erected in Norway for the reason that exceedingly cheap electric power (costing something like $4 or $5 per electric hp-year) is available there right on the coast, from where transportation on ocean steamers is also cheap. Then followed a commercial plant in Tyrol and some other plants, perhaps still more or less experimental, in France. Cheap electric power was the keynote of all these undertakings. Just for the same reason nothing has been done in the past in this country, where the cheapest electric power is sold at three times the cost quoted above for Norway. The fact that a 4000-hp experimental plant is now about to be erected in South Carolina is, therefore, of considerable interest and importance. Both the Guye process and the Pauling process shall be tried. Again the availability of large quantities of power is a decisive element, the undertaking being affiliated with the large water-power developments of the Southern Power Company. Of equal importance is the market which exists in the South for fertilizers. The outcome of this work will be watched with much interest. The technical problem appears to be solved, and it was all important not to make a commercial mistake now. It would seem that if the fixation of atmospheric nitrogen shall succeed in this country, the logical place has been chosen for the first trial.

Two Appointments

The appointment of Dr. Joseph A. Holmes to the directorship of the Bureau of Mines meets with the unanimous ap-

proval of the interests most concerned with the work of the new bureau. The appointment is gratifying for two reasons: first, it is but the just recognition of the qualifications and past work of the recipient, and, second, it augurs well for the success of the new bureau, for Dr. Holmes has shown his ability as an organizer and his knowledge of the needs of the industries which the bureau is to aid. Another appointment of interest to the metallurgical profession and one which has called forth approval from many sources, is that of Mr. George E. Roberts to the directorship of the Mint. Mr. Roberts returns to the position formerly held by him after two years service as president of a large national bank.

The Chemistry of Slime.

Improvements in the handling of slime in the cyanide process have been almost entirely mechanical, and chemical considerations have been neglected, although the latter may materially affect the former. Recent investigations in the laboratory of the Clay Products Section of the Technologic Branch of the United States Geological Survey indicate that the principles of colloid chemistry may be applied to advantage in the treatment of slime, since some metallurgical slimes are true clays and the colloidal matter from certain crushed rocks is identical with portions of colloidal matter found in clays. A slime which is largely colloidal in nature is difficult to settle or filter, but it can be improved by coagulation. Experiments by Mr. H. E. Ashley, of the laboratory mentioned, although undertaken for the clay industry, may be of importance in metallurgy. The success of all sliming methods in cyaniding is dependent on the suppression of colloidal slime; the more completely it is suppressed the more rapid and successful the filtration. It is suggested that the formation of colloidal matter can be suppressed greatly by grinding the ore in coagulating solutions, such as lime, and sulphates and chlorides of calcium and magnesium.

The Destruction of Cyanide.

The practice of filling worked-out areas of metal mines with the tailing from milling operations is not new, although practiced only in those districts where mining has been conducted for long periods of time, or where the magnitude of operations makes it necessary to provide cheap and efficient means of filling to prevent subsidences of ground and to protect the lives of the miners. When tailing from cyanide works is to be used for this purpose, it is necessary to render it innocuous as a possible source of poison after it is placed underground. To do this effectively and economically constitutes an interesting chemical problem on account of the complex reactions which may occur. The complete destruction of cyanide would involve the dissociation of the cyanogen radical, and some of the methods proposed look toward this end; but probably the formation of non-poisonous salts of cyanide would serve the purpose, provided assurance could be given that subsequent reactions occurring after underground filling would not again release poisonous gases. The chemists of South Africa are at present engaged on this problem, and some most interesting experimental work has resulted.

In investigating the possibility of utilizing the reaction between cyanide and ferrous salts Dr. Moir, president of the South African Society of Chemists and Metallurgists, found that his experiments contradicted practically all of the text-book information on the ferrocyanide reaction. This is rather startling, but indicates that in attacking any problem it is not sufficient to apply stated principles relating to the subject, but that independent research may be necessary to determine the applicability of the principles. The results of his work may be summed up thus: In the reaction between cyanide and ferrous salts, if excess of alkali is used, less ferrocyanide is formed than when smaller quantities of alkali are present; a rise of temperature above 20° C. is harmful to the reaction; the results are almost independent of dilution, and the reaction is as complete in 5 to 10 seconds as after long standing; alkalinity must be carefully adjusted to the amount of iron used, otherwise an excess of ferrous solution gives no better results than that theoretically indicated in the reaction. The method is applicable, however, as the reaction can be made sufficiently complete for industrial purposes, the cyanide having been reduced to three or four parts per million.

The more complete destruction of cyanide is proposed by other investigators, by the use of alkaline permanganates, aldehydes or alkaline picrates. The first-mentioned substance oxidizes the cyanide to cyanate, but the latter two give organic products with potassium cyanide, although they have no effect on sulphocyanides or ferrocyanides. With formaldehyde the reaction is almost instantaneous in the cold, but the other reactions require from two to four hours. For practical trials of the cyanicides it is recommended that the chemicals be used in the following proportions: For every 100 tons of sand, 20 tons of solution of 0.045 per cent $KMnO_4$ or 0.027 per cent formalin (at 32 per cent formaldehyde), or 0.019 per cent picric acid as alkaline picrate. The direct expense for the chemicals would be probably from 1.2 cents to 2.6 cents per ton of sand treated.

Railroad Consumption of Steel.

A time has been reached when it is necessary to make a clear distinction between utilization, deterioration and destruction of steel. The broad term "consumption" has been used loosely and should be replaced, where possible, by more precise terms. It has been customary, for instance, to speak of the "consumption" of steel by the railroads. Few applications of steel involve less destruction or dissipation of material, as railroad scrap yards bear eloquent testimony. A rail unfit for further service weighs within a few per cent as much as it did when new. Car wheels lose between 5 and 10 per cent and brake shoes somewhat more. Locomotives, car parts, bridges and viaducts lose scarcely any material.

There was a time when the railroads absorbed more than one-half the total iron and steel production of the country. That was when the railroads were being built. They were destroying, or reducing to scrap, very little material. The tonnage of material dissipated or reduced to a condition requiring reworking by the entire railroad system of the country is chiefly a function of the total amount of work done, and as passenger service is light relative to freight service and has increased more or less uniformly with freight movement, a concise presentation of railroad activity is furnished by freight ton-mileage statistics. The ton-mileage of the steam

roads of the country, net tons moved one mile, has been as follows, according to Poor's statistics:

1885	49,151,894,469
1890	79,192,985,125
1895	82,219,900,498
1900	141,162,109,413
1905	187,375,621,537
1906	216,653,795,696
1907	233,137,507,807
1908	215,698,911,350
1909	227,198,932,735

It is instructive to recall that in 1887 rail production amounted to 2,139,640 tons and that as large an annual total was not reached again until 1899, when production amounted to 2,272,700 tons. This long hiatus, during which freight ton-mileage increased from 61,561,069,996 in 1887 to 126,991,703,110 in 1899 illustrates the fact that in those days rails were not required principally because other rails had worn out. The railroads are now doing almost four times as much work as in 1887, but are not calling for four times as large a tonnage of rails. Indeed, it is evident from the ton-mileage table just given that it is only in recent years that rails are being worn out rapidly. From such statistics as are available an effort has been made to determine a factor of ton-mileage per ton of rails worn out. The statistics are too meager to promise an accurate result, but it is probable that the factor, subject to correction for local conditions, probably does not lie outside the limits of 250,000 and 300,000 ton-miles per ton of rails.

When the rail is made unfit for further service it comes into the steel industry for treatment. A limited tonnage unfit merely because of deformation of the head is "renewed," another tonnage is rerolled into lighter sections, another tonnage is rolled into concrete reinforcing bars, bedstead angles and the like and the remainder, the largest tonnage of all, is remelted in the open-hearth steel furnace. For the future there is promised a large flow of steel through railroad use, larger in proportion to the absorption than has obtained in the past. The flow of steel through the period of utilization is more distinct in the case of rails than in the case of any other large channel.

New Markets for Cobalt.

For a great many years the complex arsenides of silver cobalt, nickel, and iron have been a steady source of supply to the small metallurgical works of Saxony and Silesia. These ores occur in small proportions in several of the famous old mines of Europe. Their production has been small, however, and was in proportion to the consumption of cobalt oxide, for the ores were not so very high in silver. About six years ago ores of the same general character, but phenomenally rich in silver, were discovered in quantity in the northern part of Ontario. The mines of the so-called "Cobalt District," where well-located and well-managed, have been exceedingly profitable and now furnish 12 per cent of the world's silver production. The silver is present in such large amounts (about 3 per cent on the average—and it must be remembered that copper ores of less than 3 per cent can be a source of profit nowadays) that the other valuable products can be neglected, and have been neglected, commercially for some years. The course of treatment used by the principal purchaser of these ores, the Orford Copper Company, is roasting and smelting in short, small blast furnaces. From these operations three products are produced: first, a crude arsenic oxide which is refined by a volatilizing roast: second, a crude silver bullion, "800 to 850 fine."which is sold to the silver refiner, and, third, a speiss containing approximately 20 per cent cobalt, 10 per cent nickel and less than 1 per cent silver. This speiss is roasted and subjected to a chemical process for the production of fine silver, refined nickel oxide, and refined cobalt oxide. The principal source of revenue to the mining company is, however, silver.

Naturally, the increase in production of these ores has resulted in a glut in the market for cobalt oxide and arsenious oxide. An analysis of the reports of the four principal companies shows that they are producing silver for less than 20 cents per ounce. Consequently the by-products are not playing, as yet, any great commercial rôle. Indeed, some of the ores are smelted in the lead blast furnace for the silver alone and the other three constituents wasted. And in general arsenic oxide is now selling for one-half of its usual price and cobalt oxide for less than one-third of its former price.

However, in this complex modern civilization of ours the uses and needs of industry are many and varied. And the old law of economics that low prices stimulate consumption and extremely low prices increase consumption by finding new uses for the commodity seems now in this instance to be operating with rapidity and certainty. With the low prices of arsenious oxide, the demands for the fungicides of which it is the base have been increased. Especially is this true of lead arsenate, which is used by the fruit growers of this country to kill the San José scale and other deleterious parasites. The market for cobalt oxide is also steadily increasing. It has been long known that mixed, 1 part to 10,000 parts, with china clay cobalt oxide gives to the ceramics a blue cast that neutralizes the yellow cast formed by the oxide of iron. It is almost humorous to consider that the laundress uses bluing in the water for a similar purpose. Naturally, when the price is low, potteries use more cobalt oxide and produce dead-white pottery cheaper.

Cobalt, when in the metallic state, is a great hardening constituent of alloys and has been found to produce alloys of different and superior qualities to those alloys where its twin chemical brother replaces it. Furthermore, cobalt oxide can be made the base of several blue or green mineral pigments. The low price of cobalt oxide has stimulated chemical research in several lines looking to new commercial uses for cobalt and cobalt oxide. This work is eventually sure to find new and permanent applications. Increased permanent consumption will give stability to the industry. Production will be increased to meet future requirements. The veins of the Cobalt District can be characterized as rich, narrow, and short. Hence, the cost of development is sure to increase at depth, for prospecting can no longer be done by cheap surface trenching. For the future the camp must look to the low-grade ores with values in by-products for the profits. A similar evolution has been experienced in Leadville, where the once-despised zinc-sulphide ores are a large source of profit. The metallurgical treatment of these ores and the working up of the resultant by-products by chemical methods present a field actually fascinating and potentially profitable to the research chemist.

Program of Chicago Meeting of American Electrochemical Society.

The eighteenth general meeting of the American Electrochemical Society will be held in Chicago from Oct. 13 to 15 (Thursday to Saturday). Hotel headquarters will be at the Congress Hotel (Auditorium Annex).

The morning sessions of Thursday and Friday will be held in the Florentine Room of the Congress Hotel, the morning session of Saturday at the Kent Chemical Laboratory of the University of Chicago.

For the afternoons a number of highly interesting alternative excursions have been arranged as follows:

Thursday Afternoon.—Union stockyards and the packing plants of Swift & Company, Armour & Company and Morris & Company;

Or, Fisk Street power house and Market Street transformer and storage-battery station of the Commonwealth Edison Company;

Or, the telephone factory of the Western Electric Company;

Or, the factory of the Automatic Electric Company (automatic telephones), Market Street storage-battery station of the Commonwealth Edison Company, and demonstration exhibit of electric furnaces of the Hoskins Manufacturing Company.

Friday Afternoon.—South Chicago works of Illinois Steel Company (with 15-ton Heroult electric steel furnace);

Or, Gary steel works at Gary, Ind.;

Or, Sears, Roebuck & Company and the Washington and Franklin Streets exchanges of the Chicago Telephone Company.

Saturday Afternoon.—Laboratories and museums of the University of Chicago, and Field Columbian Museum;

Or, Lockport power station and drainage canal.

A special and very attractive program has been arranged by the ladies' committee for the entertainment of visiting ladies.

On Friday evening a subscription dinner will be held at the Congress Hotel.

The following is a list of the papers which will be presented and discussed at the professional sessions:

Method for determining thermal conductivity. By Carl Hering.

Heat flow through electric furnace walls. By F. T. Snyder.

The Hiorth electric steel furnace. By Jos. W. Richards.

Cast copper of high electrical conductivity. By E. Weintraub.

The electric furnace and its applications. By Marcus Ruthenburg.

Electrolytic method of producing metals. By John W. Beckman.

Reduction of tin dross in the electric furnace. By R. S. Wile.

Notes on tribo-luminescence. By W. S. Andrews.

The proximate constituents of the chemical elements with reference to radioactivity and electrons. By Gustavus D. Hinrichs.

Electrochemical preparation of calcium. By A. R. Johnson.

Efficiency in the electrolytic preparation of calcium. By Francis C. Frary, Henry R. Bicknell and Carl A. Tronson.

Electrochemistry of polonium and other radioactive elements. By A. R. Johnson.

The present status of ozone. By R. D. Small.

The evolution of the lead storage battery. By Pedro G. Salom.

A new secondary cell. By W. S. Block.

The self-discharge of Plante and Faure storage batteries. By O. W. Brown and W. G. Bowers.

Corrosion of metallic food containers, causes and prevention. By Edward Gudeman.

Electrolytic corrosion of various metallic anodes in a liquid ammonia solution, ammonium trinitride. By A. W. Brown, M. E. Holmes and J. S. King, Jr.

Electrolysis of solutions of ammonium trinitride and of potassium amide in liquid ammonia. By A. W. Brown and M. E. Holmes.

On the electrolysis of some cyanides. By H. E. Patten and W. R. Mott.

Application of electrostatic separation to ore dressing. By F. S. McGregor.

The isolation of an ion and a precision measurement of its charge. By Robert A. Milliken.

Phenomenon of crystallization through membranes. By J. H. Walton, Jr.

On the relative basicity of the metals as shown by their power to replace one another in chemical compounds. By L. Kahlenberg.

On the effect of water in causing chemical reactions. By David Klein.

The physical and magnetic hardness of some electrolytic iron alloys. By James Aston.

Dr. William H. Walker, of the Massachusetts Institute of Technology, is president; Dr. Joseph W. Richards, Lehigh University, South Bethlehem, Pa., is secretary of the American Electrochemical Society.

Prof. H. N. McCoy is the chairman and Mr. Arva B. Marvin is the secretary of the local Chicago committee.

The Western Metallurgical Field.

Copper.

The report of the Copper Producers' Association for August, issued Sept. 8, showed a reduction in surplus stock of 1,759,433 lb. as compared with the figures for July. This is the first reduction that has been noted for many months. During May the surplus was increased by 18,411,814 lb.; during June by 7,960,044 lb., and during July by 2,255,661 lb. Inasmuch as the effects of curtailment in production begun in August had not had time to affect the situation the condition is regarded as favorable.

The August production of some of the Western mines and redrction works was lower than usual. In Butte the Clark properties were closed down, with a consequent reduction of 1,250,000 lb. of copper, being the largest part of the total reduction in Butte mines of 1,646,280 lb. as compared with July. The total production of Butte mines for August was 317,210 tons of ore containing 21,724,100 lb. of copper. At Ely, Nev., one of the four units of the concentrator of the Steptoe Valley Smelting & Mining Company, at McGill, was closed, and the production of ore at Copper Flat was reduced from 8000 tons or 9000 tons per day to 6000 tons or 7000 tons. The reverberatory capacity of the smelter has not been reduced, however, and four furnaces are still being operated at full capacity. The Yampa smelter, at Bingham, Utah, also closed Aug. 1, but the ore that was treated at this point will be diverted to other Utah smelters, so that the closing of the plant will have little effect on the output of copper.

In the meantime the plant of the International Company is coming forward, matte having been tapped and converted late in August. It is also reported that the Garfield smelter is to be enlarged to handle the increased output of the remodelled mills of the Utah Copper Company. New reverberatories, flues and a new stack are to be added to the present plant. All of this will not be completed for some months, probably a year; but when completed the plant will be one of the largest in the West, having a capacity of 16,000,000 lb. per month. The new copper properties—Miami, Ray and Chino—will probably come forward slowly, so that when they have entered the list of producers the demand for copper probably will have increased so that their output will be absorbed readily.

Statistics on cost of copper production have been issued by most of the large companies, and some of them have shown remarkably low costs. It has remained for the Kennett (Cal.)

Works of the U. S. Smelting, Refining & Mining Company to show a cost of 4½ cents per pound for copper made there. This figure is arrived at by taking into consideration the gold and silver value in the ore treated, and as this is higher than in most ores, the effect on cost of production is more marked.

Important Tailings Suit Decision.

About six years ago suits were instituted by the ranchers along the Coeur d'Alene River against the mining and milling companies of northern Idaho, praying for relief from the effects of dumping tailings in the river, whereby it was claimed that the river bed was filled with debris causing the water to overflow the banks and damage the adjoining lands. In all, 65 suits were filed against the various companies, and damages aggregating $1 233,000 were claimed.

Recently one of the cases has been decided by Judge Dietrich in the Federal Court at Moscow, Idaho, by awarding $1 damage to nine plaintiffs, and dismissing the suit. A striking feature of the case was the fact that the defendant company, the Bunker Hill & Sullivan Mining & Concentrating Company, introduced no evidence, and the court concluded that the plaintiff had not shown sufficient evidence to warrant further consideration. The decision is of great importance to the Coeur d'Alene district, as it will settle a vexed question unless it is reversed by the Court of Appeals.

Mexican Institute of Mining and Metallurgy.

A general meeting of the Institute was held in Mexico City on Sept. 13 commemorative of the first year's work. President A. Grothe reviewed the results accomplished, and a number of interesting papers were read. This young Institute has met with a favorable reception at the hands of engineers in Mexico, and the *Informes y Memorias* published have reflected the interest shown. Mexico has presented metallurgical problems different from those usually met with, and it has been the aim of the Institute to foster a unity of interest among professional men and disseminate information that would enable them to solve their problems quickly.

As an example of the excellent metallurgical work done on silver ore in Mexico we quote the following from the last bulletin, Vol. 1, No. 6, referring to the plant of the San Rafael y Anexas Company, of Pachuca: "Prior to 1909 about 1000 grams of silver per ton were required to cover mining and milling charges. Since cyaniding was adopted from 300 grams to 350 grams per ton will pay for all costs, and 500,000 tons of ore, half in the filling and half on the dump, averaging some 500 grams, have become available at once and will be milled with a good profit. Since the mill was started in 1909 we have been short of power and for this reason forced to curtail the production and to resort to an expensive sorting in order to improve the value of the ore milled, raising the cost of production." The average assay of the ore treated is given as 888 grams silver and 4.08 grams gold. The waste from the sorting runs from 166 grams to 300 grams silver. Cost of milling, $4.04 (Mexican) per ton.

The mill was described in detail in Institute Bulletin No. 3, Vol. 1, but since then the capacity has been increased and the plant improved so that at present it treats from 400 tons to 500 tons per day, according to the class of ore handled. The silver extraction recently has been raised to 94 per cent and the gold to 96 per cent, the cost per ton being as stated above. The equipment of the mill comprises in the main the following: 3 Blake crushers, 9 x 15; 1 set of rolls, 36 x 16; 80 stamps; 18 tables; 7 Dorr classifiers; 7 Dorr pulp thickeners; 7 Krupp tube mills; 12 Pachuca tanks, 15 ft. x 45 ft.; 3 Moore filter units, each with 80 10 ft. x 6 ft. leaves.

Cyaniding in Colorado.

There is a marked tendency to favor the cyanide process in Colorado in the treatment of both old and new ores. The success of the Golden Cycle mill treating Cripple Creek ores by roasting and cyaniding, and the application of cyaniding to the chlorination tailings of the Portland and Standard mills has directed attention to the possibilities of the process. The new Portland and Independence mills at Cripple Creek, treating low-grade mine ore and dump ore respectively, are in successful operation on material worth about $3 to $4 per ton. More recently, however, tests on ores from other districts have proved that the process may be extended to the treatment of sulphide concentrates and raw silicious sulphide ores. In the Georgetown district an old stamp-concentration mill is being remodeled into a cyanide plant for the treatment of the raw ore. Reports from the vicinity of Breckenridge indicate the possibility of gaining 90 per cent extraction from ores that are not now treated, although roasting will have to be used in connection with cyaniding on these ores. Other experimental work on Boulder and Black Hawk district ores and concentrates indicate the success of the process.

Colorado has, for so many years, been a smelting center that many mine owners were loth to listen to suggestions of other methods for treating their ore and made few attempts to work out a treatment method if smelting could not be made to pay. This attitude is changing, however, and while smelting will continue as a means of treating a large tonnage of Colorado ore, the smelters are now on reduced supply and the outlook is not bright for an increase.

Leadville Zinc.

The recent report of the discovery of bodies of zinc carbonate in a number of Leadville mines has attracted attention to the possibilities of a new era in Leadville mining and metallurgy. Sulphides of zinc have for some years been mined and treated by magnetic separation methods, but within a short time carbonates of zinc has been found in the Maid of Erin, Adams, Wolftone, Mahala, Henriet, Morning Star, Waterloo, Chief, and other old Leadville mines. The report is to the effect that active production will commence in some of these properties very soon. The grade of the zinc carbonate discovered is reported to be from 25 to 40 per cent zinc.

Non-Ferrous Metal Market.

The September market for non-ferrous metals was almost without feature. Prices for copper and lead have remained almost stationary, but spelter has experienced an advance and holds a better position than for some time, with prospects of further improvement. Tin was subject to a slight rise during the early part of the month, but has receded to a lower level.

Copper.—With the exception of the export trade there has been little important business transacted in this metal. The refinery statistics for August naturally did not show the effect of the curtailment of production which is under way, and consumers were not moved to purchase more than for immediate use. The market has been almost stationary at 12.5 cents to 12.75 cents for lake and 12.35 cents to 12.45 cents for electrolytic. Lake copper has been especially stagnant.

Lead.—Quotations remained unchanged at 4.40 cents, New York, and 4.27 cents to 4.32 cents, St. Louis. Transactions have been of the usual volume, but the demand is easily met.

Zinc.—This metal has improved its position materially during the month and the prospect is for a still better market. Quotations advanced during the month and a good volume of business was done, as both producers and consumers appeared to have little surplus stock. New York quotations advanced to 5.55 cents to 5.60 cents, and St. Louis to 5.40 cents to 5.45 cents.

Tin.—The domestic market was affected early in September by speculation in London, and for a brief time the domestic market showed an advance. Consumers were apparently little concerned with the reported London corner, and declined to buy Straits tin at the premium usually asked. This resulted in the use of other grades, with the result that it is

possible other brands will be used more extensively in the future to the disadvantage of Straits metal. The New York quotations for September tin declined to 34⅛ cents.

Other Metals.—Aluminium and antimony markets were rather quiet during the month. Quotations on the former were 22 cents for No. 1 ingots, and on the latter stationary at 7¼ cents to 8¼ cents for various brands. The quicksilver market was somewhat improved, but prices remained unchanged at $46 per flask of 75 lb. in large orders, with advances for smaller lots.

The Iron and Steel Market.

Production of iron and steel has proceeded at a practically uniform rate during September, gaged by pig iron being made at the rate of about 24¼ million tons a year. The high point in production was in February, with a rate of fully 31½ million tons, but this represented an unhealthy spurt, for, while general business conditions were not as good as in 1907, the maximum rate attained in that year had been 28 million tons.

September opened with better buying of finished steel products and better specifying upon old contracts. About the middle of the month rumors became thick that there was extensive shading in a number of products and that a radical reduction was in contemplation by leading interests as a rejoinder. In well-informed circles these reports were recognized at the time as unfounded, for while there was some cutting, there was no more than there had been through the first half of the year, when nominal prices were continually receding. The difference in position, if any, was that in the earlier period cut prices once made soon became the market, whereas a determined stand was being made by leading interests, dating from early August, to maintain plates, shapes and bars at $1.40, Pittsburgh. Attention was given to the rumors to such an extent that a meeting was held in New York September 21 of officials of the United States Steel Corporation and the principal independents, apparently for the purpose of considering whether an open reduction should be made. The meeting developed no disposition to reduce prices and as a result no change occurred.

The present price position is that steel bars are held quite uniformly at $1.40, Pittsburg, against $1.45 on January 1, when the general level was highest; plates and shapes are held by the leading mills at $1.40, Pittsburg, except that occasionally on large structural contracts shapes are shaded, perhaps by several dollars a ton, with shading of plates by a few mills of $1 to $1.50 a ton and other products are selling at open market prices. Taking extreme prices in all cases, it can easily be shown that the general level of finished steel prices has declined at a slower rate since August 1 than from February 1 to August 1, for indeed there has been very little recession in the past two months.

On the whole September has been a better month for business than August, but bookings have nevertheless fallen quite short of shipments and this deficit is becoming serious for the reason that certain lines in which there has been little new business lately, but full production, are reaching the end of the accumulations. This observation applies particularly to rails, fabricated steel and steel cars. The slackness of orders along these lines has not been felt because of the large accumulation of business, but it has now become clear that without a large augmentation of new business a further decrease in production of primary materials will be necessary in the near future.

What is called political uncertainty is generally referred to as the chief drag upon business, but the term appears to be a misnomer for it is difficult to see how the different parties promise substantially different things. Perhaps it will be found to have been more accurate to regard the present as a period of readjustment after too great a swing in the recovery from last year's breaks in prices.

Pig Iron.

A few important sales of pig iron in August gave rise to expectations in some quarters that a general buying movement would be precipitated. These expectations have been disappointed. The market has continued to drag, although the total volume of sales has probably shown a slight increase. A disappointment has been furnished by producers discovering that consumers are better covered for this year's delivery than was assumed. In the case of Southern iron, in particular, a sharp line is drawn at Jan. 1, consumers being very well covered to that date, but having bought practically no iron for delivery beyond it. With Northern iron the same distinction can be made, but the line is hardly drawn as sharply. Roundly speaking, prices of pig iron are about 25 cents lower than at last report. In the valley-Pittsburgh district there are some evidences that a halt has at last been called upon the decline. Bessemer iron, which sold at $15, valley, at the beginning of the month, remains at that figure, and there have been a number of purchases at the price, even for first quarter delivery. Basic has declined to $13.50, valley, a price which is well below cost of production to furnaces buying ore, and the decline to this point seems to have been caused by the existence of large stocks. When the cost line has been reached a spread of $1.50 between basic and Bessemer is unnatural and as Bessemer appears established at $15 for next year basic promises an advance. There are no sellers beyond the present quarter at $13.50, valley, and it is questionable whether even $14 could be done. Foundry iron has continued at $14, valley, forge being $13.25 and malleable $14. Southern iron is not strong at $11, Birmingham.

Steel.

The market for unfinished steel has been disturbed by producers removed from Pittsburgh selling at delivered prices lower than the Pittsburgh basis, plus freight to destination. Occasionally also middlemen shade the regular basis with steel obtained through conversion deals. The Pittsburgh basis, from which there are occasional deviations, is substantially $24.50 for Bessemer billets and $25.50 for open-hearth, with $1 advance for sheet bars. Rods are quotable at $28, Pittsburgh.

Finished Products.

Sheet prices have continued to decline, concessions from the nominal basis reaching $5 a ton on black sheets, $6 on galvanized and $3 on blue annealed, compared with $3, $5 and $1 a month ago. We quote net prices in the list below. Narrow plates have become definitely quotable at $1.35, while $1.40 continues as the usual quotation on wide plates, with occasional shading. Wire products and merchant steel pipe are being rigidly held at former prices, and the demand is good, better than for any other finished steel product. Pressure for tin plate deliveries has been heavy in September, chiefly on account of an exceptionally heavy demand for the corn crop, while buying for winter requirements by the steady consumers has been light, and in order to stimulate the market now that the canning demand is over it is not improbable that a reduction of perhaps 15 cents a box will be made some time in October.

We quote the market as follows, Pittsburgh unless otherwise noted:

Standard steel rails, $28 at mill, except Colorado.

Plates, $1.40 for wide, $1.35 for narrow, tank quality, ¼ in. and heavier.

Shapes, $1.40 for beams and channels, 15 in. and under, zees, and angles 2 x 3 and larger.

Steel bars, $1.40, base; iron bars, $1.45, Pittsburgh; $1.35, Chicago; $1.35 to $1.40, Philadelphia.

Wire nails, $1.70, base; plain wire, $1.50; painted barb wire, $1.70; galvanized barb wire, $2.

Black sheets, 28-gage, $2.15 to $2.20; galvanized, $3.20 to $3.25; blue annealed, 10-gage, $1.60 to $1.65.

Tin plates, $3.60 for 100-lb. cokes.

Institute of Metals.

The autumn meeting of the (British) Institute of Metals, devoted to the metallurgy of non-ferrous metals, was held at Glasgow from Sept. 21 to 23 under the presidency of Sir Gerard Muntz.

The program included excursions to various works of metallurgical interest in Glasgow and vicinity and an all-day trip on the Clyde, visiting the Isle of Arran and the Kyles of Bute.

In the following we give abstracts of two of the papers presented, prepared from advance copies. The balance will be abstracted in our next issue.

Mr. G. Shaw Scott, Caxton House, Westminster, S. W., London, is the secretary of the Institute of Metals.

Metallography in the Brass Foundry—Gunmetal Castings.

The paper by Mr. H. S. Primrose, metallurgist to Messrs. G. & J. Weir, Ltd., of Cathcart, deals with "metallography as an aid to the brass founder." The microscopical investigation of the structure of alloys is largely superseding the method of judging quality by fracture, and is undoubtedly a more certain guide to the foundryman who wishes to escape the repetition of serious failures due to causes other than change of chemical composition.

The usual standard to which most foundries have to work is governed by rigid specification of the tensile strength and elongation of test bars from the castings produced, and while this method affords a valuable check upon the acceptance of faulty workmanship, failure generally results in throwing the onus onto the metal, but most cases of microscopic examination reveal the fact that structural deficiencies are to blame.

The ultimate strength and degree of elongation which a metal possesses depends entirely on the nature of the crystalline arrangement, and this, in turn, depends upon the rate of solidification, as also to a less extent on the rate of cooling after solidification. For each metal or alloy there is a certain size and arrangement of crystal grain which gives maximum strength and durability. If this be exceeded, then there is not sufficient interlocking of the crystals and thus large cleavage planes are formed, but if the crystal grain be too small, the interlocking may be so minute as to be almost valueless.

The author gives a series of microphotographs of gunmetal species taken from test bars which were attached to actual castings and subjected to tension to determine tensile strength and elongation. The composition of the specimens was that of the usual Admiralty gunmetal, viz., 88 per cent copper, 10 tin, and 2 zinc. The castings were made in dry-sand molds and as nearly as possible all under the same condition of cooling.

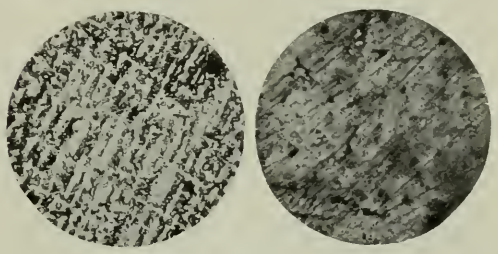
With gunmetal castings the temperature of pouring has a great influence on the strength of the finished castings. In the case of the pouring being at a high temperature, just before freezing commences, the sand of the mold becomes highly heated and so the freezing of the metal is slower than when the metal is not so hot at first, and hence the crystals have time to grow and form a good, strong interlocking structure. This is shown by the microphotograph of a test bar from a casting poured at 1100° C. and slowly cooled (Fig. 1). A similar bar cast at a much lower temperature (950°) was cooled much more quickly and in this instance the crystals not only have been prevented from growing, but the constituents of the metal have remained practically in a state of solid condition with no well-defined structure which is seen to be unresolved (Fig. 2) even with much higher magnification (150 instead of 60 diameters). With the quick, uniform contraction of the solid solution, intercrystalline pores develop and the resulting casting is porous and weak.

This is of great importance for castings which have to stand the test of high water pressure so that the temperature of pouring such castings has to be carefully watched. Further, to get the same quality in all castings, the metal for large and thick castings should not be so highly heated when cast as that which is used for much thinner work. Thus the casting temperature of gunmetal is not a constant, but must necessarily vary with the weight of the casting, as well as its internal and external form, and the number of branches, arms, and projecting parts on the casting.

Now if the founder is careful, and bears this fact in mind, many difficulties can be overcome by arranging his molds of dissimilar castings so that those of a light nature are cast first with hot metal. It is evident that by using a ladle a great variety of casting temperatures can be got, and occasional pauses may be made between some of the molds if the temperature is too high for the nature of the work.

The existence of blow-holes in gunmetal weakens the castings to a very considerable extent. The presence of the blow-holes may be due either to slight dampness in the mold when the metal is cast into it or the gases which form these holes may be actually dissolved in the metal when it is cast. They are often a great source of trouble to the founder, but a microscopical examination is the surest way of ascertaining the cause of blow-holes. If these are clean and confined to the outer portion of the castings they are certain to be due to the steam formed within the mold, in which case a simple remelting brings the metal right.

More often the pinholes in castings appear dirty under the microscope, and on polishing, some brittle material is rubbed



FIGS. 1 AND 2.—MICROPHOTOGRAPHS SHOWING EFFECT OF TEMPERATURE OF POURING.

out, causing serious scratching of the surface. This is due to segregations of included oxides either of tin or copper, which the small percentage of zinc has not been able to remove. Under this condition remelting is of no avail unless with the aid of some deoxidizing agent, such as phosphor tin or phosphor bronze. This need only be added in very small proportions, and which of these is employed will depend on whether the copper or tin is deficient. Their action is due to the phosphide attacking the dissolved oxides, reducing them to metal, and the phosphorus escapes as pentoxide.

It is not often that gunmetal is cast from large furnaces or ladles, but when this is done in large workshop practice special precautions must be taken to ensure the uniformity of the metal from beginning to end of the cast. Segregation does not so readily take place, but with a great weight of metal in the ladle some scum and dirt is apt to be intermixed; the removal of this is somewhat troublesome, even with stirring and the addition of deoxidizers unless a billet of green wood is used to agitate the metal. The stirring is sometimes done by a so-called "secret process," which consists in stirring the metal with an iron rod to the end of which an ordinary potato is affixed.

While the pyrometer is of great service in determining the temperature of the metal before pouring, it is not practicable to take the rate of cooling of each casting. Hence it often occurs that perfectly sound metal gives test bars which fall just short of the required strength, and the reason for this is another point easily explained by the microscope. One of the microphotographs of the author shows that the copper crystals have grown too rapidly in certain directions and thus the

required amount of intertwining has not been effected, with the result that the strength is somewhat low. Another microphotograph shows a rather fine-grained structure almost devoid of interlocking. The author thinks it likely that a suitable annealing would have promoted diffusion and brought the strength of these two castings up to a very much higher standard.

Comparison of microstructures very quickly enables the necessary experience to be acquired for determining the cause of troublesome defects should such occur, and this is more than half the battle in the direction of putting things right again.

Magnetic Alloys from Non-Magnetic Materials.

A paper by Mr. Alexander D. Ross deals with "magnetic alloys formed from non-magnetic materials." The author tested especially the following ternary alloys: Copper-manganese-aluminium; copper-manganese-tin; copper-manganese-antimony; copper-manganese-bismuth, and the following binary alloys: Copper-manganese; copper-aluminium; manganese-antimony; manganese-boron. As to the magnetism of the copper-manganese-aluminium alloys (Heusler alloys), the author adheres to his former view that the magnetic properties are due to the formation of solid solutions, of which the compound Cu^2Al is a constituent.

CORRESPONDENCE.

Defects in Refractory Brick.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—I have read Mr. Bailar's letter on the above subject in your issue of September, 1910.

I entirely endorse what he says about shapes. The natural surface of a pressed brick is one of its best assets. Once that has been chipped or even ground off the brick has deteriorated.

Mr. Bailar announces that he is about to take up arms in defence of the manufacturer, and two paragraphs further on proceeds to hit him over the head with his own brick. I fully agree that when a purchaser does not know what he wants he cannot complain if he doesn't get it. Anyone who orders bricks of a certain chemical composition, regardless of their physical properties, can be added to the foregoing class.

What I am complaining of is this: When the physical and chemical properties of the brick required have been carefully stated and samples submitted showing that such bricks can be manufactured and are not unsubstantial ideals and when the increased cost has been agreed to, that the net result of all the trouble and discussion and investigation on the part of the consumer should be all too frequently one or more carloads of "firebrick" (the quotation marks are taken from Mr. Bailar), showing little or no evidence that the consumer's carefully formulated requirements have been appreciated or that any serious attempt has been made to meet them.

Some day I suppose, the condition of things refractory will be as described in Mr. Bailar's last sentence. But so far as I am aware the time is not yet.

New Jersey Zinc Company, Gilbert Rigg.
Palmerton, Pa.

Electric Steel Making in Spain with Utilization of Slag.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—As your journal has covered in detail the progress of the electrometallurgy of iron and steel in various countries, the following data on developments in Spain should be interesting, especially as little has so far been published about them.

Although the iron and steel industry of Spain is of small importance compared with that of other countries, yet there are several plants in a prosperous condition, particularly in the north of Spain.

Among these establishments one of the best known is the Works of Urigoitia e Hija, its products being well known all over the country for their excellent quality. The works, located at Araya in the province of Alava, are ably managed by the brothers C. and J. de Ajuria, two metallurgical engineers who have kept themselves thoroughly acquainted with all new metallurgical developments.

In the early days of the electric steel furnace they installed at Araya a Kjellin induction furnace, but after various attempts and some weeks of operation, it was practically abandoned. They then decided to install a simple resistance furnace and selected the Louvier-Louis type. Some six months ago a 225-hp furnace of this type was installed at Araya, and since its erection it has worked without interruption and to the complete satisfaction of the owners.

Common scrap iron of all kinds is used as the raw material. To produce in the 160-kw furnace 1210 kg of high-carbon steel (containing 1.03 per cent of carbon) 970 kw-hours are consumed, i.e., 802 kw-hours per metric ton (2200 lb.) of steel produced, starting with cold raw materials.

In the production of 1230 kg of soft steel, the consumption of electrical energy is 795 kw-hours, or 646 kw-hours per metric ton produced.

As these results are obtained in a furnace of small size, it may be estimated that in a 5-ton furnace the electric power consumption would be reduced by, say, 25 per cent.

However, the efforts of Messrs. de Ajuria have not been limited to economic steel production in the electric furnace. They have employed their electric steel furnace also for other purposes. Assisted by the inventors of their steel furnaces, principally by Mr. Louis, they decided to use their Louvier-Louis furnace for the working-up of the slag which had accumulated in their yards from the operation of their blast, reheating and puddling furnaces. Until then this slag had simply been a waste, had occupied valuable space, and obstructed traffic and transportation.

It was now decided to try and utilize this slag for the manufacture of ferrosilicon and of a superior grade of cast iron in the electric furnace; the ferrosilicon to be sold and the cast iron to be used in the works for the casting of special pieces.

Although the Louvier-Louis furnace was designed for steel making, it proved very satisfactory for these new purposes. The results were as follows:

Ferrosilicon.—Two grades of ferrosilicon were made, one containing from 25 per cent to 30 per cent silicon, the other from 50 per cent to 60 per cent silicon.

In every trial a comparison was made between the use of coke and charcoal as reducing agent. The results were in all cases in favor of charcoal.

In the manufacture of 25 per cent to 30 per cent ferrosilicon the raw materials were slag from the reheating furnace and coke or charcoal in dust or small pieces. In the 160-kw furnace the manufacture of 1 metric ton of ferrosilicon required 3790 kw-hours using coke, and 3445 kw-hours, using charcoal.

In the manufacture of 50 per cent to 60 per cent ferrosilicon the raw materials were slag, to which had been added silicious ore from the vicinity of the works, and coke or charcoal. The power consumption per metric ton of ferrosilicon was 8030 kw-hours using coke, and 7105 kw-hours using charcoal.

It will be seen that the use of charcoal represents a considerable economy in electric power consumption. Moreover, the operation of the furnace was better, the chemical reduction easier, and the volatilization of the silicon almost nihil, beside a smaller consumption of electrodes.

Cast Iron.—For the manufacture of cast iron in the 160-kw electric furnace slag from the puddling furnace was employed as raw material. The power consumption per metric ton of cast iron was 1660 kw-hours using coke and 1485 kw-hours using charcoal as reducing agent.

This consumption of 1485 kw-hours per metric ton is equivalent to 0.220 hp-year per metric ton, which is a very low figure for a small-size furnace. For a large-size furnace the consumption would be considerably reduced.

Mexico City, Mexico F. L.

The Passing of Crucible Steel.

By Joseph W. Richards.

A prominent American mechanical engineer on returning from Germany declared that the Germans were 15 years ahead of the United States in some mechanical engineering work. In similar vein, let us freely admit that in the manufacture of fine steel the Germans are at least five years ahead of us, for five years ago the Remscheid plant, about to be described, shut down its crucible steel furnaces and installed in their place the Héroult electric furnace for the manufacture of all varieties of fine steels, up to the finest, while our own Crucible Steel Company of America is only just now waking up to the situation and commencing this year to "do likewise."

The members of the Fifth International Congress for Mining and Metallurgy, held this summer at Düsseldorf, were invited to visit the works of the *Stahlwerke Rich. Lindenberg, Aktien Gesellschaft*, at Remscheid-Hasten, and every opportunity was

FIG. 1.—VIEW OF OLD STEEL WORKS.

given the visitors to make themselves fully familiar with the manufacture of steel as there carried out. The entire absence of secrecy was a most convincing demonstration of the industrial strength which comes from being first in the field, and keeping ahead of competitors not by secrecy, but by sheer force of superior enterprise and workmanship. Nine times

FIG. 2.—VIEW OF THE WORKS IN 1908.

out of ten industrial secrecy is an index of industrial weakness: the really strong have no need to shut themselves up in industrial fortresses, but can act large-mindedly, conscious of their innate strength.

We will give first the general plan and description of the works as a whole, and then follow with a detailed description of the electric furnace work. Some of the information and illustrations are taken from *Deutsche Industrie-Deutsche Kultur*, Vol. VI, No. 7; the larger part is from our own direct observation.

History of the Lindenberg Works.

The firm of Gebr. Lindenberg & Company was founded in 1840-41 by Albert, Richard, and Ewald Lindenberg, and within

FIG. 3.—BOILER HOUSE.

a short time its fine steel, bearing the trade-mark "Glocke," was well known all over Germany and in almost all countries of Europe. At this time the German steel industry was just beginning to free itself from English domination, but for many years the steel bars and ingots were still brought from England. In 1864, however, the firm commenced to make its own crucible steel, and this part of its business was carried on under the name of the Stahlwerke Rich. Lindenberg, Aktien-Gesellschaft.

The erection of the first crucible works, with steam hammers

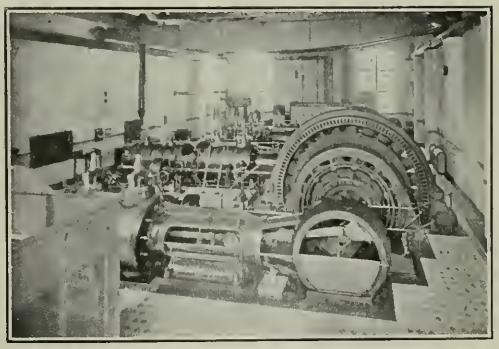


FIG. 4.—ELECTRIC GENERATING PLANT.

and rolls, was a difficult undertaking, for all the materials and machinery had to be hauled over mountain roads from Elberfeld, four miles distant, and it was only in 1868 that the works had railway connection with the rest of Germany. An early specialty of this works was "Damask spring steel" for railway car springs, which attained great popularity, and was used for 30 years for this purpose, in competition with open-hearth steel, in spite of its much higher price.

There was much difficulty, however, in those early years, in inducing German steel workers to give up buying English steel; their prejudices against German-made steel were only slowly and laboriously overcome. In 1896 the old crucible melting holes were replaced by two 24-crucible regenerative gas furnaces. In 1899 Rich. Lindenberg, Jr., became head of the works, which in 1901 he changed to a limited stock company.

In 1904 this firm was the first German works to take up the question of manufacturing its steel electrically, and in December, 1904, concluded to erect two experimental Héroult furnaces of ½ ton and 1½ tons capacity. These, with the neces

FIG. 5.—5-TON HAMMER WORKING A 1-TON ELECTRIC STEEL INGOT.

sary power plant, were erected in 1905, from plans of Director Rich. Eichoff (now professor of metallurgy in the Imperial Mining Academy in Berlin), and the first charge of electric steel was poured on Feb. 17, 1906. Since March 23, 1906, the two electric furnaces have been in uninterrupted service.

The steel thus electrically manufactured was so advantageous in regard to quickness of production, easy work, protection of the workmen, and above all in superior quality of product, particularly for tool steels, that inside of a year the demand had far exceeded the capacity of the plant, and in 1907 a third electric furnace of 3 tons capacity was installed, the power station increased to 1600 hp, a larger roll train put in, and a

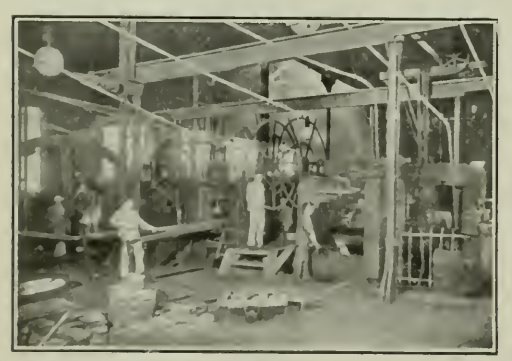


FIG. 6.—ROLLING SAW BLANKS.

special department for making magnets started. The firm was also reorganized to its present constitution by German and Swiss capitalists, with a capital of 2,250,000 marks ($550,000).

General Description of the Works.

The steel-making building is 15 m by 60 m (50 ft. × 200 ft.). At its one end is a tilting open-hearth furnace of the Wellman construction, basic hearth, capacity three tons, served by two gas producers outside the building. Next to it are two Héroult electric furnaces, each with two electrodes, single-phase, run by two transformers in parallel; the smaller is acid hearth, capacity 1.5 tons to 1.8 tons; the larger is 3 tons capacity, basic hearth of burned dolomite. The liquid metal from the open-hearth furnace is transferred to the electric furnaces. The whole installation produces 40 tons to 50 tons per day, 12,000 tons to 15,000 tons per year.

The power requirements were not easily met, since there is no cheap water-power in the neighborhood, no waste blast-furnace gases, and freight on coal from the Rhein district is high. The boiler house (Fig. 3) has six boilers with a total heating surface of 1000 sq. m (35,000 sq. ft.), giving steam at 10 atmospheres (150 lb.) pressure. The power house (Fig. 4) has one 1600-hp, four-cylinder, twin-tandem steam engine, running two dynamos on one shaft, the one a direct-current machine of 570 kw, at 450 volts, for running the roll-train motors, the other a single-phase, alternating-current machine of 450 kw, at 3000

FIG. 7.—ROLLING-DOWN BILLET.

volts, for running the transformers at the 3-ton electric furnace. Another single-tandem engine of 800 hp runs two alternating-current generators of 250 kw and 335 kw, furnishing single-phase current for the other electric furnaces. A 150-hp machine runs the lighting dynamo.

The casting floor is next to the furnace platform, the casting ladle being carried to the molds by an overhead traveling crane.

A 5-ton hammer (Fig. 5) serves to work down the 1-ton ingots, into which the steel is cast, to sizes suitable for the rolls. High-speed tool steel is cast into smaller ingots, which are worked down at a white heat, with frequent annealings, under a 1 ton hammer.

The old rolling train in an adjoining building is used prin-

FIG. 8.—HOT SAW.

cipally for making saw blanks (Fig. 6). Almost all the numerous saw works of Remscheid are supplied from this works. Its capacity is 6 tons of sheet steel in 24 hours, rolled to 0.6 mm up to 4 mm (0.025 in. to 0.16 in.). This train is run by a 350 hp steam engine.

The new rolling mill is in a building 80 m (260 ft.) long by 22 m (70 ft.) wide, with a side bay 8 m (26 ft.) wide. The billets from the steam hammer are reheated and passed through the breaking down rolls (Fig. 7), then sawed hot to proper

lengths (Fig. 8) and finally rolled down to rod in the fine rolls (Fig. 9).

Both rolls are run by electric motors placed in the side-bay and run by the 450-volt direct current. The coarse rolls handle

FIG. 9.—FINE ROLLS.

60 tons in 12 hours, the fine rolls 10 tons to 30 tons, according to the section produced. The side hall also contains roll-turning equipment, and a polishing plant in which the billets are polished before being rolled (Fig. 10).

FIG. 10.—POLISHING THE STEEL BILLETS.

For smaller sections than can be rolled, the rolled bars are worked down as usual under trip-hammers (Fig. 11).

From every charge of high-speed tool steel a sample is taken

FIG. 11.—HAMMERING DOWN THE SMALL SECTIONS.

and tested (Fig. 12) in the lathe, where it must work satisfactorily at 25 m (80 ft.) per minute velocity, 25 mm (1 in.) depth of cut and 18 mm (0.75 in.) feed, on soft cast iron, giving a turning of a certain length before becoming dull.

One of the results of this great care is seen in the "Grand Prix" received by this firm at the recent exhibition in Liege. The firm's exhibit is shown in Fig. 13.

The electric steel has, on account of its purity and uniformity, shown itself highly suitable for permanent magnets. A considerable part of the production of the works is thus used in their own magnet shop. Figs. 14 and 15 show the forging and the machine rooms of the magnet plant. In another room the magnets are magnetized by passing through their two ends an alternating current. They are thus kept in the alternating

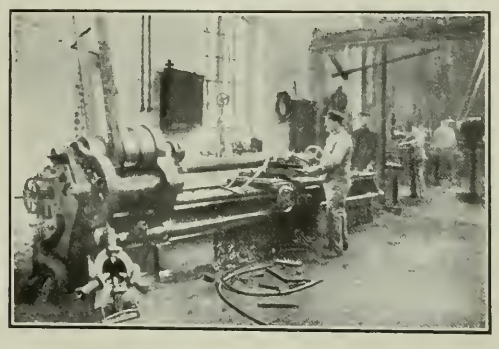


FIG. 12.—TESTING SHOP.

circuit until a maximum of residual magnetism remains in them.

Each magnet is then carefully tested by a specially constructed galvanometer, to plot with scientific accuracy the number of its magnetic lines of force and the strength of its magnetic field. The enthusiastic writer in *Deutsche Industrie* apostrophizes them somewhat thus:

"Cast in the sun-heat of the electric arc; pressed into shape by the electric rolls; given their magnetism by the electric current they are pure children of the electric force and all their

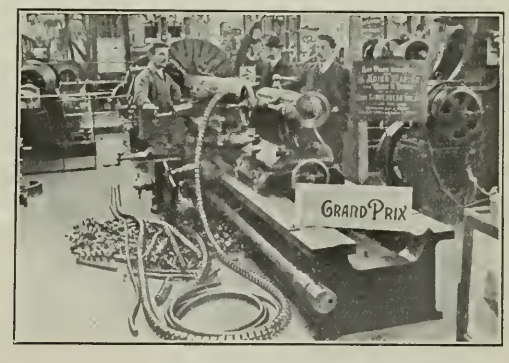


FIG. 13.—EXHIBIT AT LIEGE.

life possess electromagnetic strength. So do they through all the wide, wide world spread broad the fame of Goddess Electra!"

It is almost superfluous to say that such a works as we have just described possesses a fine chemical laboratory (Fig. 16), with microscopic and physical testing apparatus of the most modern kind (Fig. 17).

The Fine Electrical Steel Manufacture.

The first step is the manufacture of the steel in a tilting open-hearth furnace (Figs. 18 and 19). This is an ordinary basic-hearth Wellman furnace. The charges are pig iron, heavy

and light scrap, iron ore and roll scale. The charge is "over oxidized" before being poured, in order to reduce phosphorus and sulphur to the lowest possible point.

"This strong over-oxidation was formerly impossible to practise, because no means were at hand to neutralize its effects. Only the invention of the electric process made possible this high preliminary purification of the molten metal, assuring thereby the highest possible purity of the steel."

The average composition of the metal from the open-hearth furnace, as given during the visit of inspection, was: Carbon, 0.05; Mn, 0.15; P, 0.01; S, 0.05. This analysis would show it not to be the "over-oxidized" metal spoken of by the writer in *Deutsche Industrie*, and also in the Héroult patents.

FIG. 14.—MAGNET FORGING PLANT.

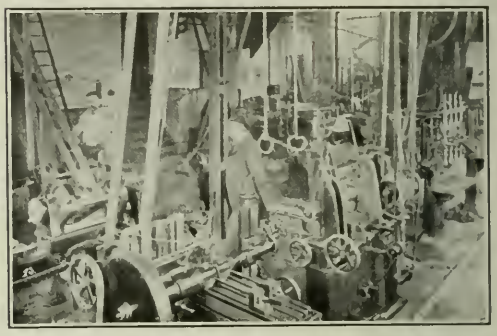


FIG. 15.—MAGNET MACHINING SHOP.

The electric furnace plant consists at present of a 1.8-ton Héroult furnace (Fig. 20) with acid hearth, and a 3-ton furnace with basic hearth, both of the usual Héroult construction. The molten steel is received from the open-hearth furnace in a 5-ton ladle, the ladle and contents weighed, then poured into the electric furnace, and the empty ladle with remaining slag weighed, so that the charge in the furnace is known to a kilogram.

The electrodes are 28 cm square, and fit snugly through the water-cooled sockets in the roof. They wear away 1½ cm to 2 cm (0.6 in. to 0.8 in.) per heat, or 2¼ kg (5 lb.) per ton of steel. The electrodes are automatically kept at 4.5 cm (1.8 in.) from the surface of the slag.

The electric current is supplied from the power station at 3000 volts, and is transformed to 100 volts or 110 volts by two transformers connected in parallel. The power-factor of the furnace varies between 0.90 and 0.93.

The power used by the 1.8-ton furnace is 250 kw, or 385 kw-hours per ton of fluid metal treated; the 3-ton furnace takes 350 kw or as low as 235 kw-hours per ton of steel treated. Of these 350 kw it requires 250 kw to keep the furnace up to a constant temperature, while the other 100 kw (Ca, 30 per cent) is used in melting the fluxes which are added and in raising the temperature of the bath.

While the temperature of the steel immediately under the electrodes rises very high, yet the average temperature of the bath as a whole is not greatly increased. Estimating the temperature of the metal put into the furnace as 1600° C., I should not place the temperature of the metal during casting at more than 1650°. The kw-hours consumed by the steel, allowing for radiation from the furnace, would not admit of more than this 50° C. increase.

The single-phase alternating current, at 25 periods, is charged to the furnace at 6 pfennigs (1.5 cents) per kw-hour; the cost of production (by steam engines) is 4½ to 5 pfennigs (1.12 to 1.25 cents). At this rate the power charge works out to 23.10 marks ($5.77) per ton of steel in the small acid-lined furnace, and 14.10 marks ($3.52) per ton of steel in the large basic-lined furnace.

The voltage drop from the main conductors to the electrode clamps is 1.8 volt, in the holders 1.8 volt, or together about 3.5 per cent of the power used in the furnace.

At the time of the writer's visit 2000 kg of open-hearth metal was put into the furnace at 8.40 a. m. Limestone and lumps of ore were thrown in, and the current put on (Fig. 21). During the next 50 minutes the heating up of the metal took place, and its oxidation by the very fluid iron-lime slag.

At 9:30 some lime was thrown in, to stiffen the slag, the front door was opened, the furnace tipped forward a little, and with a hooked bar resting on the rod before the door the slag was scraped off into the pit (Fig. 22). Fig. 23 gives a nearer view

FIG. 16.—CHEMICAL LABORATORY.

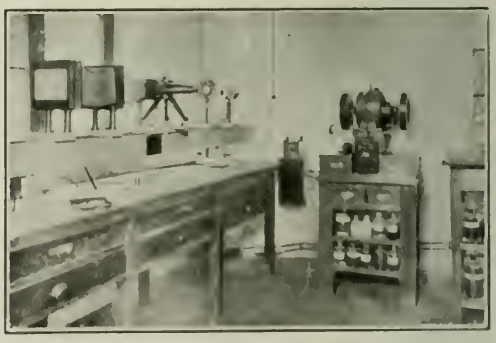


FIG. 17.—MICROPHOTOGRAPHIC LABORATORY.

The dark brown iron-lime slag being removed there were added lime and fluorspar, and as soon as these had formed a liquid slag some finely ground carbon was sprinkled over the surface. This carbon reduced strongly the iron oxide left in the slag. A sample of the slag was taken with a small cold iron rod. Before adding the carbon it was dark brown; as the carbon reduced it, it became lighter brown, until eventually it became almost white and fell to powder in the air at a little below red heat, say, at 300° C. The fineness of the carbon prevented it

FIG. 18.—CHARGING THE OPEN-HEARTH FURNACE.

from sinking through the slag and carburizing the metal. The slag was then rendered non-oxidizing, and the dephosphorization of the bath was completed.

More lime and fluorspar were then added, and from time to time the following additions made: 16 kg of 50 per cent ferro-silicon, 6.75 kg of ferro-manganese, 18.5 kg of hard (carburized) ferro-chrome, 7 kg of ferro-manganese; later a large block of "carburite" (a briquette containing half carbon and half iron) was weighed and put into the furnace, being held under the surface by an iron bar so as to reach through the slag into the steel.

FIG. 19.—CASTING FROM OPEN-HEARTH FURNACE.

A test taken at 9:50 a. m. showed 0.51 per cent carbon, and at 10 a. m. 0.50 per cent. Since 0.65 per cent carbon was desired in the steel, the weighed block of carburite was again put in at 10:20 a. m., kept in about five minutes, taken out and weighed. Since 0.15 per cent of carbon was needed by the bath, equal to 3 kg of carbon, or 6 kg of "carburite," the block of carburite was taken out from time to time and weighed on a small platform scale. When 6 kg had been dissolved from it the block was thrown aside.

The slag was again tested, and proved nearly white, with a tinge of yellow due to chromium, and of the desired disintegrating quality in the air. Some acetylene could be detected when pieces of it were dropped into water, but not much.

The heat was tested by taking a round iron rod, 2 cm (0.8 in.) in diameter, bending the end to a hook, and holding it just inside the front door, near to the bath. As soon as it began to

FIG. 20.—ELECTRIC FURNACE, WITH OPEN-HEARTH FURNACE IN BACKGROUND.

melt, and drops of liquid iron to fall from it, the workman plunged it rapidly about 0.6 m (2 ft.) into the metal, doing this twice in about one second of time, and then withdrew it. The hook was entirely melted off, and the bar melted to a sharp point.

I should judge that the bath was at this time some 50° above the usual open-hearth finishing temperature. A test ingot was poured, which set quiet like lead. A shovelful of fluorspar

FIG. 21.—CURRENT PUT ON THE ELECTRIC FURNACE.

was then thrown in, a red-hot ladle brought before the furnace, and the charge poured at 10:40 a. m.

The final slag is said to average 15 per cent silica, 50 per cent lime, and the rest calcium fluoride, calcium sulphide, calcium carbide, and traces of chromium protoxide and manganous oxide and an iron oxide lower than FeO—possibly Fe^2O.

The steel which was aimed at in this heat was: Carbon, 0.65; Mn, 0.6; Si, 0.4; Cr, 0.5 = 0.6; P, less than 0.01; S, less than 0.01 per cent.

The steel was cast (Fig. 25) into six different sized ingots. It poured and stood absolutely quiet in the molds, like melted lead. To avoid shrinkage a little steel was poured in afterward to fill the shrinkage cavity. The stopper worked accurately, without any splashing or foaming around the nozzle. In all respects it acted like best dead-melted crucible steel.

The banks of the furnace were then made up with burnt dolomite, and at 11:10 a. m. another charge put into the furnace. The actual treatment took two hours; 10 charges are usually worked per day.

As to the reactions in the process, phosphorous and other oxidizable elements are completely eliminated by the first slag; copper and arsenic are not, and should, therefore, not be in the molten metal charged in any harmful quantity. Sulphur is

FIG. 22.—REMOVAL OF SLAG. FURNACE IN BACKGROUND BEING TAPPED.

eliminated by the action of calcium carbide formed in the second slag:

$$CaC^2 + FeS = CaS + FeC^2.$$

In this second stage, silicon, manganese, chromium, nickel, vanadium, molybdenum, boron or tungsten are added to the steel with quantitative accuracy, scarcely a trace of any of these additions passing into the slag and the steel having the exact calculated composition.

Comparing the electric process as here developed with the

FIG. 23.—ELECTRIC STEEL FURNACE.

crucible process here displaced, the management, after five years' experience, affirms with the greatest positiveness that the electric steel is more nearly of the desired composition, more uniform in composition, can be made from far cheaper raw materials, costs far less for the smelting operation, far less for labor, the labor required is much easier and safer, the output per given sized plant is greater, the steel is more free from blow holes, develops less edge-cracks or surface faults, and for a given ductility will carry more carbon and therefore show higher strength.

Americans, wake up!

Zinc Ore Dressing in Colorado

By H. C. Parmelee.

Marion Mines and Mills Company.

An interesting example of the development of economical ore dressing under conditions which, at first sight, might be considered unfavorable is to be found in the operations of the Marion Mines & Mills Company. Given an ample supply of ore, good milling facilities and adequate water power, the company has demonstrated that it will be able to operate successfully despite a 26-mile wagon haul to put concentrates on board cars.

The company's property is located at an altitude of 9600 ft. in the Greenhorn range, at Fairview, Custer County, Col., about 45 miles southwest of Pueblo. The mining property comprises 12 claims, giving 4500 ft. on the main vein and protecting the sites of reservoir, pipeline, mill and town. The mineralized fissure occurs in granite and is composed of schist and altered granite with sulphides of zinc, lead and copper. The ore is found in zones from 300 to 400 ft. in length and of an average width of 15 ft. Where the main tunnel cuts the vein the ore is 25 ft. thick and assays 22 per cent. zinc. The zinc content varies from 12 to 22 per cent.

FIG. 24.—TESTING THE TEMPERATURE.

Power.

The development of cheap power is the key to the situation, and water power has been provided for all mining and milling operations. Located at an altitude of 10,780 ft., or about 1100 ft. above the mine and mill, and distant about 1¼ miles therefrom, is the reservoir and dam. The area of the former is 41.5 acres and its capacity is 21,800,000 cu. ft. The surrounding watershed is heavily wooded and snow and rainfall are well conserved. The water thus provided is sufficient for operating the mill 24 hours per day throughout the year.

The dam is of heavy timber and rock construction with its foundations on bedrock. When completed according to present plans, it will be 360 ft. long between massive natural rock abutments, 50 ft. wide at bedrock and 16 ft. wide at the top, which is 52 ft. above bedrock, or 45 ft. above the earth level at the center of the dam.

The pipe line from reservoir to mill is 6300 ft. long. Beginning at the dam, 10-in. spiral riveted pipe with Root joints extends for 5220 ft., increasing gradually in strength from No. 16 to No. 8 steel. The next 400 ft. of pipe are of 8-in. spiral riveted No. 8 steel and the last 680 ft. to the mill are of 8-in. No. 8 lap welded steel. The pressure gage at the water wheels in the mill shows 1070 ft. of water in the column and a pressure of 465 lb. per square inch. Engineers' estimates of the power developed by this system average about 300 hp. The power now consumed in milling operations is but 65 hp,

and the proposed addition to the mill, mentioned later, will require but 15 hp additional.

The water wheels used are Hug and Pelton high-pressure wheels. For mining operations one 24-in. Hug wheel drives the air compressor and one 15-in. Hug is direct-connected to the hoist. Air is supplied for ventilation by a fan driven by a 12-in. Pelton wheel, and electric lighting for mine, mill and camp is provided by a dynamo driven by a 12-in. Hug wheel. The mill is operated by two wheels, the crusher, rolls, elevators and tube mill being driven by one 24-in. Pelton and the 11 concentrating tables by one 12-in. Hug. The 24-in. Pelton just referred to is one of two on the same shaft so that either may be used for present operations, or both in case increased power is required. The new addition to the mill will have one 18-in. wheel for the tables and one 12-in. wheel for the pump elevating middling. The nozzles used in connection with the wheels mentioned vary in diameter from 3 16 in. to 5/8 in., and the valve gates are only partly opened. Fire protection is provided at several points in the mill, the water for this system being taken from the main pipe line through a reducing valve.

One of the advantageous features of this water power system is the steady transmission of power in the mill. After the water wheels have been adjusted to their proper speed there is no variation in their running, and the result is a remarkably steady operation of every machine.

Milling.

The Marion ore lends itself quite readily to a system of wet dressing, which has been adopted after testing the efficiency of several other methods. The mineral particles in the ore are small and quite finely disseminated throughout the gangue, necessitating fine grinding for their liberation from the gangue and for the separation of the different minerals from each other.

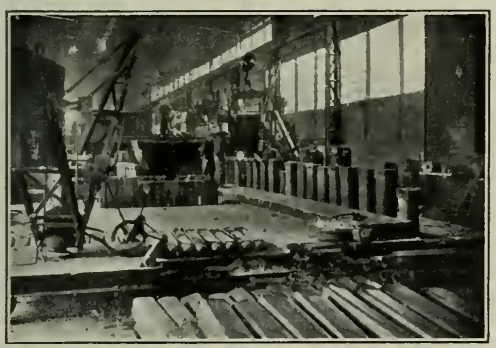


FIG. 25.—CASTING THE STEEL.

After considerable experimenting an adequate system of ore dressing has been devised which, while it may be subject to minor changes and improvements, has been adopted essentially as given below. One of the problems encountered in treating this ore, viz., the necessity for fine grinding to liberate the mineral particles, has been met with an innovation in the use of the tube mill for grinding a large part of the table feed to 40 mesh. The tube mill has been a successful adjunct to metallurgy where very fine grinding or sliming is a preliminary operation, and it has also been introduced for the regrinding of middlings prior to retreatment on concentrating tables, but its use on the first break of the ore prior to any classification or dressing is believed to be novel. Its successful application in this instance may widen the possibility of its use in ore dressing.

The flow of the ore through the mill and the details of the operation of some of the machines are given below. The mill is located at the mouth of the tunnel through which the ore is trammed a distance of about 200 ft. Coarse crushing is done with a No. 4 Samson 8 x 16 crusher, reducing the ore to 1-in. size. This product passes to 250-ton bins whence it is delivered by an automatic feeder to a set of 14 x 27 McFarlane rolls, discharging to a 14-in. B. and B. elevator which raises the pulp to the trommel screen line. The trommel screens vary from 5-mesh to 40-mesh, the undersize from the latter being a finished product, and all intermediate sizes being returned for regrinding. Trommel No. 1 has a 5-mesh screen and discharges its oversize to a second set of McFarlane rolls and its undersize to trommel No. 2. This is fitted with 8-mesh screen; the oversize is returned to a third set of rolls and the undersize to trommel No. 3. This last trommel has 40-mesh screen, the undersize being ready for table treatment and the oversize passing to the tube mill for further reduction. The tube mill is Denver Engineering Works type 10 ft. x 4 ft., revolving at the rate of 28 to 30 r.p.m. All rolls discharge to the same elevator which supplies the trommels. The tube mill product, however, is not screened again, but is combined with the undersize of the 40-mesh trommel screen and delivered to a 6-ft. Callow thickening cone preparatory to classification.

The pulp is classified for table treatment by a Richards 4-in. six-compartment pulsator classifier making 320 pulsations per minute. The first two compartments discharge to two 4-ft. Callow thickening cones (which also receive the overflow from the first Callow cone), where the fine grades are again thickened before going to the tables. The third, fourth, fifth and sixth compartments are discharged directly to the tables. Card tables are used throughout the mill and give excellent satisfaction. The overflow from the second and third Callow cones passes to two more cones which thicken the feed for one table treating the finest grade.

All concentrates are discharged automatically to filter bins beneath the table floor, where they are dewatered for shipment. These bins are of the usual double floor and wall construction with cocoa matting in the floor and burlap in the walls.

Data.

The table products are (1) lead concentrate, (2) zinc concentrate, (3) zinc middling, (4) tailing and (5) slime. The first is very small in quantity, but carries the major percentage of the gold and silver in the ore. The zinc concentrate averages a little better than 40 per cent zinc. The middling assays about 15 per cent zinc, and it is for the retreatment of this product that a new addition will be made to the mill. Experiment has shown that the middling can be concentrated on tables without futher crushing, and it has been found advisable first to make as high grade and clean zinc concentrate as possible and then retreat the comparatively rich middling on separate tables. Throughout the mill the idea has prevailed that each machine should be allowed to make the best possible finished product and that unfinished products should not be returned to the same machine but rather to another which can rectify the errors of the first. The slime waste offers a problem which has not yet been solved. This material consists mainly of fine scales of mica which concentrate in the overflow of the Callow tanks. It carries about 15 per cent zinc and is at present collected in settling boxes.

Prior to the introduction of the tube mill the rolls used for fine crushing were unable to reduce the ore fine enough to liberate the mineral particles, and large quantities of screen oversize accumulated so that occasionally it was necessary to shut off the mill feed until the oversize could be reduced by repeatedly passing it through the rolls.

The performance of the tube mill has been satisfactory in that it handles 8-mesh material and discharges a granular product without an excess of slime. The mill has a silex lining and carries a load of pebbles equal to about two-thirds the load used for very fine grinding. The condition of the product is regulated from day to day largely by the quantity of water supplied. Capacity tests indicate the possibility of handling

75 tons per 24 hours in this tube, and it is now crushing about 40 per cent of the mill feed, 60 per cent being handled by the rolls. The following screen analysis shows the work of the mill on an 8-mesh feed.

	Per cent
On 30 mesh	1.28
40 "	4.49
50 "	12.18
66 "	17.30
86 "	17.95
116 "	16.67
172 "	14.10
200 "	4.49
Through 200 "	11.54

This shows about 65 per cent of the material between 40 mesh and 116 mesh, with the production of but 11 per cent slime. Various other tests indicate that about 10 per cent slime can be maintained in steady operation.

The Richards pulsator classifier is doing effective work on this ore and is superior to the device employed prior to its introduction. The following screen analyses show its work:

First Compartment. Slime, 18.6 per cent Zn.

	Per cent.
On 66 mesh	2.92
97 "	12.13
200 "	48.54
Through 200 "	36.41

Second Compartment. Fine, 24.4 per cent Zn.

	Per cent.
On 50 mesh	2.3
97 "	34.1
200 "	45.4
Through 200 "	18.2

Third Compartment. Medium Fine, 15 per cent Zn.

	Per cent.
On 50 mesh	2.13
86 "	44.49
116 "	22.24
200 "	22.24
Through 200 "	8.89

Fourth Compartment Medium Fine, 13.4 per cent Zn.

	Per cent.
On 40 mesh	2.85
66 "	25.72
86 "	20.00
116 "	22.85
172 "	11.42
Through 172 "	17.16

Fifth Compartment Medium Coarse, 17.1 per cent Zn.

	Per cent
On 40 mesh	2.00
58 "	25.00
86 "	31.25
116 "	22.91
172 "	6.25
Through 172 "	6.25

Sixth Compartment Coarse, 13.4 per cent Zn.

	Per cent
On 40 mesh	3.33
50 "	33.33
74 "	30.00
116 "	20.00
Through 116 "	13.33

The cost of mining and milling is given by the management at about $1 per ton, divided about equally between the two operations. Transportation costs $5 per ton for the wagon haul and $1 or $2 per ton for freight to the Pueblo or Kansas markets respectively. This makes the total cost of mining, milling and transportation $7 or $8 per ton according to the market supplied. As transportation is the largest item of the total cost, special attention is given in the ore dressing to the quality of the concentrate produced, rather than to the quantity, and refinements will be introduced whenever possible to reduce losses in the by-products.

Acknowledgment is made of the courtesy of Manager Henry J. Gray and Consulting Metallurgist J. M. McClave, who have supplied the data herewith given.

The Influence of Impurities on the Properties of Copper.

By F. Johnson, M.Sc.

So profound are the effects of traces of impurities upon the physical and mechanical properties of copper and so mysterious the diversity of those effects, that the writer deems it unnecessary to apologize for publishing the following notes, based on the work of others and amplified by his own observations. He does so, therefore, in the hope that they may contribute toward the stimulation of the desire of metallurgists to have supplied the present lack of knowledge in this important subject.

Arsenic hardens, toughens, increases tensile strength and ductility when present up to at least 1.5 per cent. It does not impair the forging properties when added to pure copper, even beyond 0.5 per cent, except that the forging temperature may not be quite so high as that for pure copper; it invariably *improves* the forging properties where added to impure copper. It is now largely specified for in material—such as locomotive and boiler tubes, fireboxes and firebox stays, steam pipes, etc.—which is required to withstand high temperatures, since mechanically hardened arsenical copper is not softened at so low an annealing temperature as pure electrolytic copper, which has undergone identical treatment. The strength and toughness, too, of arsenical copper is greater at high temperatures than are those of pure, unadulterated copper or those of most brands of impure copper without arsenic. Arsenic enters into solid solution with the copper which, in the cast state, is not perfectly homogeneous, but attains in the forged and annealed state, crystalline uniformity of composition as a result of diffusion and recrystallization. Copper containing arsenic and oxygen is stronger, "ceteris paribus" than copper containing arsenic and no oxygen (deoxidized).

Micrograph No. 1 shows the structure of cast arsenical copper containing 0.5 per cent arsenic. "Cores" of arsenically contaminated copper merge gradually into, and are completely involved by, a network of a solid solution of copper richer in arsenic, paler and stronger than the "cores" of primary solidification.

Micrograph No. 2 shows the structure of a piece of the same copper which has been forged hot and annealed.

Arsenic lowers the electrical conductivity of copper in a most pronounced and startling way. Thorns & Lamb (J. S. C. I., May, 1906) offer a theory that arsenic improves copper by reason of its deoxidizing effect on cuprous oxide. This is erroneous, as a furnace-charge will contain an excessive quantity of oxygen side by side with arsenic. The oxygen may be removed by poling, leaving the proportion of arsenic unaltered.

Arsenic reduces the solvent power of molten copper for reducing-gases and so renders it less susceptible to going over the pitch by excessive "poling."

Mr. Tomlinson (see discussion page 47, "The Treatment of Copper for Steam Pipes," J. T. Milton, Inst. Marine Engrs, 1908-1909) has supplied the following data regarding the superiority of tough arsenical copper over pure copper:

(1) Samples of each, which had been hardened by rolling, were kept at a temperature approaching 400° Fahr. for three or four weeks. This treatment did not affect the arsenical copper, as shown by tensile tests, but the pure copper samples were practically annealed.

(2) Similar samples were heated to 1600° Fahr. (870° C.) (approaching bright red) for four hours. The sample of arsenical copper lost only 6 per cent of its tensile strength, while the sample of pure copper lost 44 per cent.

The percentage of arsenic in the arsenical samples was 0.36

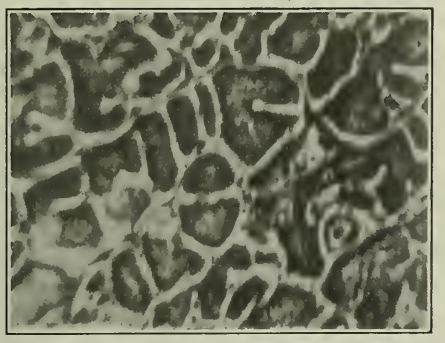


FIG. 1.—COPPER CONTAINING 0.5 PER CENT ARSENIC AS CAST.

per cent, so that copper containing arsenic does not become annealed at the temperature of steam at the highest working pressures, whereas pure copper does. Also, prolonged heating does not so seriously lower its strength as it does that of pure copper.

Arsenic undoubtedly improves the hot-working properties of copper vitiated by traces of bismuth.

Lead.—This element is to be found in all ordinary commercial copper, but is not desirable in any proportion over 0.10

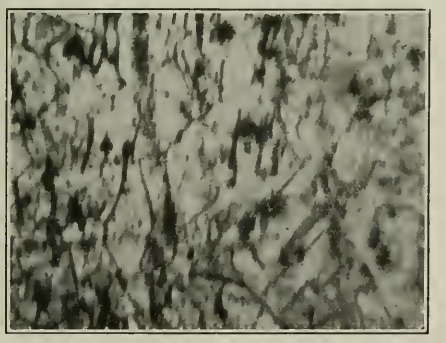


FIG. 2.—SAME AS FIG. 1 AFTER FORGING HOT.

per cent, and the lower the proportion under 0.10 per cent the better.

The lead reduces the strength, ductility and toughness of copper even in the presence of arsenic, the reason being that copper has very little solvent action for lead, the latter merely honeycombing the structure with more or less uniformly distributed prills of lead or prills of lead containing traces of copper.

Comparisons of tests carried out on samples of copper sheet and tube by Mr. J. M. Allen and described before the N. E. C. I. of Engineers and Shipbuilders, January, 1909, go to prove this statement.

CHEMICAL ANALYSES.

	Sheet per cent.	Tube per cent.
Copper	99.040	99.360
Lead	.407	.160
Arsenic	.410	.304
Oxygen	.042	.050
Iron	.035	.041
Nickel	.050	.047

	Tensile Strength. Tons	Elongation. % on 3".	Toughness. No. of times bent to and fro.
Sheet, untreated	14.80	42.0	36
Tube, "	15.37	41.7	61.5
Sheet, heated 6 times to bright red and quenched	14.22	40.5	49.5
Tube treated in same way	15.0	45.0	74.5

Lead existing free in copper, and having so low a melting point, will render the copper more susceptible to *"burning"* during any reheating or annealing process, and also affect its malleability at a red heat.

The addition of lead to copper has the effect of lowering the affinity of copper for reducing-gases.

Nickel alloys with copper in all proportions, forming a complete series of homogeneous solid solutions. Traces of nickel in copper are beneficial, imparting strength, toughness

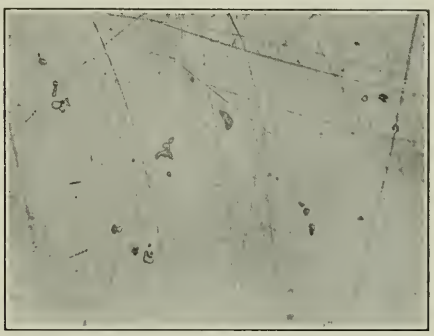


FIG. 3.—COPPER CONTAINING 1 PER CENT ARSENIC. VERY SLOWLY COOLED.

and increased resistance to deformation at high temperatures. Nickel must be kept low where arsenic is present, 0.10 per cent being quite sufficient to harden arsenical copper which has to withstand severe working. The stronger affinity which nickel has for arsenic lends color to the belief that an arsenide of nickel is formed which has a much lower degree of solubility in copper than either nickel or arsenic alone, and seriously hardens the metal. Below 0.05 per cent, even in the presence of arsenic, its effect upon the physical properties, excepting electrical conductivity, may be considered almost nugatory. Copper containing traces of nickel is no less susceptible of dissolving reducing-gases than pure copper. Webb's experiments on locomotive tubes showed that copper tubes containing 3 per cent nickel were first in order of merit, while Johnson found that copper stays containing 3 per cent nickel and 1.6 per cent iron showed admirable results when tested at high temperatures.

Micrograph No. 3 shows the structure of pure copper containing 1 per cent arsenic. The alloy had been very slowly cooled, the endeavor being made to render the *process of solidification* as protracted as possible. The usual "core" structure of cast arsenic-copper alloys is much more subdued, and, in fact, is barely discernible. Small isolated areas of Cu_3As are discernible. Micrograph No. 4 shows the structure of an alloy treated exactly similarly to the above, but containing 1 per cent nickel in addition to 1 per cent arsenic. No new

compound or constituent appears, in fact, even the Cu_3As areas are fewer, but the "cores" are very much more pronounced, indicating greater crystalline hardness.

Bismuth has by far the most deleterious action of all the impurities found in copper. One hundredth per cent renders commercial copper noticeably less malleable at a red heat, but (and here the writer must reassert the value of arsenic in suppressing the ill-effects of bismuth), if arsenic is present, as it should be in all wrought copper not required for electrical purposes, the allowable percentage of bismuth may be considerably higher. By regulating the proportions of arsenic and oxygen in pure copper the writer has obtained with the addition of 0.1 per cent bismuth very respectable tensile results.

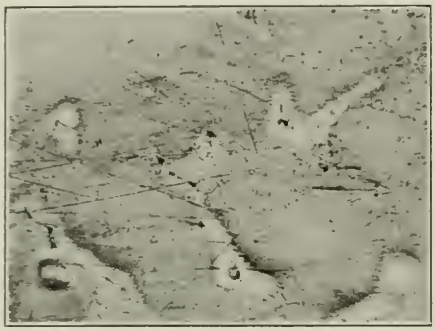


FIG. 4.—COPPER CONTAINING 1 PER CENT ARSENIC AND 1 PER CENT NICKEL. VERY SLOWLY COOLED.

The metal forges well at a red heat (not bright red), but is inferior in this respect to pure copper or to good tough arsenical copper. The writer finds, however, that the forging properties of copper, originally inferior in this respect in the cast state, are considerably improved by hot-forging. We must here assume that the combined effects of distortion of crystals and their simultaneous recrystallization which the process of hot-forging embraces must also create, in addition to a refinement of structure, greater inter-crystalline tenacity, and, perhaps, the literal incorporation, as it were, into the architecture of each crystal of those external envelopes of less pure copper, which, in the cast state, *involved* each crystal and lowered the malleability of the metal as a whole. (This refers to copper containing any impurities which are commonly associated with it.)

Continuing this reasoning, it is safe to assume that the impure envelopes would have a lower freezing point than the purer centers, and that their malleability would represent the malleability of the copper. But intermolecular diffusion and consequent rectification of crystalline composition which reheating and hot-forging have produced render the metals less susceptible to the incipient disintegration, which develops—in the cast and unstable state—when it receives the first pressure of the hammer or of the rolls

Bismuth does not enter into solid solution with copper, but finds its way, in the metallic form, toward the boundaries of the crystals and there solidifies, long subsequent to the solidification of the copper crystals (See cooling curves of Cu-Bi Alloys, Plate 32, 2d report, Alloys Research Committee, Inst. Mech Engrs., 1893.) Hence the strength of the copper is practically represented by the strength of this brittle membranous material But if oxygen be present, the segregation of bismuth is, if not prevented, markedly retarded, and it is the belief of the writer that cuprous oxide assumes a physical relationship toward bismuth, and, when rejected to the boundaries by the solidifying crystals, it shepherds it into areas of eutectic form, where minute granules of cuprous oxide occur uniformly in a matrix of solid copper constituting a solid eutectic necessarily stronger and less deprivatory of inter-crystalline adhesion than brittle plates and films of a metal of low freezing point, such as is shown in micrograph No. 5. The bismuth is trapped in the eutectic in the form probably of an oxide of bismuth.

Returning once more to the theory of the benefits of hot-forging, we may argue that such a constituent of copper as the solid eutectic just described will be more harmful when existing isolated in large angular areas as in the *cast* material than in the more uniformly disseminated form consequent on forging, the dangerous coalescence having been broken up to the production of greater homogeneity by distribution.

Summing up the effects of hot-forging:

(1) The structure is refined and strength increased thereby.

(2) Copper-cuprous-oxide eutectic areas are broken up and uniformly distributed.

(3) Crystalline composition is rectified by diffusion of impurities from the envelopes of brittle material into the softer primary crystals or "cores" (as in the case of arsenic).

(4) Porosity, due to occluded gases, is greatly removed.

The writer feels bound to think that all theories as to the formation of bismuthide or bismuthate of copper are untenable. If bismuth existed free as metallic bismuth in copper containing oxygen, then there should be an arrest in the cooling curve, after the solidification of the copper-cuprous-oxide eutectic. If such be the case, and if that arrest were capable of detection, then the beneficial effect of oxygen can only be physical in coercing what would otherwise be separating films between the crystals into less harmful prills or globules.

It may be that cuprous oxide has a solvent action for bismuth which copper does not possess and that bismuth, therefore, is less harmful in the form of a solid solution in cuprous oxide than as free metallic bismuth. Or, finally, the cuprous

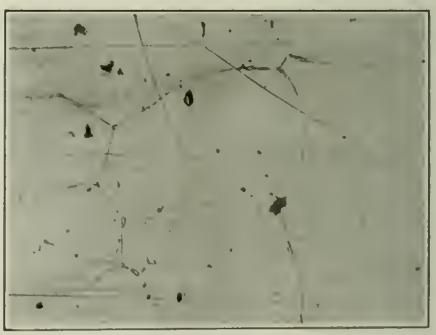


FIG. 5.—COPPER CONTAINING 0.5 PER CENT BISMUTH

oxide granules of the eutectic may be partly replaced by bismuth oxide granules, bismuth taking up oxygen at the expense of cuprous oxide, thus throwing some copper out of combination. This is the view most favored by the writer

Cobalt—Little is known about the influence of this metal. Personal experiments confirm the writer's belief that it toughens and strengthens copper when present up to at least 1 per cent without impairing its hot-working properties. Probably it acts very similarly to nickel, conferring greater durability at high temperatures and raising the annealing temperature, while toughening, hardening and strengthening the copper in the cold. Cobalt, in the low percentages in which it is found in copper, is, if anything, beneficial, and moreover, it does not disagree with arsenic to so great an extent as nickel does.

Antimony hardens and strengthens copper and is not detrimental when present up to 0.5 per cent if other impurities are absent and a small percentage of oxygen is present

Johnson (Birmingham Metallurgical Society, 1906) records that copper containing 0.5 per cent antimony had, after hot and cold working and annealing, a tensile strength of 15.6 tons per square inch, and an elongation of 45 per cent (on 2").

Hiorns & Lamb (J.S.C.I., May, 1909) state that antimony does not lower the electrical conductivity of copper so profoundly as arsenic, that it has not such a hardening effect as measured by the scleroscope, that it is not so efficacious in decreasing the porosity of cast copper, and that it impairs the cold malleability to a greater extent than arsenic. They failed to draw into wire copper containing more than 0.6 per cent

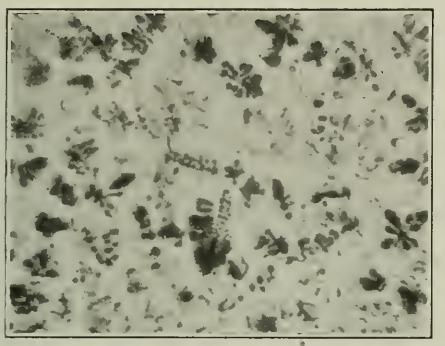


FIG. 6.—COPPER CONTAINING 16 PER CENT CUPROUS OXIDE.

antimony. The writer finds that by adding oxygen it is possible to draw into wire copper containing 3 per cent antimony.

Sir H. Vivian stated that 1/1000 part of antimony (0.10 per cent) will convert best selected copper into the worst conceivable. It must be remembered, however, that it is when the copper is used for alloying that the effect of antimony is so marked. In wrought copper, antimony is not so detrimental.

Tellurium.—Very little is known of the effect of tellurium in commercial copper. A writer in the Journal of the Society of Chemical Industry (1907, p. 695) remarks that 0.05 per cent tellurium is discernible in copper under the microscope. "Wallaroo" copper and some Japanese brands are said to contain traces of tellurium. Cloud (Inst. M. & M. Journal, 1906-7, p. 67) gives an analysis of "Wallaroo" copper showing 0.003 per cent tellurium. Prof. Huntington thinks that the minute traces of tellurium, if alone, are not so harmful by themselves as when they occur together with similar traces of bismuth. He thinks that the single effect of each in mere traces may be insignificant, but that their occurrence together signifies the formation of a telluride of bismuth whose ill effects are very much more pronounced. Prof. Roberts-Austen (2d Report) found that copper containing 0.2 per cent tellurium had a tensile strength of 10 tons per square inch, and an elongation of 13.6 per cent, compared with pure copper, which, having undergone the same treatment, had a tensile strength of 13.2 tons per square inch, and an elongation of 27 per cent —a loss of 24 per cent tensile strength and 50 per cent elongation.

Iron occurs in refined copper in the merest traces, which are quite harmless. The average of eighteen analyses of firebox plates given by Mr. Dean in the second report to the Alloys Research Committee shows only 0.006 per cent iron. When all the sulphur has gone from a charge of copper, a sample taken from the furnace shows an unblistered surface, and is said to be "set-copper." At this point all the iron has been eliminated save that trace which is destined to remain in the copper to the bitter end.

Iron acts as a deoxidizer when added to copper. Copper containing only 1 per cent iron is rendered feebly magnetic, will forge well at a red heat, is quite malleable, tough and strong, even in the presence of arsenic. It lacks the fluidity of pure copper when poured at the same temperature.

Copper deoxidized by iron exhibits a closer surface when cast in an open mold; the characteristic waves as seen on the surface of a "tough-pitch" bar are replaced by waves more minute, more numerous and more regularly arranged. Even 0.05 per cent iron causes a central longitudinal furrow in a cast bar.

Tin hardens copper more than any other element. It occurs very rarely in commercial copper, being readily eliminated during the process of smelting. Low percentages of tin improve the tensile strength, ductility and resistance to corrosion, and maintain these improvements at high temperatures, but the natural softness of copper and its red color are both materially removed, its malleability also decreasing.

Tin rarely occurs in smelted and refined copper in quantities over 0.10 per cent.

Locomotive tubes containing 1 per cent tin, and made from castings deoxidized with phosphorus, came seventh in order of merit in a practical trial made on ten makes of tube in a L. & N. W. Ry. engine, 1898-1901 (Webb. *Proc.* I. C. E., CLV, 1903-4).

Silver is rarely present in sufficiently high percentage to impair either the hot or cold working properties of copper. It is more injurious on the hot-working properties of copper when arsenic is not present. Five one-hundredths per cent silver in pure copper does not affect its malleability at a red heat, while 0.2 per cent can exist in copper containing 0.3 per cent arsenic with quite beneficial effects on the tensile strength and no diminution of its malleability.

Sulphur.—Arnold* has shown that under the artificial conditions of slow-cooling and entire exclusion of oxygen

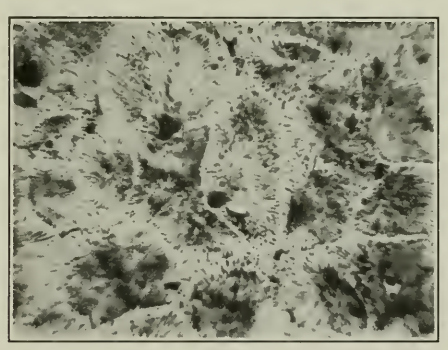


FIG. 7.—COPPER CONTAINING 3.2 PER CENT CUPROUS OXIDE AND CONSISTING WHOLLY OF THE COPPER-CUPROUS OXIDE EUTECTIC.

0.2 per cent sulphur in copper appears as attenuated sulphide walls, and as green globules in the interior of the crystals, or sometimes near the edges in groups of elongated leaves. Hampe (Zeitschrift für Berg-Hütten und Salinen-Wesen im Preussischen Staate, 1873, Vol. XXI, p. 274) found analytically that cuprous oxide, cuprous sulphide and sulphur dioxide could exist together in "tough-pitch" copper, and Heyn (Metallurgie, Vol. 3, p. 73, 1906) showed that cuprous sulphide, like the oxide, was of a bluish color until etched in hydrofluoric acid, when the oxide was stained black and the sulphide retained its blue color. Sulphur rarely occurs in more than harmless traces in commercial copper, yet it may occur to the extent of 0.03 per cent (SO_2) in electrolytic copper having a conductivity of 101.2 per cent. The presence of this element may possibly account for the unsatisfactory nature of copper used in jewelers' alloys. It derives its origin from two sources, viz., (1) the incomplete removal of sulphur from

*Engineering, 1896.

the sulphate liquor in which the copper cathodes were deposited, and (2) the sulphurous gases from the fuel of the reverberatory refining furnace being partially dissolved by the bath of molten metal, as they pass over it on their way to the flue. In the first case the sulphate would probably be reduced to sulphide by the reducing action of "poling," and in the second case, sulphur dioxide would be absorbed and retained as such. Suffice it to say that cuprous sulphide is a highly dangerous, brittle constituent, and its presence in any alloy would be very detrimental.

Lewis (Engineering, Dec. 4th, 1903) states that manganese and aluminium when present in quantity of 0.5 per cent are capable of neutralizing the effect of 0.1 per cent sulphur. His explanation is that manganese by dissolving in the copper crystals gives them the property—which pure copper does not possess—of holding copper sulphide in solution. This is probably erroneous, the effect possibly being analogous to that of manganese in steel, MnS being formed owing to sulphur having a greater affinity for manganese.

Oxygen.—All furnace-refined copper contains some oxygen. The amount will depend upon the quantity and nature of impurities present, and upon the skill of the refiner in producing copper at the highest possible stage of "tough-pitch." That is to say, it is possible to produce ingots, wire-bars, etc., which shall roll or forge, and withstand the severest deformation in cold-working or cold mechanical tests, but, it is still possible, by reducing or increasing the oxygen content, as the case may be, to obtain that delicate balance of oxygen on the one side and impurities (including dissolved gases) on the other, where strength, ductility, malleability and toughness reach their relative maximum values.

It may at once be said that oxygen is indisputably essential in all commercial impure wrought copper which has to withstand repetitions of small stresses, exposure to atmosphere, influence of corrosive agents, and varying thermal conditions. Steam pipes made of pure *electro-deposited* copper have been known to disintegrate after two or three years' exposure to the atmosphere, and even after two or three months, if under steam-pressure. The reason is their susceptibility to annealing or softening at such low temperatures as that of ordinary steam-pressure. If pure electrolytic copper be melted down in a refining furnace and brought up to pitch, i.e. brought to the stage where it contains about 0.05 per cent oxygen, and cast into billets or bars it will, after going through the various stages of manufacture, give in its finished state excellent results, both as regards accommodating itself to stresses and withstanding high temperatures. With the addition of arsenic these properties are still further enhanced.

Electro-deposited copper may be said to be structureless, and its ductility when annealed is possibly not to be excelled by any other form of pure copper; Le Chatelier has shown that at 400° C pure copper is singularly liable to disruption. Its very ductility is a source of danger, for even the deformation produced by successive expansions and contractions will destroy it. What has already been written regarding "slip-bands" may apply here, for it is readily conceivable that the "slip-bands" induced by stresses will be the more easily propagated and developed through a mass of metal where there are no arresting planes such as occur in a more definitely crystalline metal of homogeneous structure. (See micrograph No. 9 of "tough-pitch" electrolytic copper.)

So that, possibly, the influence of oxygen is indirect, in that it is only necessary to ensure the production of sound castings possessing a crystalline as distinct from an amorphous structure, and a sound as distinct from the porous condition which cast copper assumes when the oxygen has been removed and replaced by reducing gases such as hydrogen and carbon monoxide, soluble in the copper while molten, but expelled during solidification when the copper has been poured and allowed to set where atmospheric air has free access to it.

Concentration of stresses on one spot, accompanied by local softening (due to the fact that electro-deposited copper anneals at the low temperature of 200° C) may account for the inevitable failure of such material. Also, it may be that impure copper having its annealing temperature so much higher never undergoes this local softening and all stresses are taken up uniformly throughout the length of a steam pipe (The Treatment of Copper for Steam Pipes," J. T. Milton, Inst. Marine Engrs., 1908-9, Vol. XX), but this theory of the failure of electro-deposited copper fails to explain why such material fails after being in use at only normal temperatures.

M. Le Chatelier has shown that electrolytic copper, worked and annealed, has at 450° C less than half the strength which it possesses at normal temperatures. (See Plate 31, 2d Report, Alloys Research.)

Electrically deposited copper is less durable than commercial copper containing oxygen, when submitted to the corrosive action of sea-water.

By wrapping copper wire in iron wire or ribbon this corrosion is prevented, an action taking place in the formation of a protective deposit of iron oxide.

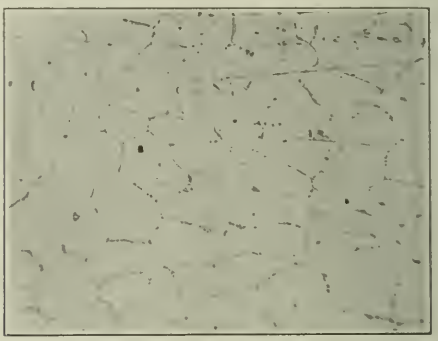


FIG. 8.—"TOUGH-PITCH" COPPER CONTAINING 0.08 PER CENT OXYGEN (0.7 PER CENT CUPROUS OXIDE).

The following micrographs illustrate how oxygen occurs in copper (in the form of cuprous oxide) from the stage of "set-copper," when poling is commenced, up to the stage of "tough-pitch." Heyn (Metallographist, Vol. VI. 1903, pp. 48 to 64) has shown that copper and Cu_2O are completely miscible when fused, but are mutually insoluble in the solid state. The accompanying micrographs show how the cuprous oxide in excess of that required by the copper to form an eutectic separates out first, in cooling, in the form of dendrites (of a blue color as viewed under the microscope). (See micrograph No. 6 showing Cu_2O dendrites surrounded by eutectic.) The eutectic point as seen in the curve given by Heyn corresponds with the presence of 3.4 per cent Cu_2O or 0.30 per cent oxygen. At this point the copper is completely saturated with Cu_2O, and, as seen under the microscope, is a uniform solid eutectic containing neither excess copper, nor excess cuprous oxide. (See micrograph No. 7.) Below this point copper crystals are the first to solidify and are surrounded by eutectic of later solidification. (Micrograph No. 8 of "tough-pitch" copper shows this clearly.)

Mr. Archbutt (Analyst, Vol. XXX, No. 357, December, 1905, p. 392) states that "the effect of the presence of oxygen in ordinary refined copper was to 'neutralize the injurious effect of certain other impurities, and a certain amount of oxygen was necessary in order to make the copper tough. A little more than was actually needed for that purpose would do no harm, and a difference of 0.05 per cent in the region of 0.15 per cent would be of no moment."

This may be true for fairly pure copper, but that there is such a wide range admissible in all kinds of commercial copper, varying as they do in the nature and properties of impuri-

ties, the writer feels bound to dispute. Some brands of copper exhibit infinitely greater sensitiveness in their susceptibility to suffer a loss of their best mechanical properties by a slight addition, or deprivation, of oxygen than others. It is all a question of what impurity or impurities the copper contains and in what proportion.

Segregation of Impurities.—There is no analogy, so far as the writer is aware, of any such segregation of impurities as occurs in steel and iron. The solvent action of molten copper is remarkable and absence of segregation would seem to strongly confirm a perpetuation of this property through a falling temperature. The rapid setting of copper castings also tends to limit any tendency for impurities to segregate. The precious metals, however, do segregate to a marked degree, quite sufficient to warrant very careful sampling (Heath, "Methods for the Complete Analysis of Refined Copper," J. Amer. Chem. Soc., Vol. 27, 1905).

To sum up, it may easily be seen from what has been said concerning the effect of impurities on copper that to systematically investigate their effect singly in measured quantities on copper containing the necessary oxygen to render it "tough-pitch" presents a vast field of research hitherto practically unexplored. When this has been completed, the work will have only been

FIG. 9.—CAST COPPER COMPLETELY DEOXIDIZED.

commenced, for it will then be necessary to carry out similar experiments to demonstrate how these impurities behave in copper in the presence of another. As Dr. Watson (2d Report, Alloys Research) said: "The whole question of the proportion of oxygen which ought to be present in order to confer the best properties on copper containing various impurities had hitherto not been at all successfully investigated, so far as he knew. It was a subject presenting a really bewildering perplexity and involving difficult chemical analyses."

That pure copper or copper of more than usual purity, assaying, say, 99.8 per cent, is inferior to impure copper, both in mechanical properties, durability and resistance to corrosion has been made abundantly manifest from many independent sources. Webb has shown first that locomotive tubes of a purity of 99.8 per cent and over have proved to be far inferior in practice to tubes containing nickel or arsenic; secondly, that locomotive firebox stays behave satisfactorily when containing arsenic or tin. Dean proved that, on the whole, the purity of copper used in firebox plates was almost sufficient guarantee of its inferiority in practice. Roberts-Austen, Milton, Handscomb, Le Chatelier, Rudeloff and others have testified to the improvement in mechanical properties of copper conferred by the presence of traces of other metals.

The difficulty remains to show which impurities are most desirable, in what proportion, and the extent to which undesirable impurities may be present.

With regard to the effect of impurities on the electrical conductivity of copper, much research work has been done by Addicks and others. In this connection Sir W. Roberts-Austen, in a lecture delivered to the Institution of Civil Engineers, in 1902, said:

"I should have been glad, had time permitted, to deal with the improvements in the quality of copper used in electrical engineering. I will only say that if a metallurgist were now to offer to the electrical engineer such copper as was supplied in the middle of the past century, he would deserve to be relegated to the immortal pillory in which St. Paul placed Alexander, the coppersmith who did him much harm."

Electrolytic Chlorine at Mount Morgan, Queensland*

By B. du Faur.

As the Mount Morgan electrolytic chlorine plant is the first of its kind installed in Australia, and as the details of this method of producing chlorine have, up to the present time, been kept more or less secret all over the world, the author trusts that this paper may fill a gap in the technical literature of the subject.

When an electric current is passed through a strong solution of sodium chloride the following reaction takes place: $2\,NaCl + 2\,H_2O = 2\,NaOH + Cl_2 + H_2$. Chlorine gas is liberated at the anode, while caustic soda and hydrogen are formed at the cathode. The all-important problem in designing an electrolytic chlorine cell is the provision of some effective means of isolating from each other the chlorine and caustic soda as soon as liberated from the electrolyte.

In the type of cell which has been given a severe test for efficiency and reliability at Mount Morgan the products of anode and cathode are kept apart by means of an asbestos cloth diaphragm within reasonable limits (for there must always be some interaction taking place through the diaphragm, though this may be kept within bounds).

The Cell.—Each cell (see Fig. 4) is divided into three compartments by means of two sheets of iron *b* perforated with small holes. These sheets are bolted to the bottom and ends of the cell by means of 1-in. angle iron. The two diaphragms *c* are placed over the inside walls, and extend right to the bottom of the cell. The bottom and two ends of the central or anode compartment are now covered with cement 1 in. thick so that no iron is exposed to the action of the chlorine liberated in this compartment. These diaphragms extend upward to the point *d*. They are held in position by the earthenware top *e* of the cell, the intermediate space being filled in with cement. The top piece *f* of the cell can be lifted off for the renewing of anodes from time to time, but while the cell is in commission it is kept perfectly gas-tight by covering the top of it with a coat of hot pitch. It hardens almost immediately and forms a perfect seal, not too difficult to remove.

The anodes *a* are of Acheson graphite from the International Acheson Graphite Company, of Niagara Falls.

The anode holders have a composition of 80 per cent lead and 20 per cent antimony. They are sweated into a hole bored in the top of the anode block. In the top of each holder a groove is cut, and into this groove the electrical conductor bar is sweated.

The glazed earthenware pot *g* is connected to the anode compartment by means of an ebonite pipe *k*, so that the level of the solution in this pot shows that of the solution inside the anode compartment. Each cell is electrically insulated from earth by means of a glass plate on the top of each supporting column.

Purification of the Electrolyte.—Before further describing the plant it will be necessary to state here that all commercial samples of crude sodium chloride, and nearly all natural waters, contain salts of calcium and magnesium. It is absolutely essential that these salts must never be allowed to get

*A paper presented at the Mount Morgan meeting of the Australasian Institute of Mining Engineers.

into the electrolytic cells, for there they would be immediately precipitated and soon close the cell completely.

The following is the scheme devised and successfully carried out at Mount Morgan for throwing these salts out of the brine solution before it reaches the cells:

A, in Fig. 1, is a galvanized-iron tank with a water pipe *Q* running horizontally across the bottom, and is itself pierced with holes. This tank is filled to within a foot of the top with crude salt. It is shoveled into the top of the tank from a platform, not shown in the figure. The water is now turned into the pipe *Q* and rapidly rises through the salt, dissolving it quickly until, on reaching the overflow spout to tank *B* it contains, on an average, 3 lb. of salt per gallon.

Throughout the time that the water is turned on one man is kept busy shoveling salt into the top tank *A* so rapidly does it dissolve.

In the top of the empty tank *B* has previously been placed 10 lb. of soda ash dissolved in a bucket of hot water, also ½ gal. of an 8 per cent caustic soda solution derived from the electrolytic cells. As the tank *B* fills with brine, an air hose, which extends to the bottom, is turned on slightly, thus keeping the whole well stirred until the tank is full. The air is then turned off, and the stream of brine from *A* diverted to the second mixing tank *B*, which is filled in a similar manner.

After being allowed to settle for two hours the thick white precipitate of calcium and magnesium hydrate has practically all settled down below the pipe *R*. This pipe is then opened and the comparatively clear solution run over into a settling tank *C*, where it is again left undisturbed for two hours.

The small amount of sediment carried over into tank *C* has now completely settled, and the perfectly clear brine, free from all precipitated lime or magnesium salts, is run through pipe *S* to the brine storage tank *D*, from which it is fed to the electrolytic cells.

This method of cleansing the electrolyte has been so successful that on opening the cells at the end of a six months' run without stop they were found to be practically free from precipitated lime and magnesia salts. Before this was resorted to it was found to be impossible to run the cells for more than three weeks without cleaning out the deposited salts.

There was left in tank *B* 18 in. of deposited salts, mixed with salt water, which latter has now to be extracted from the solid matter. This sludge is run out of the bottom of the tank into a wooden launder, which carries it on to the filter cloth of a vacuum filter *E*.

On starting the filter a vacuum is created beneath the cloth by sucking out the air by means of a steam ejector

When the vacuum gage shows a vacuum of 23 in. of mercury steam is shut off, and a valve between the ejector and the filter closed, thus maintaining the vacuum. Experience has shown that this valve would hold the vacuum above 15 in of mercury for half an hour, when by turning on steam for a minute it could be brought back to the original 23 in

This filter filtered slowly (10 gal. per hour) owing to the exceedingly slimy nature of the precipitate, but, as it cost practically nothing to run, and could cope with the slime supplied to it, this did not matter.

The dry slime is shoveled off the filter cloth into a wheelbarrow on the floor and taken away.

Once a week it is found necessary to run the bottom portion of the solution in tank *C* into the filter also.

The clear brine filtrate from the bottom of the filter *E* is run into the brine storage tank *D*

It is found that the calcium and magnesium contents of the crude salt varied enormously, and, as a regular amount of precipitant was at first run into each tank, it follows that there was sometimes an excess of precipitant and sometimes an excess of calcium and magnesium left in the solution. Therefore, the following method of testing the purified electrolyte was necessary:

From tank *B* a sample was taken, filtered, and divided into two portions to be tested, the one for excess of free alkali and the other for excess of calcium and magnesium salts.

Portion 1.—Take 25 c.c. in a flask and titrate with a decinormal solution of sulphuric acid, using methyl orange as an indicator. If the end points were reached before 0.6 c.c. of sulphuric acid had been added from the burette, the tank was passed as being sufficiently low in free alkali. This was equivalent to 0.024 grain of free caustic soda per gallon, or 2.86 lb. in the 24-hours' supply of 300 gal. If more than 0.6 c.c. sulphuric acid was used, more salt water from tank *A* was run into tank *B* to neutralize excess alkali.

Portion 2.—Add a few drops of soda carbonate solution and

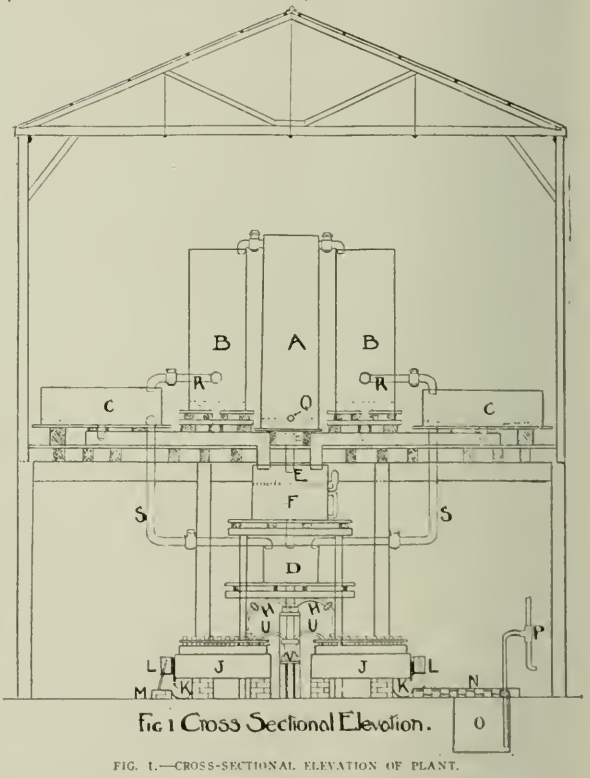


FIG. 1.—CROSS-SECTIONAL ELEVATION OF PLANT.

shake well; no precipitate shows absence of calcium salts. Add also a few drops of caustic soda solution and shake well.

If only a slight precipitate resulted the tank was passed as being good enough. If a heavy precipitate resulted more precipitant was added to tank *B* until it was found to be correct.

It is essential that not more than a certain amount of free alkali should be left in the purified brine, and so reach the anode compartment of electrolytic cell

The results of a greater excess of free alkali in the anode compartment are:

The cells would start to supply oxygen instead of chlorine at anode, according to the following equation.

$$NaOH + H_2O = NaOH + H_2 + O.$$

The reason for this is that of the two compounds, caustic soda and sodium chloride, simultaneously present in the elec

trolyte, the former will be first decomposed by the electric current.

The author has had this happen on several occasions, and was at a loss to understand the cause until, on testing the supply brine, he found too large an excess of free alkali present.

This combines partly with the chlorine, according to the following equations, forming sodium hypochlorite and sodium chlorate:

$$2\ NaOH + Cl_2 = NaCl + NaOCl + H_2O.$$

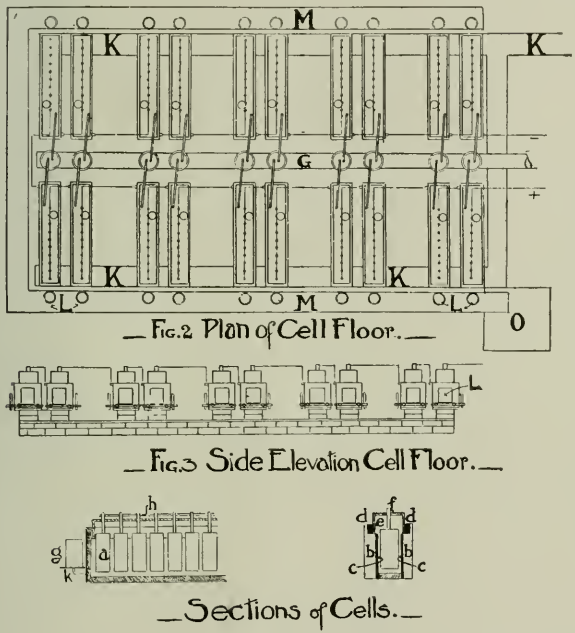
$$6\ NaOH + 3\ Cl_2 = 5\ NaCl + NaClO_3 + 3\ H_2O.$$

The presence of the former, at any rate, if not that of the latter also, causes rapid disintegration of the anodes.

The chief impurity in the salt was calcium sulphate, which is easily thrown out by the use of soda carbonate:

$$CaSO_4 + Na_2CO_3 = CaCO_3 + Na_2SO_4.$$

The magnesium chloride was partly got rid of by using caustic soda.



FIGS. 2, 3 AND 4.—HORIZONTAL AND VERTICAL SECTIONS OF ELECTROLYTIC CELL.

$$MgCl_2 + 2\ NaOH = Mg(OH)_2 + 2\ NaCl.$$

The purified brine from storage tank *D* drips into the anode compartment of each electrolytic cell by way of the glass funnels shown on top of the cells. Portion of this electrolyte passes through the diaphragms as caustic soda and undecomposed sodium chloride solution, leaving the cell via the movable arm *K*, which can be turned down to empty the cathode compartment at will.

The remainder of the electrolyte leaves the anode compartment by way of the porous pot *L*, and from there goes to the sump *O*. From there it is pumped up to a storage tank at sufficient elevation to force it through the salt-dissolving tank *A* again.

Chlorine gas is evolved at the anodes, and rising to the top of the compartment is drawn off by suitable means.

Hydrogen gas is liberated at the cathodes and escapes through the open top of the cell to the atmosphere, while the sodium set free at the cathode combines with the water present to form caustic soda, which leaves the cell in solution.

Salt Consumption.—The author has found that on a working scale where only chlorine is saved and caustic soda allowed to run to waste the production of 1 lb. of chlorine gas required 3 lb. of sodium chloride, dissolved in 1 gal. of water, to be fed to the electrolytic cells.

This sodium chloride is not absolutely pure, but in the following example we will take it as being so:

Then:

Chlorine present in 3 lb. sodium chloride..... = 1.80 lb.
Chlorine abstracted from sodium chloride..... = 1.00 lb.

Chlorine left as sodium chloride..... 0.80 lb.

This 0.80 lb. chlorine, representing 1.33 lb. sodium chloride, passes partly through the diaphragms and contaminates the caustic solution, and partly through the overflow pot *L*. If the caustic solution, which is not allowed to exceed 8 per cent in strength, is later on evaporated to produce strong caustic soda, this sodium chloride is thrown out of solution during the process and may be recovered. This would reduce the sodium chloride consumption per pound chloride produced from 3 lb. to 1.67 lb. If, on the other hand, the caustic soda is not evaporated, this salt is lost.

It has been found necessary to empty each cathode compartment of each cell once per 24 hours in order to keep the strength of the cathode liquors down to 8 per cent caustic soda. For, although the solution in the anode compartment stands at a slightly higher level than that in the cathode compartments, diffusion is always carrying a certain amount of cathode liquor into the anode compartment, and it follows that the stronger this liquor in caustic soda the more caustic soda gets into the anode compartment in a given time.

Evil Effects of Caustic Soda in Anode Liquor.—This caustic soda which finds its way into the anode liquors does harm in three different directions:

1. It reduces the available amount of caustic soda.
2. It reduces the available amount of chlorine.
3. The anodes are disintegrated and fall to powder, due to the presence of sodium hypochlorite and sodium chlorate, formed as shown previously.

It might be suggested that, if the rate of flow of electrolyte from anode to cathode compartment through the diaphragms were made great enough, the former would be kept free from caustic soda. Probably it would, but what would be the results? They would be:

1. Increased consumption of sodium chloride per pound chlorine produced.
2. Great loss of chlorine, due to the fact that the anode liquor, which is now passing rapidly into the cathode liquors, is saturated with chlorine gas, all of which becomes lost as available chlorine, besides also reducing the caustic soda yield.

Experience has shown that by keeping the caustic soda strength down to 0.8 lb. per gallon, or 8 per cent, a good output of chlorine and caustic soda is assured, though the anodes used become completely disintegrated after a continuous run of six months at a current of 350 amp, or 54 amp per square foot of anode surface.

Every six months the cells require to be opened for the introduction of fresh anodes and the clearing out of the disintegrated carbon. This disintegration of the carbon is common to all electrolytic chlorine cells using them. One American firm, however, claims to have found a method of reassembling the graphite dust into anodes again.

Power Consumption.—On starting the Mount Morgan plant with new anodes it was found that 88 volts were required to drive the 350 amp of current used through the 20 cells placed in electrical series. On the completion of the six months' run, when the anodes were nearly corroded away, the voltage had risen to 98.

This meant that the total electrical resistance of the plant had increased from 0.251 ohm to 0.280 ohm. Putting this in terms of electrical hp-hours of Board of Trade units, the total Board of Trade units required per 24 hours at the starting of the plant to give a certain daily output of chlorine were 739.20, while at the end of the run they had increased to 823.20.

Taking the cost of power at 1 d. (2 cents) per unit delivered at the switchboard, this meant that during the six months the cost of power per 24 hours had gradually increased from 739.20 d. ($14.78) to 823.20 d ($16.46).

Efficiency.—1. The efficiency of the plant taken on the basis of salt consumption per pound chlorine produced is 55 per cent, when the caustic soda is allowed to run to waste, while it becomes 99.80 per cent if it be saved. 2. The electrical current efficiency (amp-hour efficiency) is 63.72 per cent.

It may be said that this current efficiency is low, but it must be remembered that it was obtained by analyzing continuously the chlorine water actually delivered on the vats for leaching. This efficiency, therefore, covers all leakage of chlorine water tanks and loss of gas in chlorine water towers due to the insufficient supply of water at times.

3. The energy efficiency (watt-hour efficiency) was 49.24 per cent.

The technical data given in this paper will enable the reader to work out for himself the cost of producing chlorine in his particular locality by filling in the prices ruling there for the several commodities, such as power, salt, soda carbonate, wages, and water.

The above-described plant was run by four men, two on day shift and one on each other shift. These men could just as easily run a plant five times as large.

It is necessary that they should understand the duties of a dynamo attendant and be capable of being taught how to carry out the volumetric assays for chlorine water and caustic soda strengths.

In conclusion, the author would like to point out that since power is the main item of expense in the working of this process it follows that this power must be obtained at a reasonably cheap rate. To obtain this the dynamo must be directly driven, either by a steam engine or by a suction gas-producer plant. If electrical power has to be conveyed any considerable distance it is well known that the losses in transmission and transformation are so great as to put transmission out of consideration when comparing it with either of the above methods.

Except for water-power suction gas is now undoubtedly the cheapest available and most reliable source of power. The makers of many plants are now willing to guarantee a fuel consumption of coke of not more than 1.30 lb. per brake hp-hour, delivered by the engine, when running on full load. Consequently, when considering the installation of an electrolytic chlorine plant, the author strongly recommends suction gas as the source of power.

Hardness and Its Measurement.

By G. A. Rusch.

The physical property termed *hardness* has been defined in several different ways, and many different methods have been devised to measure it. Owing to the fact that hardness is in itself not a definite property, but the result of a combination of several different properties, this has given rise to results that are comparative only in a small degree, since the operation of one of the methods may depend on the measurement of an entirely different property or set of properties from some other method.

For example, it is evident that the elasticity and ductility of a material form a larger factor in the test where hardness is defined as the resistance of a body to permanent deformation than where hardness is defined as the resistance to penetration by another body. Hence the desirability of analyzing the exact components of any hardness tests that may be applied, and establishing such relations between these components that definite conclusions can be drawn from the results obtained. This is, however, a matter of considerable difficulty, and has not yet been accomplished.

Recent investigations have shown results pointing to a definite relation of this kind, but just exactly what it is cannot yet be stated. The object of this paper is the comparison of the hardness data[1] and the other physical tests[2] on the alloys of nickel, copper and Monel metal with electrolytic iron that have been made in the Chemical Engineering Laboratories of the University of Wisconsin, working under grant from the Carnegie Institution.

As has been said, the relative hardness of any material varies with the kind of stress to which it is subjected. The different properties which apparently may enter into and influence the results obtained by the different methods of testing are density, elasticity, ductility and tenacity. The degree in which these properties, taken in combination, are affected by the stress applied determines the hardness of the material under these con-

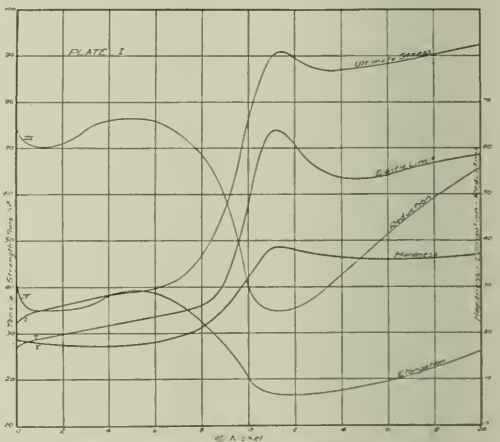


FIG. 1.—TENSILE STRENGTH AND HARDNESS OF FORGED NICKEL ALLOYS.

ditions. One and the same metal may show an entirely different degree of comparative hardness, depending on which of these components predominates in the final result, and a metal that is shown harder than another by one method may be softer when tested by a different method.

A series of samples arranged according to increasing hardness as measured by machining in a lathe will not necessarily remain in the same order when tested by a compression method. On this basis Turner[3] divides hardness into four different classes—tensile hardness, cutting hardness, abrasion hardness, and elastic hardness.

The method used for determining the hardness in the investigations cited was the scleroscope method devised by Albert F. Shore in 1906, and is described as follows. "The instrument consists of a vertical glass tube provided with a graduated scale, suspended over a steel anvil on which the sample to be tested is placed; a small steel plunger with a diamond point, suspended in the top of the tube, is allowed to drop on the

[1] *Met. and Chem. Eng.*, (a) p. 468, Aug., 1910, (b) p. 522, Sept., 1910.
[2] (a), Physical Properties of Iron Copper Alloys, Burgess and Aston, *Trans. Amer. Electrochem. Soc.*, 16, 241-256, 1909; (b), The Magnetic and Electrical Properties of Iron Nickel Alloys, Burgess and Aston, *Met and Chem. Eng.*, 8, 23-26, 1910; (c), The Magnetic and Electrical Properties of Iron-Copper Alloys, Burgess and Aston, *Met. and Chem. Eng.*, 8, 79-81, 1910; (d), The Strength of the Alloys of Nickel and Copper with Electrolytic Iron, Burgess and Aston, *Bull. Univ. of Wis.* Eng. Series, Vol. 6, No. 2, (e), The Strength of the Alloys of Electrolytic Iron and Monel Metal, Burgess and Aston, *Met and Chem. Eng.*, 8, 452-456, 1910.
[3] *Jnr. Iron and St. Inst.*, 1909, (1), p. 426.

sample clamped on the anvil, the rebound of the plunger being read off on the scale at the back of the tube. When the plunger reaches the surface of the metal under test it carries a certain amount of potential energy, and the weight of the plunger, the height of the fall and the area of the striking surface are so proportioned that in all cases this energy is sufficient to exceed the elastic limit of any material which may be under test.

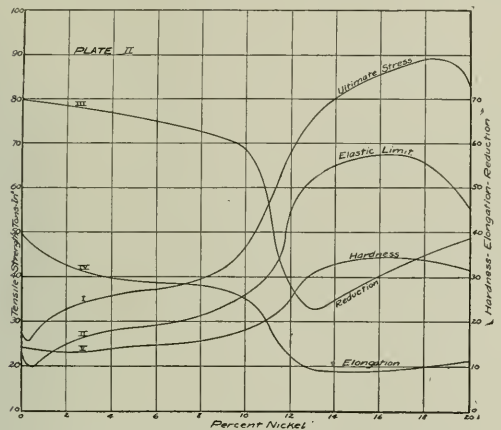


FIG. 2.—TENSILE STRENGTH AND HARDNESS OF ANNEALED NICKEL ALLOYS.

"This being the case, the material on receiving the impact of the plunger is stressed beyond its elastic limit and given a permanent deformation. If the elastic limit of the material under test is high, the excess energy of the plunger is small and only a small dent is made on the surface. That portion of the material immediately adjoining the deformed portion has then been stressed only up to its elastic limit, and in regaining

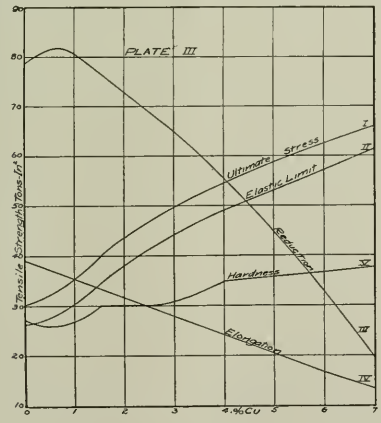


FIG. 3.—TENSILE STRENGTH AND HARDNESS OF FORGED COPPER ALLOYS.

its original form causes the plunger to rebound a certain distance. If the metal under test has a low elastic limit a smaller portion of the energy of the falling plunger is used in overcoming the elasticity, and a greater portion is used in displacing metal. The stress of the adjoining metal is then only sufficient to give the plunger a small rebound. This rebound is then taken to represent the hardness of the material under test."

Basing his conclusions on this line of reasoning, Shore makes the statement[4] that the curve of hardness as determined by the scleroscope is very nearly parallel to the curve of elasticity, but since the value obtained in any test for hardness is the resultant

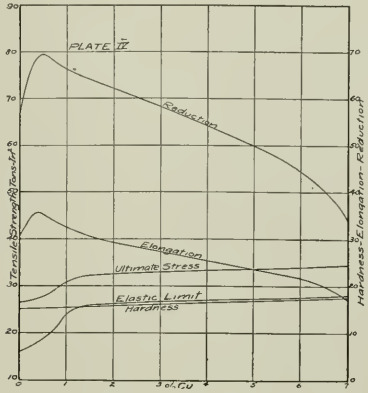


FIG. 4.—TENSILE STRENGTH AND HARDNESS OF ANNEALED COPPER ALLOYS.

of the different factors mentioned above it can be seen that this will be true only when those factors other than elasticity enter into the result to a very small extent, and that frequently the variation will be large.

Reference may also be made here to the statement made in the first paragraph of this paper in relation to the different properties affecting a hardness test, and the interpretation of

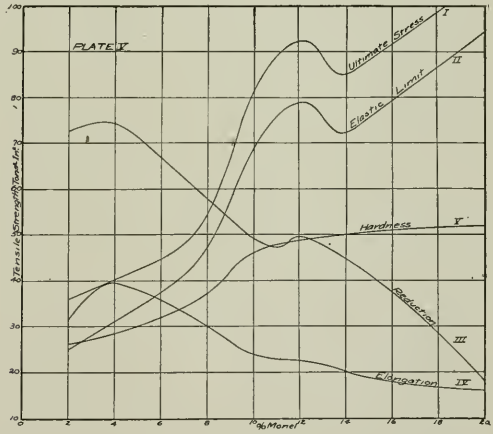


FIG. 5.—TENSILE STRENGTH AND HARDNESS OF FORGED MONEL ALLOYS.

the results for different purposes. In the article by Burgess and Aston referred to above[5] the iron-nickel alloys between 11 and 20 per cent are classed simply as "hard," no reference being made to any particular difficulty in machining, while the copper alloys above 5 per cent are said to have been machined with difficulty. The comparative hardness of the copper alloys, however, as measured by the scleroscope, is some 25 per cent lower than the nickel alloys.[6]

The comparison of the hardness curves with other tests that

[4] *Amer. Mach.*, Nov. 14, 1907.
[5] See reference above, 2d, pp. 10 and 20.
[6] See reference above, 1a, Tables I and III.

have been made on the same materials is a matter of considerable interest. In Figs 1 to 6 are reproduced the hardness curves for the nickel, copper and Monel alloys referred to,[1] along with the tensile tests on the same bars taken from the articles by Burgess and Aston cited above.[2] In each of these plates curve number I is the ultimate stress, II is the elastic limit, III is the reduction of area, IV is the elongation and V is the hardness.

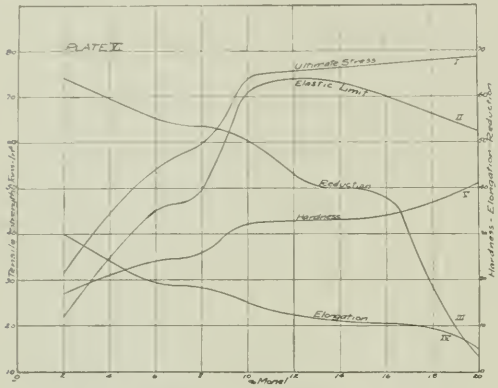


FIG. 6.—TENSILE STRENGTH AND HARDNESS OF ANNEALED MONEL ALLOYS.

As has been said, the hardness curve roughly approximates the general direction of the curve for the elastic limit, but in many points there is a wide difference. If the two curves followed each other exactly, it would be a simple matter to plot a curve showing the relationship between the two, which could be used to convert hardness tests directly into tests on elastic limit.

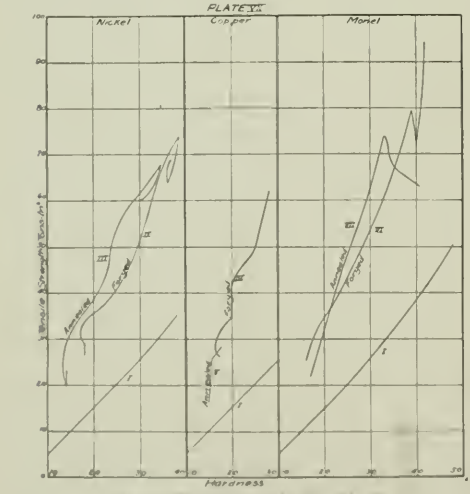


FIG. 7.—STRENGTH HARDNESS CURVES.

Among the data sent out with the scleroscope by the manufacturers is a table for converting hardness into elastic limit. This table is said to serve as a means of obtaining the strength tests on all metals below a hardness of 60, except cast iron and brittle alloys such as those of antimony. The data in this table are plotted in graphical form in curve 1, Fig. 7. This shows a simple relationship between the hardness and the elastic limit varying only slightly from a straight line function. This curve is obtained by plotting hardness as abscissæ and elastic limit as ordinates.

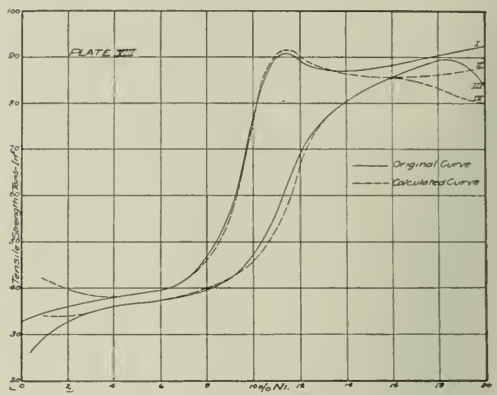


FIG. 8.—ULTIMATE STRENGTH OF NICKEL ALLOYS.

The relations between the hardness and the elastic limit as shown by the curves in Fig. 1 to 6 fail, however, to follow any such simple relation as this, nor are the values obtained at all close to those obtained by Shore. Furthermore, each one of the six series tried out here gives a different relation, and not one simple relation for all.

Curves II and III of Fig. 7 show respectively the results obtained by plotting in this same way the hardness and elastic limit of the forged and annealed nickel alloys; curves IV and V are from the forged and annealed copper alloys, and curves VI and VII from the forged and annealed Monel alloys.

In the nickel and Monel series the shape of the curves for the forged and annealed samples is not radically different, showing that the existing relations between the hardness and elastic limit are not materially changed by annealing the material, but in the copper series the change is quite marked.

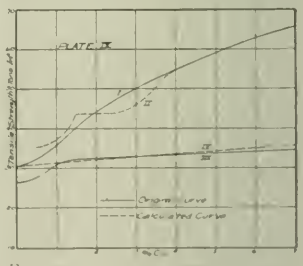


FIG. 9.—ULTIMATE STRENGTH OF COPPER ALLOYS.

A comparison of the hardness curves with those of elongation and reduction of area shows that it is the ductility of the material that causes the greater part of the variation of the hardness and elastic limit curves. Since the difference between the elastic limit and the ultimate strength depends to a certain extent on the ductility of the material it is probably more nearly correct to compare hardness with ultimate strength rather than with elastic limit. Following out this idea the attempt was made to reproduce the ultimate strength curves from the hardness curves, with the results shown in Figs. 8, 9, 10.

A conversion factor was obtained by dividing the ultimate strength shown at the middle of the curve by the hardness at the corresponding point, and this factor was then used to convert the remaining points on the hardness curve into values for ultimate strength. In each case the original curve for the forged samples is number 1 and the calculated curve is num-

[1] See reference above, 1a and b.
[2] See reference above, 2d and e.

ber II; the original curve for the annealed samples is number III and the calculated curve is number IV. These conversion factors, of course, differ somewhat for the different materials, but the variation is not as large as might be expected. The factors are as follows:

For forged nickel	2.38
For annealed nickel	2.60
For forged copper	2.38
For annealed copper	2.02
For forged Monel	2.25
For annealed Monel	2.30

None of the curves calculated in this way coincide throughout with the original, but they fall close enough to be of value in noting the points where the variations come, and may, in the light of further data, help in the location and evaluation of the factor or factors that are lacking to make the two curves duplicates.

It appears then that the scleroscope gives measures of hardness only approximately proportional to the elastic limit of the material, and will require individual calibration curves for each series of alloys in order that its readings may be resolved into values for strength. Six partial curves are given herewith in Fig. 7.

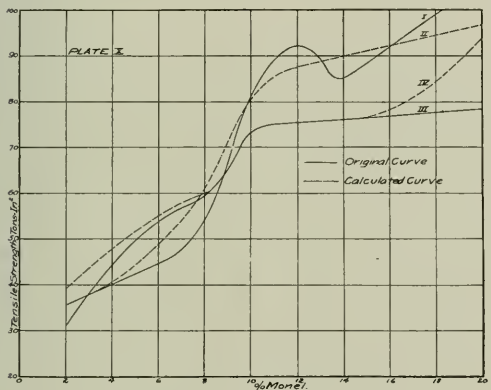


FIG. 10.—ULTIMATE STRENGTH OF MONEL ALLOYS.

While the results shown here point to the existence of a definite relation between hardness and elastic limit or ultimate strength, it appears that this relation is of such a complicated nature as to require further data than are now at hand to determine its exact nature and reduce it to mathematical terms.

University of Wisconsin,
Madison, Wis.

Gas Engines.—From the Pittsburgh office of the Bessemer Gas Engine Company, Grove City, Pa., comes a catalog illustrating their line of gas engines and auxiliaries. The Bessemer design has several distinctive features and might best be described as "different." The engine is two-cycle, thus getting a power impulse at every stroke, and uses a cross-head to reduce cylinder wear. The combustion chamber is valveless and the single valve used is not exposed to high temperature or pressure. Sizes are from 5 hp to 150 hp, the single, horizontal type running from 5 hp to 60 hp and twin cylinder 35 hp to 150 hp. Natural gas and producer-gas designs are furnished for commercial work and special lines for oil-field and electric light service. In addition to gas engines a line of direct gas-engine driven air and gas compressors, and direct-driven small pumps, also a separately driven triplex pump in sizes from 10,000 gal. to 500,000 gal. of water per day, are shown in the catalog.

A Convenient and Inexpensive Furnace for Very High Temperatures.

By D. F. Calhane, Ph.D.

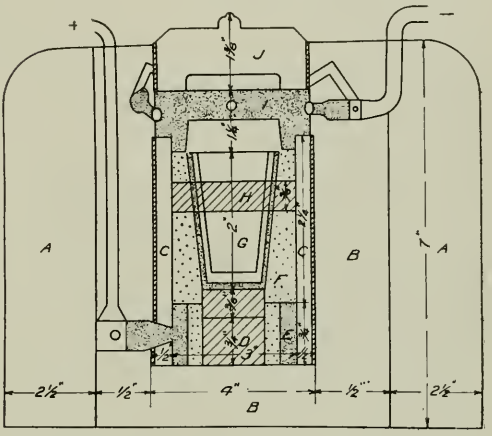
In an article by Pip in the August 15th number of the *Zeitschrift für Elektrochemie* is described a small furnace for high temperature work.

It occurred to me that a description of a furnace that I have used for the past two years in my laboratory might be of interest.

Some two years ago, in connection with work on nickel alloys, I had occasion to employ high temperatures, over 1500° C. It was especially desired to obtain the melts without carbon contamination. In the furtherance of this work, the following simple and efficient furnace was devised.

Referring to the figure:

A shows the ends of two fire-bricks. Four of these were



A. Fire Brick
B. Portland Cement Magnesia Oxide + Asbestos
C. Fire Clay Muffle Iron Band
D. 2 Alundum Blocks
E. Carbon Ring
F. Powdered Carbon
G. Alundum Lined Graphite Crucible
H. Alundum Insulating Ring
I. Graphite Ring
J. Fire Clay Cover

LABORATORY ELECTRIC FURNACE.

employed to form the outside of the furnace. The two end bricks were cut down to give the proper width to the body of the furnace.

C represents the cross-section of a small fire-clay muffle, such as is sold by the Buffalo Dental Company in connection with its No. 40 Fletcher's crucible furnace. This sets inside the enclosure formed by the fire brick and the extra space is filled with a heat-insulating composition composed of Portland cement, magnesium oxide and powdered asbestos. This is shown by B in the figure.

At the bottom of the muffle is placed an alundum block D. On this rests a No. O graphite crucible, shown by G in the figure.

In this graphite crucible is cast an alundum lining, which keeps the melts from carbon contamination.

The current enters by the lead at the bottom of the furnace, and is evenly distributed by a graphite ring E, that fits flush at the bottom of the muffle. This ring was cut from an odd end of a graphite electrode. The graphite electrodes lend themselves to this kind of work, as they are easily machined whereas carbon is not.

The carbon resistor is denoted by F. This is composed of carbon powder obtained from the National Carbon Company. It is an alundum ring, that is doughnut-shaped. The inner hole is filled to within about $\frac{1}{4}$ in. around by the crucible.

This design gives a simple method of insulating the main portion of the resistor, and directs the current flow along the walls of the crucible up through the ring between the wall of crucible and inner edge of the ring. Herein lies the high efficiency of the furnace. Intense heat is concentrated where it is wanted, namely on the walls of the crucible. By varying the area of the space between the crucible and inner side of the ring, and the height of the block D, the voltage and wattage of the furnace can be varied widely.

On the top of the muffle sets a graphite ring I, also made from an electrode end. This ring has contacts screwed in as shown, connected with leads for the exit of the current.

On the upper edge of the ring sets the cover of the muffle J. The top, from the upper surface of the fire bricks over to the edge of the cover J, is built up with the same insulating mixture used at B, and is fashioned to give a symmetrical outline to the furnace. The whole outside may be painted with water white to give a neat appearance. The whole body of the furnace rests on a cement base that is contained in a shallow, oblong box. The furnace can be thus carried about, as the whole thing weighs only about 25 lb.

A couple of examples showing the efficiency of the device may be of interest:

In one case the heat-resistivity of the device was pushed to the limit. For this size furnace 1330 watts were found to produce a temperature that fused alundum and must have been over 2500° C. In this experiment the furnace absorbed 70 amperes at 10 volts.

In another instance the furnace was calibrated with a platinum-iridium thermocouple and galvanometer up to the limits of this apparatus, namely 1600° C.

The results showed that at 700 watts a temperature of 1600° C. was recorded by the pyrometer. A rough extrapolation on this basis to 1330 watts would indicate a temperature in the neighborhood of 2800° C.

It might be asked how such a temperature could be attained without melting down the alundum completely. The answer is that the intense heat zone is only along the walls of the crucible, where no alundum is directly in contact.

A furnace on the above lines of 6 kw capacity will soon be in use here.

The cost of the size of furnace above described is very little. The graphite portions can be turned out on the lathe from electrode odds and ends by anyone who can operate a lathe. The muffle costs about $1.00. The alundum portions, that is the block, crucible lining and ring, were made by the Norton Company. The whole affair should not cost over $4.00 at most.

Sixty to seventy grams of metal may be melted in the 1-kilowatt size.

Worcester Polytechnic Institute,
Electrochemical Laboratory.

Electro-Metallurgy in Switzerland.—According to the London *Electrical Review* a group of important French and Swiss bankers have opened a credit of $1,600,000 to the Société Electro-metallurgique Paul Girod, of Neuenburg, in order to render it possible for the undertaking to unify its current liabilities and extend its plant. The Franco-Swiss banking group which has turned attention to the Girod Company is under the lead of the Société Marseillaise de Credit Industriel et Commercial, which occupies itself specially with enterprises engaged in the metal industry. A French subsidiary of the Girod company—the Compagnie des Forges et Chantiers de la Mediterranee of Ugine—was formed some time ago with a share capital of $800,000, which is held by the Swiss parent concern, and the company has been able to secure large orders from the French naval and military authorities.

The Chemical Import Trade of Germany.

By W. Pollard Digby.

Supremacy by any country in any manufacturing sphere involves a relatively small import trade in the commodities produced by the group of manufactures concerned. An import trade may and often does exist, such a trade being a subordinate one in the sense of affording to the works raw or partially manufactured material, which undergoes further treatment and becomes a finished article either for local consumption or for re-export. The import trade may also cover identical rival commodities entering to a small extent only and dependent for its persistence upon special conditions, such as cheap freights between the points of production and consumption.

After having discussed the chemical export trade of Germany we will now deal with its chemical import trade, considering the total chemical import trade of Germany for the 10 years ending 1908:

1899	$79,066,250
1900	82,841,250
1901	82,898,250
1902	80,716,750
1903	87,704,250
1904	94,407,000
1905	107,712,500
1906	109,431,000
1907	95,112,500
1908	88,550,750

the total value is about 75 per cent of that of the total chemical exports (Fig. 1). But very nearly one-third of the imports

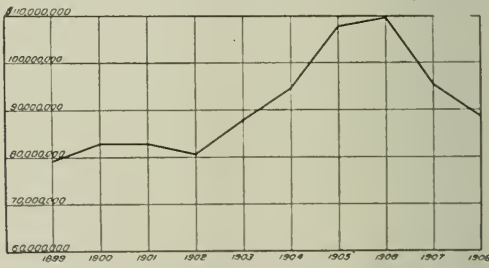


FIG. 1.—TOTAL CHEMICAL IMPORTS OF GERMANY.

consist of sodium nitrate, which, while classified correctly as an article of a chemical nature, serves to assist no other industry than that of agriculture by enriching the poor and sandy soil so familiar to all who travel through northern Germany.

Examining in detail the leading items and their origin the following data have been compiled from the official publications:

Sulphuric Acid.—The imports of sulphuric acid increased rapidly till in 1906 they exceeded the value of the exports. Although in the depression of 1907-08 imports declined, they are still in excess of the exports.

	1904.	1905.	1906.	1907.	1908.
Great Britain	.....	.....	.....	.....	.....
Rest of Europe	$175,000	$375,000	$908,250	$740,750	$760,500
United States	.....	.....	.....	.....	.....
Brit. No. Amer.	.....	.....	.....	.....	.....
Other Brit. Col.	.....	.....	.....	.....	.....
South America	.....	.....	.....	.....	.....
Rest of World	.....	.....	9,750	6,250	7,000
Total	$175,000	$375,000	$917,000	$747,000	$767,500

Caustic soda, soda ash and caustic potash.—While, as shown on page 465 of this journal, the total exports of these three commodities reached some $3,000,000 in value, the import trade is entirely negligible, amounting in 1908 to $6,500 for caustic soda, $6,500 for soda ash and $3,750 for caustic potash.

Bleaching Powder.—Imports of bleaching powder are also negligible, the total value in 1908 being $33,000.

Copper Sulphate.—The import trade has in 1907 and 1908 exceeded the total value of the exports. Fig. 2 shows the fluctuation of the imports.

	1904.	1905.	1906.	1907.	1908.
Great Britain...	$50,000	$100,000	$50,750	$418,000	$420,500
Rest of Europe.	50,000	75,000	51,250	104,250	62,000
United States..	50,000	50,000	96,750	.....	.....
Brit. No. Amer.	.....	.....	.....	.....	.....
Other Brit. Col.	.....	.....	.....	.....	.....
South America.	.....	.....	.....	.....	.....
Rest of World..	25,000	25,000	25,500	132,750	20,250
Total.....	$175,000	$250,000	$224,250	$655,000	$502,750

Sodium and Potassium Sulphates.—Here, again, the import trade is small, and the tables scarcely worth reproduction. Imports of sodium sulphate reached their maximum value of $78,250 in 1907 and declined to $33,000 in the succeeding year. Potassium sulphate was valued at $6,250 in 1908.

Sodium and Potassium Nitrate.—These two commodities, in which a fair re-export trade is carried out, account, as already pointed out, for a large proportion of the total value of the imports. Fig. 3 shows the import trade in sodium nitrate, the source of which is wholly South America.

Sodium Nitrate:

	1904.	1905.	1906.	1907.	1908.
Great Britain...	.....	.....	.....	.....	.....
Rest of Europe.	.....	.....	$42,750	.....	.....
United States..	.....	.....	9,000	.....	.....
Brit. No. Amer.	.....	.....	.....	.....	.....
Other Brit. Col.	.....	.....	.....	.....	.....
South America.	$24,650,000	$27,675,000	31,137,000	$31,785,250	$29,152,750
Rest of World..	25,000	50,000	20,500	17,500	12,250
Total.....	$24,675,000	$27,725,000	$31,209,250	$31,802,750	$29,165,000

Potassium Nitrate:

	1904.	1905.	1906.	1907.	1908.
Great Britain...	.....	.....	.....	.....	.....
Rest of Europe.	$225,000	$250,000	$198,250	$196,500	$231,500
United States..	.....	.....	.....	.....	.....
Brit. No. Amer.	.....	.....	.....	.....	.....
Other Brit. Col.	.....	.....	.....	.....	.....
South America.	.....	.....	.....	.....	.....
Rest of World..	.....	.....	15,750	7,750	10,500
Total.....	$225,000	$250,000	$214,000	$204,250	$242,000

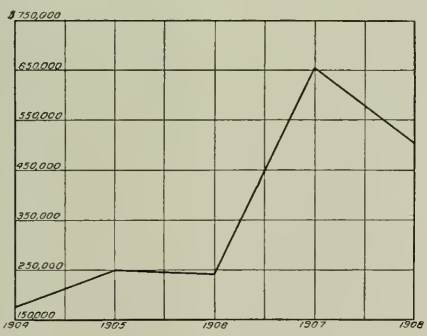


FIG. 2.—IMPORTS OF COPPER SULPHATE TO GERMANY.

Carbonate of Potash.—Small supplies are obtained from adjacent countries as indicated below.

	1904.	1905.	1906.	1907.	1908.
Great Britain...	.....	.....	.....	.....	.....
Rest of Europe.	$100,000	$100,000	$143,500	$175,000	$128,000
United States..	.....	.....	.....	.....	.....
Brit. No. Amer.	.....	.....	.....	.....	.....
Other Brit. Col.	.....	.....	.....	.....	.....
South America.	.....	.....	.....	.....	.....
Rest of World..	25,000	25,000	8,750	9,250	5,000
Total.....	$125,000	$125,000	$152,250	$184,250	$133,000

Aniline Dyes and Indigo.—The imports of aniline dyes and indigo are small in volume. The former represents about 4 per cent of the value of the export trade under this head, while the indigo imports, never large, fell in 1908 to about two-thirds of their value in 1904, and represented only about 2½ per cent of the exports.

Aniline Dyes:

	1904.	1905.	1906.	1907.	1908.
Great Britain...	.....	$100,000	$176,700	$116,250	$75,750
Rest of Europe.	.....	1,075,000	1,145,250	1,159,500	968,250
United States..	.....	.....	16,750	.....	.....
Brit. No. Amer.	.....	.....	.....	.....	.....
Other Brit. Col.	.....	.....	.....	.....	.....
South America.	.....	.....	.....	.....	.....
Rest of World..	.....	25,000	31,500	34,000	28,750
Total.....	.....	$1,200,000	$1,370,000	$1,309,750	$1,072,750

Indigo:

	1904.	1905.	1906.	1907.	1908.
Great Britain...	$25,000	$25,000	$29,750	$18,000	$29,750
Rest of Europe.	25,000	50,000	.....	.....	.....
United States..	.....	.....	.....	.....	.....
Brit. No. Amer.	.....	.....	.....	.....	.....
Other Brit. Col.	150,000	125,000	68,250	86,750	110,500
South America.	75,000	25,000	.....	.....	.....
Rest of World..	75,000	75,000	109,250	166,000	80,250
Total.....	$350,000	$300,000	$207,000	$207,750	$220,500

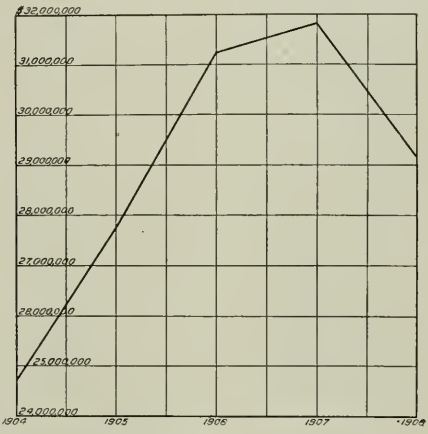


FIG. 3.—IMPORTS OF SODIUM NITRATE TO GERMANY.

Ammonium Sulphate.—The imports of ammonium sulphate have been fairly steady in value, Great Britain almost regaining in 1908 the total value supplied by her to Germany in 1905.

	1904.	1905.	1906.	1907.	1908.
Great Britain...	$1,100,000	$1,675,000	$767,500	$843,750	$1,494,750
Rest of Europe.	1,025,000	1,275,000	1,172,500	1,239,000	1,337,750
United States..	.....	.....	.....	.....	.....
Brit. No. Amer.	.....	.....	.....	.....	.....
Other Brit. Col.	.....	.....	.....	.....	.....
South America.	.....	.....	.....	.....	.....
Rest of World..	75,000	100,000	270,250	12,500	3,500
Total.....	$2,200,000	$2,950,000	$2,210,250	$2,095,250	$2,836,000

This brief article covers the small import trade of Germany in chemical substances. Only the leading imports have been described. There are, of course, numerous minor items, but these do not merit separate mention. Deducting nitrate imports the remaining balance bears a small proportion to the German exports. The forces which have tended to produce this end were briefly indicated in the August issue of this journal.

The increased use of cement is noted in a report issued by the U. S. Geological Survey. More cement was made and used in 1909 than in any other previous year, and the price per barrel was also lower. The production in 1908 was 52,910,925 barrels valued at $44,477,653; in 1909 the production was 64,196,386 barrels valued at $51,232,979. The average price of Portland cement in 1909 was less than 81 cents per barrel as against 85 cents in 1908. There were 103 Portland cement plants in operation in 1909 in this country, an increase of 5 over 1908.

Modern Hydrometallurgy in Australia.

By J. R. Masson and J. E. Edwards.

Hydrometallurgy has made such rapid strides during the past few years that it may be said to be almost, if not quite, the universal method of treatment for gold and silver-bearing ores. Although the original principle of the cyanide process still remains, there are more up-to-date methods of application being introduced every day. Rapid extraction and quick separation of the solution have long been aimed at, and these problems have been solved.

A short sketch of Australian experience will make this clear. The old treatment of heavy sulphide ore was as follows:

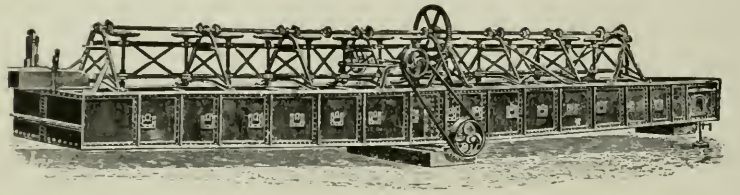


FIG. 1.—TILTING TYPE.

1. Crushing (wet).
2. Concentrating.
3. Roasting of concentrate.
4. Chlorination of roasted concentrate.
5. Separation of sand and slime from residue after concentration.
6. Cyanidation of sand by percolation.
7. Cyanidation of slime by agitation.
8. Filter pressing of slime.

More recently the treatment has been considerably simplified, owing to the development of mechanical devices which will be described later, and now consists of:

1. Crushing.
2. Roasting.
3. Agitation of sand and slime together in cyanide solution.
4. Filtration of solution.

The ore is usually crushed to 30 mesh, roasted in Edwards furnaces and then agitated with cyanide solution in A. Z. agitators. Until this agitator was introduced the above method was not possible, as never before had there been a machine capable of mechanically agitating and aerating sand and slime together. The great advantage found in the agitation of the combined products is that a quick separation of the cyanide solution is easily accomplished, and may be made in an ordinary percolation vat. The most up-to-date method however, is to discharge the contents of the agitator into what is known as the Edwards rapid filtration tank, in which the solution is driven through and separated from the treated ore under pressure. This simple method is giving every satisfaction and is very rapid.

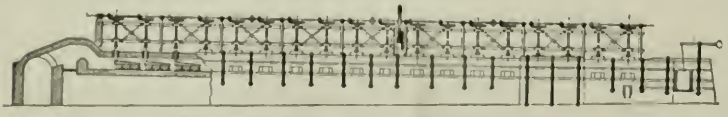


FIG. 3.—SIDE ELEVATION OF FIXED HEARTH TYPE.

Roasting.

In the design of mechanical roasting furnaces certain desirable conditions of operation and proportions in construction have been adapted from the old-fashioned hand-rabbled furnace. The correct height of arch, the proper relation of width of hearth to height of arch, and the elimination of dead spaces are the important features sought and attained in the Edwards furnace. It has a rabbling mechanism which permits of the construction of a low arch, so that the most economical application of the heat is obtained. The advantage of this is found in lower fuel cost, more complete oxidation of the ore, and more even distribution of heat over the hearth.

The dead space in this furnace is confined to small triangular areas next the walls, similar to the spaces between the side doors of the hand-rabbled furnace. The intersection of the circular paths of the rabbles in the Edwards furnace provides for effective agitation of the ore, at the same time advancing it gradually toward the discharge end. Another point in the construction of this furnace is the provision for the introduction of air wherever necessary in order to effect the most complete oxidation. This is accomplished by arranging numerous doors along the sides of the furnace. All of these considerations make the furnace most effective in the roasting of sulphide and sulpho-telluride ores prior to treatment by cyanidation or chlorination.

Two different types of furnace are made, that known as the simplex having but one row of rabbles, and the duplex having two rows. These two types are built in different forms, known as the tilting (shown in Fig. 1) and the fixed hearth, Fig. 2. The former is built of steel lined with brick, and is balanced at its center support and can be adjusted to any desired slope of hearth. The latter, as its name implies, is of fixed reverberatory construction.

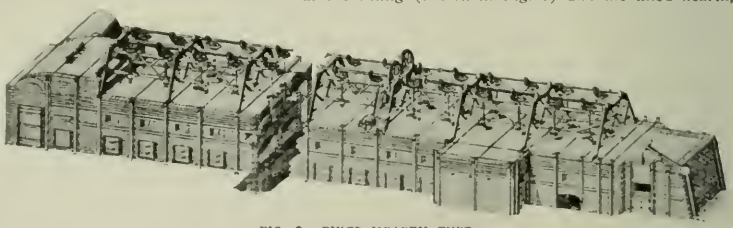


FIG. 2.—FIXED HEARTH TYPE.

As shown in Figs. 3 and 4, which are respectively side elevation and plan of the duplex fixed-hearth type, the rabbles consist of horizontal arms with depending teeth. Each rabble is attached to a vertical spindle which projects through the arch of the furnace. The spindles are actuated by gear mechanism on top of the furnace so adjusted that each rabble rotates in direction opposite to the next, and the circle of rotation of one rabble intersects that of the next adjoining rabble. In the duplex furnace the intersections also occur between the two rows of rabbles. The rabble and that portion of the spindle which is within the furnace is cooled by a positive circulation of water.

The Edwards furnace is most widely used in Australia, but has also met with a favorable reception in this country, where it is in use at large plants treating both sulphide and sulpho-telluride ores. The customary size of the tilting type is 57 ft. x 6 ft. 6 in., and of the duplex fixed hearth type, 100 ft. x 11 ft. The power required

varies with conditions, but may be placed approximately at 2 hp to 3 hp for the tilting, and from 6 hp to 9 hp for the duplex fixed hearth type.

Following are exact data on the operation of 12 tilting furnaces at the Great Boulder mine, Western Australia:

Ore contained 3.5 per cent sulphur; calcines contained 0.05 per cent.

Capacity of furnaces, 25 tons each in 24 hours.

Slope of hearth, 1 in. in 75 in.

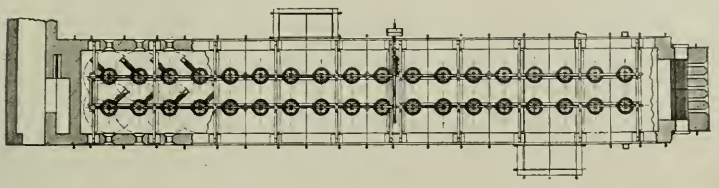


FIG. 4.—PLAN OF FIXED HEARTH TYPE.

Power required per furnace, 2 hp.

Fuel consumption, 0.2 per cent by weight of ore roasted.

Water required for rabbles, 400 gal. per furnace per hour.

Temperature of water raised 30 deg. in cooling process.

Data on other operations show a reduction of the sulphur content of various ores from 35 per cent to 0.285 per cent and 0.235 per cent. On sulpho-telluride ores such as Cripple Creek produces the furnaces installed at the Golden Cycle mill, Colorado Springs, Col., have a capacity of 100 tons per 24 hours, giving a sweet roast prior to cyanidation. The sulphur content of the ore is 2.5 per cent, and of the roasted product 0.08 per cent. The fuel consumption at this plant is 11 per cent, and the cost of roasting between 55 and 60 cents per ton.

These examples will serve to show the place which roasting

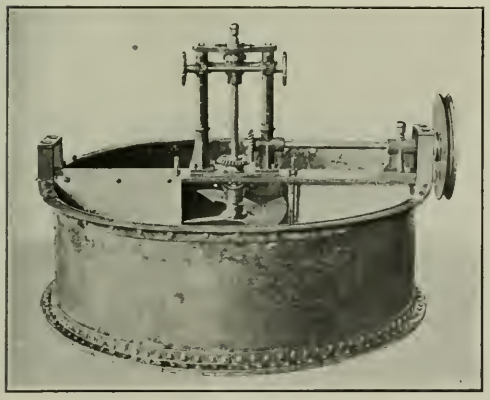


FIG. 5.—MODEL OF A. Z. AGITATOR.

is assuming in hydrometallurgy, where it is impossible to treat the ore chemically in its natural state.

Agitation.

After roasting the extraction of precious metal takes place in the A. Z. agitator. This device is simple in construction and can be applied to any circular tank. As shown in Fig. 5, which represents a small model, it consists essentially of a propeller placed centrally in the tank, with two deflector plates, one on each side of the propeller. The deflectors are fixed in position and extend almost to the bottom of the tank, while the propeller is adjustable up or down and when in operation revolves in a horizontal plane within a shallow cylinder open at both ends. The diameter of the propeller is usually about one third that of the tank.

The combination of propeller and deflector plates brings about an efficient agitation by setting up underlying centrifugal currents radiating from the bottom and center of the tank to all points of the tank wall, and overlying centripetal currents from the wall to the center. The deflectors prevent the entire mass from acquiring a swirling motion in the tank.

Fig. 6 illustrates a 20-ft. tank, of a capacity of 50 tons a day, in operation. The effect of the agitation is seen on the tank contents. The surface assumes a slightly concave form as the currents rising at the wall are constantly drawn downward toward the center of the tank. In the downward flow air is naturally drawn into the pulp and thor-

FIG. 6.—THE A. Z. AGITATOR IN OPERATION.

oughly mixed with it, and the evidence of thorough aeration is seen in the effervescence which occurs when the agitator is stopped.

The advantages claimed for the agitator are:

1. Saving of time in treatment.
2. Reduced consumption of cyanide.
3. Increased extraction.
4. Low power requirements.
5. Application to any existing round tanks.
6. Agitation of mixed sand and slime, eliminating classification.
7. Aeration without accessory air-compressing apparatus.
8. Rapid quiescence and settling of the pulp when the propeller is stopped.
9. Ease in restarting, even after long settlement of pulp.

Since its introduction in Australia the A. Z. agitator has been installed in all of the important metallurgical works in that field. It is used not only in the cyanide process, but also as a thorough mixer where uniform delivery and mixture of pulp are demanded, as in the De Bavay zinc separation process, and for concentration-table feed. In ordinary cyanide work it is found that the extraction is usually at its maximum in about four hours. Either sand, slime or a mixture of both may be treated, either roasted or raw, excellent extraction being obtained on some raw concentrates.

An example of the work done by this machine is found in the operations of the Waihi Grand Junction Gold Company, Waihi, N. Z. Table concentrate is cyanided raw in 20 ft. x 7 ft. A. Z. agitators, with a recovery of 92.3 per cent of the precious metal values, after 24 hours' agitation. The charge consists of 50 tons of concentrate and 46 tons of solution, the

total of 96 tons being agitated with 11 hp. The specific gravity of this concentrate is 4.5. Another example, from Western Australia, is found at the works of the Associated Gold Mines, where the time of agitation was reduced from 12 to 4 hours, or one-third the time formerly required with other methods. The improvement in recovery was as high as 4 per cent more than that obtained without these agitators.

In addition to its work in metallurgy, the A. Z. agitator is efficient in various branches of chemical engineering where thorough mixing, emulsifying, dissolving or evaporating is required.

Treating Antimonial Gold-Bearing Ores.

Not so long ago it was considered unlikely that the cyanide process would ever be used successfully in connection with the treatment of certain refractory ores. Recently this theory has been upset. Antimonial gold-bearing ores have always given trouble and have been avoided, but to-day these ores can be handled successfully, and the improvement in hydrometallurgy is responsible for the solution of the problem. A description of the Masson process (for which patents have been issued) may be found of interest. This process is in extensive use in the Wandong district, Victoria, Australia, and especially at the famous Golden Dyke antimony mine. The ore treated usually contains about 20 per cent antimony (stibnite) and from $10 to $12 gold per ton. The method of treatment is as follows:

The ore is ground to pass a 40-mesh screen and is then agitated in A. Z. agitators with hot solutions of caustic alkali. The agitation is so perfect that the antimony goes into solution very rapidly, usually in about two hours. The liquor containing the antimony is then separated from the ore, and run into precipitation tanks where the antimony is precipitated very rapidly as pure metal on sheets of aluminium. After the extraction of the antimony, if the ore contains a large percentage of sulphide of iron, it is roasted in Edwards furnaces before being subjected to agitation with weak KCN solution for the extraction of the gold contents. This second agitation also takes place in A. Z. agitators, and it is found that after roasting no trouble is experienced in obtaining high extraction of the precious metal, as high as 97 per cent of the gold being recovered.

The consumption of cyanide is very slight, in fact the average consumption at the Golden Dyke mill is 0.04 per cent KCN per ton of solution used. The cost of the whole treatment is approximately $3 per ton of ore treated.

Numerous experiments were made at Wandong to ascertain whether stibnite actually consumed cyanide or not, as it was generally believed that it was the cause of heavy consumption. The results of these experiments on an old accumulation of tailings clearly proved to the writers that raw native sulphide of antimony is not responsible for the heavy consumption of cyanide, as is generally believed, provided the ore has been properly prepared for treatment. The difficulty seemed to lie in the proper neutralization of the ore. When using the leaching method this was very noticeable and gave great trouble.

Lime was used as a neutralizing agent and after various trials it was found advisable to make up an alkaline liquor by dissolving as much of the lime as possible, and then leaching this solution through the tailings in the vats. Complicated reactions took place very rapidly, and it was found that as soon as the surface of the ore became neutral the alkaline solution following would dissolve a portion of the stibnite, which would, in turn, be precipitated as red sulphide with the generation of sulphureted hydrogen gas as soon as the solution reached the strongly acid portion of the ore. The consequence was that the charge would become full of red sulphides, which react most injuriously with cyanide.

Leaching methods were then abandoned and agitation with the A. Z. agitator was tried. The agitators were filled with water and started. The ore was then introduced, and when everything was in suspension the lime was added. This was a complete success, as the entire ore content came in contact with the alkaline solution at the same time. When it was seen that the solution was fast gaining in alkalinity, as the lime was added gradually, it was at once recognized that it could be brought to any desired point, care being taken not to exceed the neutral point. After the separation of this solution the ore was subjected to agitation with a 0.15 per cent KCN solution. The consumption of cyanide was only 0.03 per cent and the extraction of gold was 87.32 per cent after one and one-half hours' agitation. The average value of these tailings was $6 in gold and 7 per cent of antimony.

Besides these experiments many others were made to compare the efficiency of the A. Z. with that of other forms of agitators under the same conditions. These results are also of interest as all other forms failed to give the results sought. Some, while agitating the finer products, failed thoroughly to agitate the material of higher specific gravity. Again the aeration in the other devices was faulty, and this would, probably to a great extent, account for the great amount of time saved by the A. Z. over the other machines tried. The perfect aeration of the A. Z. is one of the most important of its many good qualifications, being largely responsible for the reduction in time and improvement in recovery mentioned before.

International Congress of Applied Chemistry, 1912.

As repeatedly mentioned in these columns, the eighth International Congress of Applied Chemistry will meet in 1912 in the United States in accordance with an invitation extended to it by the President of the United States and as provided for by joint resolution of the Sixtieth Congress.

The official delegates from the Government of the United States to the Seventh International Congress of Applied Chemistry, held in London in 1909, were designated by that body as members of the organizing committee of the 1912, or Eighth Congress, with power to add to their number.

These official delegates, numbering 13, added 30 new members to this organizing committee. This so augmented committee held a meeting on Aug. 26 in New York City and again enlarged its membership. This meeting was very harmonious and it was evident that everybody endeavored to organize the Congress on the broadest foundation of chemical democracy.

The honorary president of the congress will be Dr. Edward W. Morley, West Hartford, Conn.; the president, Dr. William H. Nichols, New York City. The secretary is Dr. Bernhard C. Hesse, 25 Broad Street, New York City.

The Congress will be held in the first half of September, 1912. The place of the opening meeting will be Washington, D. C., with all subsequent sessions of the congress and its various sections to be held in New York City. Thereafter as many other cities as practicable shall be visited by the congress.

Nothing else has so far been definitely decided upon. Several of the more urgent matters are to be decided at a meeting of the organizing committee, to be held on Oct. 8, 1910, in New York City.

The congress will have the following sections and subsections: 1, analytical chemistry; 2, inorganic chemistry; 3a, metallurgy and mining; 3b, explosives; 4, organic chemistry; 4a, colors; 4b, physiological chemistry and pharmacology; 5, industry and chemistry of sugar; 6a, starch industry; 6b, fermentation; 7, agricultural chemistry; 8a, hygiene, 8b, pharmaceutical chemistry; 8c, bromatology, 9, photochemistry; 10a, electrochemistry; 10b, physical chemistry; 11, law, political economy and legislation affecting the chemical industry.

While the above sections and subsections are now provided for, new subsections may be formed, and the following four have so far been proposed: chemotherapy, conservation, india rubber and plastics.

Standardization as a Phase in Engineering Evolution.

It may fall perchance to the lot of some engineer who may have attained to all the honors which his branch of the profession can accord to him, to sit down in the closing years of his life to write a comprehensive account of the evolution of engineering. Not, however, to write a mere text-book type of narrative such as those chapters of books on steam engineering which merely describe early inventions without describing the force which led to their development, and which fail to narrate succinctly the manner or reason of their subsequent supersession. There exists already a surplus of such engineering literature, incomplete historically, but affording material among which the philosophic historian may obtain information sadly needing philosophic codification and study. The engineer-historian-philosopher in his work of modification can certainly make some quotations from Mr. Herbert Spencer's works, which afford the best ground upon which such an amplification may be based.

Mr. Spencer's early years of engineering work enabled him, when using engineering analogies to illustrate points in his writings, to select examples peculiarly applicable to the case under discussion. In none of his writings is this more noticeable than in "First Principles," where he first enunciated a definition of evolution which has not been upset by the later developments of scientific and social progress. More striking even than the definition itself are the chapters showing that the various distinct stages of evolution are common to the human race in its function of government, language, food and clothing, as well as to the solar system, or to the strata of the earth itself, or to the arts of sculpture, painting, music or literature. It is, therefore, an entirely permissible matter when taking into account what place standardization really occupies in engineering history, to consider it in the light of his definition of evolution.

That definition, as many readers will remember, is that "evolution is an integration of matter and concomitant dissipation of motion, during which the matter passes from a relatively indefinite incoherent homogeneity to a relatively definite coherent heterogeneity, and during which the retained motion undergoes a parallel transformation."

The historian of engineering evolution will have to consider many forces, such as that of segregation due to a following of the line of least resistance. Mr. Spencer points out: "If, then, there be any locality which, either by its physical peculiarities or by peculiarities wrought on it during social evolution, is rendered a place where a certain kind of industrial action meets with less resistance than elsewhere, it follows from the law of direction of motion that those social units who have been molded to this kind of industrial action will be segregated by moving toward this place. If, for instance, the proximity of coal and iron mines to a navigable river gives to Glasgow an advantage in the building of iron ships—if the total labor required to produce a given vessel, and get its equivalent in food and clothing, is less there than elsewhere, there is caused a concentration of iron-ship builders in Glasgow, either by detention of the population born to iron-ship building or by immigration of those elsewhere engaged in it, or by both."

Or, the phenomena of integration which causes certain animals to hunt in packs, or integrations "which arise when, out of several places producing a particular commodity, one gaining more and more of the business, draws to it masters and workers, and leaves the other places to dwindle, as witness the growth of the Yorkshire cloth districts at the expense of those in the west of England; or the absorption by Staffordshire of the pottery manufacture, and the consequent decay of establishments at Derby and elsewhere. We have those more special integrations that arise within the same city, whence result the concentration of corn merchants about Mark Lane, of civil engineers in Great George Street, of bankers in the center of the city. Industrial integrations which consist, not in the approximation or fusion of parts, but in the establishment of centers of connection are shown in the bankers' clearing house and the railway clearing house. While of yet another species are those unions which bring into relation dispersed citizens who are occupied in like ways, as traders are brought by the Exchange, and as are professional men by institutes by those of civil engineers, architects, etc." * * *

"The progress from small and simple tools to complex and large machines is a progress in integration. Among what are classed as the mechanical powers the advance from the lever to the wheel-and-axle is an advance from a simple agent to an agent made up of several simple ones. On comparing the wheel-and-axle, or any of the mechanical appliances used in early times with those used now, we see that in each of our machines several of the primitive machines are united. A modern apparatus for spinning and weaving, for making stockings or lace, contains not simply a lever, an inclined plane, a screw, a wheel-and-axle joined together, but several of each—all made into a whole. Again, in early ages, when horse-power and man-power were alone employed, the motive agent was not bound up with the tool moved, but the two have now become in many cases joined together. The firebox and boiler of a locomotive are combined with the machinery which the steam works. A much more extensive integration is seen in every factory. Here numerous complicated machines are all connected by driving shafts with the same steam engine—all united with it into one vast apparatus."

Or, again, a universal economic tendency toward greater heterogeneity is caused by the increasing ties, inter-commercial, inter-scientific, which bring into closer and dependent relationship town and town, industry and industry, state and state, continent and continent.

The student of engineering evolution will find it a very complex matter to set out the whole historically by reason of the close interrelation of section and section. The advance of the steam engine is not merely a matter of increasing sizes and a striving after economy of production, but is dependent upon the development of larger factories and a demand for larger engines, as well as upon the toolmaker who provided the engineer with heavier lathes, no less than upon the boilermaker for boilers capable of giving high-pressure steam. Both engine maker and boiler maker alike owe their great advances to the metallurgist for cheaper, lighter, and stronger raw materials.

The advance of the steamship to the 20,000-ton and 30,000-ton liners of to-day has been an instance in which the ship builder, the engine maker and boiler maker, with their allies at the furnace and the foundry, have responded to the demands of altered commercial conditions. The progress of the dynamo from the days of Faraday's experiments in electromagnetic induction to the numberless dynamos and motors of to-day has been a progress acted upon by many diverse forces. While the progress may be traced in its logical scientific sequence through several contributing fields, such as those concerned with magnetic permeability or with insulation or conductivity, there must also be studied the influence of parallel forces which render these subordinate developments possible, and which provided successful commercial employment for what would otherwise have been as useful industrially as a Wimshurt machine would have been to Christopher Columbus.

The present movement in favor of standardization in engineering must logically date back to the days when palaeolithic man expressed a general tribal preference for an axe head of a certain average weight, and when he hewed his flint arrowheads to as nearly a common size as possible. His successor demands guns of an exact bore, while cartridges are not made on any haphazard principle as to size.

Standardization really reaches its highest development in connection with military affairs. The highly drilled and uniformed regiments, armed with weapons whose parts are inter-

changeable, has slowly evolved from the days when the warriors of nomadic tribes followed their chiefs into battle. As in warfare, and in its preparations, the best machines are those which are the most fully equipped for the duties likely to confront them, so in the commercial struggles being waged, or about to be waged, between firms in the same industry, or between country and country, the organizations most likely to succeed are those whose methods secure the widest range of activities with the greatest economy.

Standardization in engineering has been a factor from the days when iron founders started to store their patterns in order to furnish sets of castings the exact duplicates of one another; later the almost universal adoption of the Whitworth standard threads for bolts and nuts of moderate and large sizes secured universal adoption, because of the real demand for the interchangeability of bolts and nuts of equal diameters. Standardization, too, also became the rule in certain factories, and side by side with the disappearance of the old millwright or all-round craftsman came the introduction of specialist workers with specialized duties engaged in the manufacture of standard interchangeable parts for machines of special types.

Standardization to a certain degree has been of great importance in railway design, and while one may regret the result of the battle of the gages in the United Kingdom, the benefits resulting from the adoption of one standard gage are undoubted. So also a standard loading height above rail level because of bridges and tunnels. Although certain dimensions in regard to railway work have become the heritage of the railway engineer, a considerable elasticity yet remained. With the increase of loads and speeds has come the demand for heavier permanent way construction, a matter in regard to which finality, while not yet attained, has resulted in the distinct classification of standardized weights per yard, sections, tests, and a rigid chemical analysis pertaining to bull-headed rails, flat-bottomed rails and tramway rails.

The British Engineering Standards Committee has also standardized pipe flanges (thereby affording infinite relief to many mechanical engineers), screw and pipe threads, Portland cement, copper conductors, and dielectric thicknesses, telegraph material, tramway poles, trolley grooves, trolley wires, speeds, pressures and frequencies for electric motors and generators, and, above all, rolled sections for constructional iron and steel. Locomotives have also been standardized for use on the Government railways in India. Mention must also be made of incandescent lamps and boiler tubes, rolled sections for bridges, building construction, and ship building; tests for iron and steel materials used for ships and for their machinery; steel castings and forgings for marine work; tires, axles and springs; electric supply meters; copper and its alloys; railway rolling stock, underframes and cast-iron pipes, while electric tramways, electrical plant accessories and other matters are under consideration. This list does not cover the whole category of engineering work, but so much has sprung from the appointment by the council of the Institution of Civil Engineers, in January, 1901, of a committee to consider the advisability of standardizing rolled iron and steel sections that ultimately one may conceive the Engineering Standards Committee with its definite tests as the supreme court of jurisdiction in regard to all engineering work, covering dimensions of component items, their composition and their physical properties.

Such a body will naturally be confronted time and again with the question to standardize or not to standardize. The history of every branch of engineering is a history of rival types and rival methods. In the competition between these, before the unsuitable went to the scrap heap, the successful types underwent several changes, increasing in complexity and acquiring a larger adaptiveness. When pre-eminence is secured for any particular practice, then comes the time for standardization. Among present problems of this nature may be said to be that of electric traction.

The need for standardization upon electric railways was urged by Mr. James Swinburne and Mr. W. R. Cooper in a joint paper read by them before the Institution of Electrical Engineers in 1902. In the discussion of the paper the present writer urged that at that time it was premature to talk about standardization of pressures or other electrical conditions so long as the gage and height above rail level were preserved, and so long as electric locomotives were employed. Interchanges between system and system could be effected by changing locomotives at junctions. At the present day direct-current, 500-volt systems have the longest mileage in England. The details as to the position of the conductor rails vary on different systems. There is, however, a keen fight ahead between direct-current and single-phase alternating-current systems, with the great probability that the latter system will be adopted in cases where both the suburban and main line systems will require electrification.

The above instance has its parallel in a number of other cases. The only moral to be drawn would be that the standardization should not be too rigid, and that the various committees and subcommittees should be always ready to revise existing standards, scrapping types which are obsolete as ruthlessly as they would scrap a bolt from which the thread has been stripped, and always ready to advise the introduction of new standards accordingly as the condition of the industry demanded. In short, the industry demands the same foresight and the same qualities of shrewd judgment from the supreme authority that characterize the administration of a large commercial firm.

Without this elasticity, standardization can only be the forerunner of stagnation, and perhaps of dissolution. Sharpness of definition "growing both greater and more variously exemplified as societies advance to maturity, is extremest in those which have reached their full development or are declining." Ultimately "decay follows a fixity which admits of no adaptation to new conditions."

Standardization may be said to mark that point in engineering evolution when the indefinite incoherent homogeneity has become merged into the definite coherent heterogeneity and a sectional condition of equilibrium more or less constant results. This process of equilibration tends ever to a greater completeness. "During early stages of social evolution, while the resources of the locality inhabited are unexplored and the arts of production undeveloped, there is never anything more than a temporary and partial balancing of such action. But when a society approaches the maturity of that type on which it is organized, the various industrial activities settle down into a comparatively constant state. Moreover, advance in organization, as well as advance in growth, is conducive to a better equilibrium of industrial functions. While the diffusion of mercantile information is slow and the means of transport deficient, the adjustment of supply to demand is very imperfect. Great overproduction of a commodity is followed by a great underproduction, and there results a rhythm having extremes that depart widely from the mean state in which demand and supply are equilibrated.

"But when good roads are made and there is a rapid diffusion of printed or written intelligence, and still more when railways and telegraphs come into existence—when the periodical fairs of early days grow into weekly markets, and these into daily markets, there is gradually produced a better balance of production and consumption; the rapid oscillations of the price within narrow limits on either side of a comparatively uniform mean indicate a near approach to equilibrium. Evidently this industrial progress has for its limit that which Mr. Mill has called 'the stationary state.' When population shall have become dense over all habitable parts of the globe; when the resources of every region have been fully explored, and when the productive arts admit of no further improvements there must result an almost complete balance, both between the fertility and mortality in each society and between its producing and consuming activities. Each society will exhibit only minor deviations from its average number, and the rhythm of its in-

dustrial function will go on from day to day and year to year with comparatively insignificant perturbations."

Before the sectional equilibrium is merged into universal equilibrium, many ages must elapse, while yet longer periods will intervene before the ultimate dissolution, which is a matter for philosophic speculation, will confront our descendants. So far as standardization is concerned, we are only on the threshold of a great effort to give uniformity and efficiency to manufacturing conditions. Its happening has been as inevitable as the evolution of the law of the civilized state from the traditionary rules of uncivilized communities. The standardized uniformity of the future must maintain a certain flexibility, or else it will be swept away by the pent up forces of a delayed progress.

But of this there seems little fear. The Anglo-Saxon race has always been eminently individualistic, and so long as standardization does not involve the suppression of the improvements which the individual thinker or individual manufacturer offers to the community, standarization will merely mean law and order in engineering design and installation, but law and order subject to further evolution and continuous modification. Nationally, the immediate advantage of standardization is great, for improved commercial efficiency has already been attained.

While there is a faint danger element that standardization may delay progress, just as the inertia of established industries tends to preclude a change in our systems of weights and measures for all save absolutely new branches of industry, the action of the committee in having revised certain details in their original standard specifications for Whitworth standard threads, Portland cement, and of ingot steel forgings, copper conductors, and material used in the construction of railway rolling stock is more than fair proof that this spirit of *laissez-foire* is not to be dreaded.

When in addition to this it is remembered that a report was issued in 1905 on the "effect of temperature on insulating materials" it will be fully recognized that such an administrative body has great potentialities in the incitement of the technical and scientific research and progress of the Empire whose evolution from "the relatively indefinite incoherent homogeneity to a relatively definite coherent homogeneity" calls to-day for constructive's statecraft of the highest order.

London, England. TURRIS.

Notes on Chemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

The Manufacture of Cyanamide in France.

The production of cyanamide has been started by the Société Commerciale de Carbure et de Produits Chimiques near Notre Dame de Briançon. The works are adjacent to the Metal Carbide Company's factory, whence the supply of carbide is conveniently obtained; and current is supplied from the Carbide Company's power house, about 11 km distant, at 15,000 volts, three-phase, 42 cycles. The nitrogen is furnished by the Linde method from air freed from carbon dioxide and moisture, with Burckhardt compressors operating in three stages. Carbide is received in large masses, which are broken up by stamps and powdered in a ball mill. The furnaces, of which there are 30, have a capacity of 300 kg each, and the present output of cyanamide is about 10 tons per day, which amount is capable of being doubled with the space and power now available.

Electrical Smelting of Iron Ore.

A British consular report from Stockholm states that the Swedish installation for electrical smelting at Trollhattän is progressing satisfactorily, and that experimental work with the old furnace at Donnarfvet is again going on. As a consequence, it has been officially announced that the results are superior to those from earlier efforts, and the superintendent is of opinion that a definite solution of the problem of replacing the blast furnace by electrical smelting has been worked out.

From a report of the British Legation at Christiania it appears that an experimental furnace at Tinnfos is turning out iron at a cost of £3 per ton practically equal in quality to best Swedish charcoal iron which costs £5 to £5 10s. per ton. At present the output is only three tons per day, but it is proposed to increase the power to 100 hp and turn out 25 tons per day, and also to proceed with the production of steel, at an estimated cost of £4 10s. per ton as against about £10 per ton for charcoal steel of the same grade.

Portland Cement.

The British standard specification for Portland cement has again been revised by the Engineering Standards Committee. The outcome of the committee's inquiry as to the determination of the time of initial setting was that a discrepancy was found between the results obtained with the British standard needle and those with the Vicat needle, although the times of final setting agreed fairly closely. It was considered advisable that the specification should direct the use of only one instrument for determining both the initial and final times of setting; and, as the Vicat needle is pretty generally used, the committee decided to adopt it. With regard to soundness tests, the question of providing against expansion of the cement in cold water was negatived, as the investigations showed that results of tests of the plunging order depended on the time of setting more than on the soundness. As to chemical composition, a *minimum* for total lime has been laid down as a provision against the admission of cements which are not "Portland." The loss on ignition and the total sulphur (sulphides as well as sulphates) have also had their limits specified. The directions for gaging in paragraph 6 of the first revision were not very precise; and these have been altered to the extent of defining the requisite consistency of the gaged cement. The increase of tensile strength of briquettes of mixed cement and sand for longer periods than seven days is now more explicitly stated in the same way as that for neat briquettes.

Unless the 1/16-in. square needle weighted to 2½ lb. had points of superiority over the Vicat needle in determining the time of final setting it may safely be assumed that the former would never have been specified by the committee; and the reasons now given for discarding an instrument which the Standards Committee had selected as the most suitable will hardly commend the change to the large section of practical cement users to whom the time of initial setting is a negligible quantity compared with the time for complete setting. The B. E. S. needle would give determinations of the initial setting time quite as consistently as the Vicat; and determinations of this ill-defined point by different operators with either instrument might vary as widely as the results with both.

Researches published early in 1909 on the influence of sulphates on the deterioration of cement work indicated that sulphates exercised a much more deleterious effect than was commonly recognized, especially in submerged work; and the existing limit for sulphuric anhydride might be materially reduced with advantage.

The British Association.

The program for the forthcoming meeting at Sheffield includes the following papers: *Section A* (Mathematical and Physical Science)—"Atmospheric Electricity," by Dr. C. Chree; report of Gaseous Explosions Committee. *Section B* (Chemistry)—"A Fourth Recalescence in Steel," by Professor Arnold; "Influence of Chemical Composition and Thermal Treatment on the Properties of Steels," by Prof. A. McWilliam; "Ferrosilicon," by Dr. S. Monekton Copeman; "Corrosion of Iron and Steel," by Dr. J. N. Friend; "Crystalline Structure of Iron at High Temperatures," by Dr. Rosenhain; "Allotropy or Trans-

mutation?" by Professor Howe. *Section G* (Engineering)—"The Testing of Lathe Tool Steels," by Professor Ripper; "Optical Determination of Stress," by Professor Coker; "Heat Insulation," by F. Bacon; and report of the Gaseous Explosion Committee, joint discussion with Chemical Section.

The Mallet-Boulouvard Ammonia Soda Process.

This process as carried on by the Compagnie Générale des Produits du Midi at Sorgues has been described in detail and somewhat adversely criticized by M. K. W. Jurisch, who states that while the estimated daily production is actually reached, an increased output cannot be attained when required to the same extent as in other methods. The system of filtration was good but expensive. The cost of calcination was too high, and another calcination process would be preferable. The factory cost a million francs and the production is 10,500 kg of sodium carbonate per day. A Solvay plant to turn out 20,000 kg to 25,000 kg per day could be added for 250,000 francs.

Market Report.

AUGUST, 1910.

Copper rose to £55.10.0 on the 3d, touched £56.7.6 on the 17th, then dropped to £55.8.9 and at the end of the month is firm at £55.15.0.

Tin mounted steadily from £151 on the 3d to £154.12.6 on the 17th and after a slight decline in the following week was bulled to £163 on the 31st.

English lead participated to a slight extent in the general improvement during the first week, but has since remained steady at £12.17.6.

Iron, Hematite, after easing to 64/- recovered to 65 3 on the 24th and remains at that figure.

Scotch Pig, with a better inquiry, went up to 56/- at mid-month, but has declined gradually to 55/6½.

Cleveland Warrants have been firmer, and reached 50/- on the 17th, but have declined again to 49. 5.

Alum, lump, loose, per ton	£5.12. 6.
Antimony, black sulphide, powder, per ton	24. 0. 0.
Borax, British refined Xtl, per ton	16. 0. 0.
Sulphate of ammonia, per ton	11.16. 3.
Sal ammoniac, firsts, delivered U. K., per ton	42. 0. 0.
Copper sulphate, per ton	18.12. 6.
Caustic soda, ash, 48 per cent, ordinary, per ton	5.10. 0.
Bleaching power, 35 per cent, per ton	4. 5. 0.
Sulphur, per ton	5. 0. 0.
Litharge, flake, per ton	15. 0. 0.
Red lead, genuine, per ton	15.10. 0.
Shellac, T. N. orange, spot, per cwt	4. 4. 0.
Hydrochloric acid, per cwt	5. 0.
Copper ore, 10 to 25 per cent, per unit	9 - to 10. 0.
Tin ore, per ton	£95-97. 0. 0.
Carbolic Acid, liquid, 97 to 99 per cent per gal	11½.
Creosote, ordinary, good. liquid, per gal	2⅛.
Naptha, solvent, 90 per cent at 160 deg. C., per gal.	1. 1.
Camphor, 1-oz. tablets	1. 7.
Mica, small slab, original cases, per lb	6d. to 1. 0.
Platinum, per oz	6.15. 0.

India rubber declined to 8/- on the 10th; recovered to 9/7 on the 17th, and has steadily fallen to 7/11 on the 31st.

London, August, 1910.

Synopsis of Metallurgical and Chemical Literature

Gold and Silver.

A Comparison of Slime Treatement Methods, filter pressing compared with decantation and the Adair-Usher process, is outlined in the April, 1910, issue of the *Journal* of the Chemical, Metallurgical and Mining Society of South Africa, by JOHN D. O'HARA. The author took charge of a plant at which filter presses had been installed for the treatment of accumulated slime. The system of treatment included the agitation of the slime with neutralizing agents, after which it was run by gravity to a Montejus, whence it was sent to the presses by compressed air. Here it was washed with the requisite cyanide solutions and final water wash, after which air was blown through the presses to dry the cake before discharging. This treatment was successful, but the working cost was too high. The average extraction from residues of irregular value was 88 per cent. The solution obtained from the presses was not clear enough to go at once to the precipitating boxes and had to be filtered in large vats. The causes of failure of the press plant are given as: 1. Cost of filter cloths. 2 Heavy consumption of cyanide and lime. 3. Cost of handling slime from dam to dump. 4. Small capacity of the presses, three in number, the usual quantity handled being 48 tons to 56 tons per 24 hours.

Decantation was then tried and the slime was pumped from the decantation plant to the agitators and thence to the filter presses, with the idea of recovering the gold in solution and reducing the quantity of undissolved gold. But this did not pay the cost of handling, and it was cheaper to discharge the slime to the dam and pump back the solution. Then the experiment was tried of using the presses to clarify the solution decanted from the slime and this proved successful and profitable. The extraction gained was 85.7 per cent.

Later the Adair Usher process was installed, with the result that the extraction was improved, at the beginning amounting to 87.6 per cent and later rising to 90.3 per cent. The author regards the Adair-Usher process as an effective means of washing slime and displacing the gold solution. Waste steam is used at this plant for heating the slime solutions, especially in the winter, as the heating is found to aid quick settling.

Copper.

Data on Copper Converting in Australian smelters are contributed to the *Australian Mining Standard* of June 1 and 8, 1910, by Mr. H. SCHRODER. The largest converters in use are 10 ft. 6 in. by 7 ft. 6 in. With these vessels one ton of blister copper can be produced per hour from 45 per cent matte, blowing only one vessel at a time, with 3000 cu. ft. of air per minute at a pressure of 9 lb. per square inch. With 35 per cent matte the blister is reduced to 0.75 ton per hour. The supply of air should preferably be 4500 cu. ft. per minute at a pressure of 10 lb. to 12 lb. per square inch. The large converters take an initial charge of 6 tons of matte, and a charge of 10 tons to 12 tons on the sixth, seventh and following blows. With 45 per cent matte they can average 27 tons of blister per converter. The best run known to the author was 36 tons of blister, the converter handling 11 charges before relining. The data on a month's run, with one converter in blow at a time, are as follows: Average blows per converter, 8.4; average blister per converter, 27 tons; blister per hour, 1.17 tons; blister copper produced, 216.6 tons; matte treated, 460 tons, together with 6 tons copper refinery slag and 25 tons white metal.

Grade of Matte.—The author finds the most suitable grade of matte to be 45 to 48 per cent copper, although higher grades, such as 65 to 70 per cent matte, may be converted successfully if alternated with the 45 per cent grade. The worst grade of matte to deal with is 55 per cent copper, as it does not form enough slag to skim properly, and yet there was too much to skim with a rabble, the slag granulating and forming an un-

desirable product. The average copper produced from 46 per cent matte was 1.17 tons per hour; from 66 to 70 per cent matte, 1.8 tons per hour.

Converter Lining.—Three mixtures are used for tuyeres, body and cover. For tuyeres, 68.5 to 70 per cent SiO_2; for body, 80 to 82 per cent SiO_2. The lining mixtures are made up of silicious material, clay and old lining. Following are the quantities used:

	Silicious Ore.	Clay.	Old Lining.
Patching mixture	1	1	..
Cover mixture	1	1	2
Body mixture	5 to 8	1	1
Tuyere mixture	2 to 3	1	..

If clean, old lining can be used alone for cover mixture, and it is advisable to use as much old lining as possible throughout the converter. As high as 6 parts of old lining, 5 of silicious ore and 1 of clay has been used, but such linings do not suffice for the production of more than 16 tons of blister. The author states that the method of lining has a great deal to do with the way in which a converter stands up to its work, and that within certain limits of silicon content the method of lining the converter is more important than the chemical composition. The ramming must be carefully done, the back or tuyere side being brought up straight to the throat, and the front only being tapered to the throat, which is finished to a diameter of about 2 ft. The converter is dried out with a wood and coke fire for 20 to 30 hours, and is at a dull red heat when it is put in use.

Blowing.—No fixed rules can be laid down regarding the slagging and finishing points of the blow. Mattes of different composition give different flame colorations, varying from heliotrope to green when the slagging point is reached. The finishing points are even more varied than the slagging points, but the color at slagging points can be depended upon when using the same matte. The author describes in detail the various changes that can be expected in the course of the blow to slag and to copper, including the phenomenon noticed when the charge is overblown or burnt.

The cost of converting 45 per cent copper matte in the large vessels described, producing 1 ton of blister per hour, one vessel only being in operation at a time, is given as follows: Cost of power, 1.1 cent (0.55d.) per kw-hour; silicious material, $1.50 (6s.) per ton; fireclay, $3 (12s.) per ton; retreatment charges, 37 cents (1s. 6d.) per ton for slag, and $2.50 (10s.) per ton for old lining. On this basis the converting costs $4.58 (18s. 4d.) per ton of blister produced; relining, $2.97 (11s. 11d.); retreatment charges, 87 cents (3s. 6d), or a total of $8.42 (£1 13s. 9d.) per ton of blister produced. In the smaller converters the cost is from $2.50 to $3 per ton more. If the grade of the matte is but 35 per cent copper the cost in the large converters is $11.45 per ton of blister. If copper is produced in reverberatory furnaces, the cost is about $17.31 per ton and the product contains but 97 per cent copper as compare with 98.5 per cent copper in the blister produced from the converter.

Zinc.

The Chemical Treatment of Zinc Ores with the sulphurous acid evolved in the roasting of the same is reported to be in successful operation at Swansea, according to the *Mining Journal* (London) of July 16, 1910. In our current review of metallurgical patents will be found the specifications of the United States patent granted on this process. The ore is roasted, leached in a sulphurous acid solution, forming bisulphite of zinc, which is precipitated by passing steam through the solution, driving out the excess of SO_2. The precipitated sulphite of zinc is then roasted to zinc oxide. When a complex sulphide is treated the products are: (1) Zinc oxide practically free from lead; (2) the filtered residues carrying lead and any of the precious metals that may have been in the ore; (3) sulphurous acid which has been put in excellent condition for the manufacture of sulphuric acid. It is reported that a large tonnage of zinc-lead ore has been treated by this process with a recovery of 89 per cent of the zinc content. Cost is reported to be 75 cents (3s.) per ton, but the conditions are not given.

Tin.

Tin Dressing in Tasmania at the Mount Bischoff mine is described in detail by J. D. Millen in the *Mining Journal* (London) in the issues for Aug. 13 and 27. The ore treated has an average tin value of 0.54 per cent Sn. It is first crushed in stamps weighing 562 lb. each. The dies wear at the rate of about ½ in. per month for a duty of 150 tons. The shoes wear at the rate of 1.2 oz. per ton of ore treated. The stamps drop 88 times per minute through a height of 8 in., and have a duty of 7.5 tons each per 24 hours. After stamping the pulp is classified in hydraulic spitzlutten, of which there is a pair to each 10 head of stamps. The overflow goes to settling spitzkasten and the spigot discharges to coarse and fine jigs. The jigs are of the Harz type and make finished hutch products. The jig for coarse material has 200 pulsations per minute and a stroke of 5/16 in. The fine jig has 250 pulsations and a stroke of 3/16 in. The former handle 10 tons per day and the latter 13 tons. The jig tailings are conveyed to classifiers and thence to Card and Wilfley tables.

The overflow from the first spitzlutten mentioned above is settled as described and the finer sand is treated on Luhrig vanners and round tables, and the coarser sand on Card and Wilfley tables. The round tables are 16 ft. in diameter and have a slope of 5.5 degrees. They make one revolution in two minutes. The Card and Wilfley tables have a feed from the tailing of the jigs, the coarse sand from the spitzkasten and the middling from the round tables. The fine jigs followed by the Card tables make a recovery of 86.7 per cent of the total tin content drawn from the No. 2 spigot of the spitzlutten. The Wilfley tables and the fine jigs make a recovery of 86.06 per cent Sn.

The Luhrig vanners are of the duplex type; slope, 4.5 degrees; 150 bumps per minute; belt travel, 6 ft. per minute. They receive a product which all — 100 mesh and handle about 18 tons per day on both belts. The pulp fed assays 0.56 per cent Sn, and the tailing assays 0.03 per cent Sn. The middling from the vanners is treated on Sperry vanning tables. These tables are adjusted to have a circular vanning motion of 200 throws per minute, 0.3 in. in length; revolution of the table is once in 1.25 minutes; slope, 5.5 degrees.

The slime plant consists of rotary and Crad tables, and the procedure here is practically the same as described for the initial treatment. A new stamp mill of 10 heads of 1050 lb. each has been placed in operation recently, with an accessory equipment consisting of revolving screens, grinding pans and Card tables.

Grading.

Grading Analyses and Their Application.—The determination of the relative merits of different crushing appliances from grading analyses of their crushed products has been receiving considerable attention of late, particularly in connection with the controversy over the introduction of the heavy-stamp tube-mill combination in South Africa. In the May, 1910, issue of the *Journal* of the Chemical, Metallurgical and Mining Society of South Africa, Mr. H. Stadler proposes a new system of grading crushed products, based on the reduction of the volume (or weight) of the particles. The sizes of the author's proposed grading scale are in close agreement with certain of the standards established by the Institution of Mining and Metallurgy. On nine grades, from 1-in. mesh down to 30 mesh, the true average weight of the particles of each grade has been ascertained, and the ratio of decrease in weight of these particles proved to correspond remarkably well with the correct reduction scale of theoretical cubes. The relation

of the true average size of the particles to the corresponding theoretical cubes representing the mean grade between the grades of the screens varies in nine tests between 58.3 and 66.9 per cent, and the average of 61.4 per cent can, therefore, be considered fairly representative for Rand ore and for a product coming from breakers.

The author claims for his system of standardization of grades the following advantages:

1. No really new departure from standard practice.
2. Equality of steps with regard to reduction in volume.
3. A sufficient number of grades.
4. Independence of wire gage, and, therefore, immediate applicability to screens of any description with round or square holes.
5. The standard *grades* are exactly the same for screens in practical use in metallurgical operations as for laboratory purposes, while the *screens* may vary in accordance with the gage of the wire used, with degree of accuracy in their manufacture.
6. No limit to extension in either direction or by interpolation.
7. The variation of all other functions of the particles, besides diameter, can be expressed by equations or plotted in regular curves.
8. Any odd screen of which the clear aperture is known can readily be inserted in the scale by use of a formula deduced by the author.
9. The ordinal numbers represent at the same time the relative value of the energy required to produce the respective grade from the unit.
10. The average values of screen products determined by possibly far distant screens are readily obtained.
11. Any projected standardization of battery screens can easily be made to fall in with the present scale.
12. The so-called 60-mesh and 90-mesh screens (aperture 0.01 in. and 0.006 in. respectively) are already in general use on the Rand.

The author presents the theory of the application of grading analyses to the calculation of the relative mechanical efficiency of crushing machines, and gives two examples of calculations for the stamp and tube mill. He concludes that we are now in a position to determine with a comparatively high degree of accuracy the relative merits of different crushing appliances, or the mechanical efficiency of one and the same machine working under varying conditions.

Chemical Engineering.

Fuming Sulphuric Acid.—The old method of testing the strength of fuming sulphuric acid by titrating with a normal alkali solution is not considered satisfactory for technical use by Mr Henry Howard, who has presented to the Society of Chemical Industry the details of a new method. The principle involved is the measurement of the heat developed by combining the free sulphuric anhydride with water. As this heat value is very great, a small variation of the percentage of anhydride makes large variations in the heat produced. To eliminate the heat of dilution of sulphuric acid, the author uses sulphuric acid of certain concentration instead of water with which to mix the fuming acid. When a 20 per cent to 25 per cent fuming acid is to be tested he uses a sulphuric acid of 92 per cent H_2SO_4 as his source of water. A Dewar vacuum tube is used for the work.

One hundred grams of the fuming acid to be tested are weighed into a Dewar tube. The temperature (a) is noted, and then temperature of the reagent acid, after which 100 g. of the 92 per cent acid are poured into the Dewar tube and mixed with the fuming acid. The temperature (c) of the mixture is recorded. The heat of reaction equals $(c - \frac{a+b}{2})$, making an allowance for the heat capacity of the tube. In order to avoid corrections for this or other slight errors, tests are made in this manner with fuming acids of known composition, and from these results the following table is compiled:

Temp. deg. F	Free SO_3.	Temp. deg. F.	Free SO_3.
72.0	21.0	87.6	23.6
73.2	21.2	88.8	23.8
74.4	21.4	90.0	24.0
75.6	21.6	91.2	24.2
76.8	21.8	92.4	24.4
78.0	22.0	93.6	24.6
79.2	22.2	94.8	24.8
80.4	22.4	96.0	25.0
81.6	22.6	97.2	25.2
82.8	22.8	98.4	25.4
84.0	23.0	99.6	25.6
85.2	23.2	100.8	25.8
86.4	23.4	102.0	26.0

The table shows a difference of 6° Fahr. for each per cent increase in anhydride, and as the readings can be depended upon to within ½° Fahr., the results obtained are extremely accurate.

The same principle can be applied to the determination of the strength of ordinary sulphuric acid between 95 per cent and 100 per cent. The reagent used is fuming sulphuric acid and it is used in excess, so that the heat generated represents the uncombined water in the regular sulphuric acid. The same formula applies.

Sulphate of Ammonia.—The precedure at the Dumfries Gas Works (Scotland) was outlined in a paper read before the Western District Division of the Scottish Junior Gas Association, at Glasgow, by Mr. J. H. Smith. After describing the plant for the manufacture of ammonium sulphate, the author gave the following résumé of costs:

	Per ton of salt.
Acid and oil, 1700 lb.	$7.43
Lime, 170 lb.	.30
Wages	3.52
Cartage	.14
Steam and water	.75
Liquor, 11 tons, 1630 lb., 2.1 per cent.	24.37
Interest on loan at 3½ per cent.	2.85
Depreciation	3.25
License	.11
	$42.72

The average price of the salt for the year 1908-9 was $53.03. The sulphate produced per ton of coal was 25.5 lb.

Suction Gas Plants are increasing in use in every part of the world, and are apparently finding favor with power users. In western Australia there are several types of producers and engines in use at mines, mills and industrial plants of every description. In the July 30, 1910 issue of the *Mining Journal* (London), Mr. M. von Bernewitz gives some details regarding these plants, having secured his data from personal inspection. Charcoal is the fuel generally used, as the coal available produces too much tar, and apparently the tarless producer has not been developed there as it has in the United States. Both steam and electricity have been superseded by gas in many instances; two cases where steam was abandoned in favor of gas showed savings of $350 to $400 per month in fuel alone. The average consumption of charcoal per b.h.p. is 1 lb., while for coal (when used) it is 1 lb. to 1½ lb. Most of the plants are small, i.e., under 150 hp. Most of the engines are built with two heavy flywheels, with the driving pulley on one side overhanging. The author regards this as a mistake in construction, and found many managers of plants to agree with him. There should be an outside bearing to relieve the crankshaft. The author found it difficult to arrive at definite costs, but believes that 2 cents (1 d) per b.h.p. may be said to be a fair average cost

Following are some data on actual installations: A 104-hp mill plant, using about 86 hp, fuel consumption per hp-hour, 0.9 lb.; cost per hp-hour, 1.488 cents (0.744 d.). Another plant using 70 hp showed a cost of 1.64 cents (0.82 d.) per hp-hour. This was favorable as compared with 5 cents when electricity was used. At still another milling plant the comparative costs for steam and gas power were $550 and $100 per month, respectively. Other data are given on various kinds of plants.

Recent Metallurgical and Chemical Patents

Iron and Steel.

Manganese Steel.—A manganese steel ingot, devoid of dendrites and other objectionable freezing structures and of a strong uniform fine-grained structure throughout, for rolling and forging, is produced by a combination of chemical and heat treatment devised by Mr. Winfield S. Potter, of New York City. The first step is chemical and consists in the preliminary deoxidation and cleansing of the metal. The manganese steel is compounded from dephosphorized and desulphurized molten steel from an open-hearth or other furnace and molten ferro-manganese, mixed in a casting ladle of large capacity, say 60 tons or more. The ladle is acid-lined and a layer of acid slag (produced from silicious material and fluxes, like sodium silicate, lime or lead oxide) protects the metal in the ladle from surface oxidation. The proportion of silica should be slightly more than sufficient to balance the oxides in the steel plus the small amount of basic furnace slag which may have come into the ladle. The acid-slag-making materials are charged into the ladle either before or during the charging of the molten steel in order to obtain such an intimate contact of the acid slag formed and the metal as to insure the washing out of oxides from the steel. The metal in the ladle contains about 12 per cent manganese and from 0.8 per cent to 1.1 per cent carbon. The temperature of the metal in the ladle during the cleansing operation is from 1400 deg. C. to 1450 deg. C. The metal, with its protecting acid-slag layer, is permitted to rest in the ladle long enough to insure separation of slag and metal and to cool to about 1375 deg. C., when it is poured through a bottom pouring aperture. While the ingot is still in the mold it is cooled in such a manner that when stripped from the mold the outer portions (to a depth fully including the thickness of the zone of the dendrites) will have an average temperature in the neighborhood of 1000 deg. C. (say from 950 deg. C. to 1050 deg. C.) except at the edges, where it will be unavoidable considerably lower. This enables it to be transferred without suffering any rupture into a soaking pit, where it is to be subjected to the heat treatment designed for reconstructing its exterior portions; and it also permits this heat treatment to take place without cracking the ingot. At the time of its entrance into the soaking pit the ingot has set and developed its freezing structures to a depth of, say, 6 in. or more inwardly from its outer skin or periphery and a zone of dendrites has formed which usually occupies, say, about one-half of this depth. In order to break down these dendrites and to substitute therefor a strong uniform structure of fine grain the temperature of the soaking pit is slowly raised to 1175 deg. C. or a somewhat higher temperature, this being effected in a smoky or reducing atmosphere, so that the periphery of the ingot will not be oxidized. This requires a comparatively short period of time, say half an hour. This stage of the heat treatment is then arrested by lowering the temperature so as to prevent the building up within the mass of larger and coarser grains. To this end the temperature prevailing within the soaking pit or other heating furnace may be lowered by the injection of water vapor, in the form of a steam jet carrying with it water in suspension, with the result that the periphery of the ingot is reduced in temperature to, say, 1100 deg. C.; or for the same purpose the ingot may be cooled by transferring it into a cooler soaking pit. The lowering of the temperature of the exterior portions of the ingot is accompanied with the formation of a tough skin, which shrinks upon and works the still plastic interior. Finally, after the cooling of the outer portions of the ingot has occurred and the steam jet has been cut off the outer portions of the ingot are permitted to rise again in temperature to, say, 1125 deg. C., in order to equalize the surface temperatures and to obtain the desirable maximum ductility, whereupon the ingot may be removed from the soaking pit and is in condition to be immediately rolled into rails or other products in mills of ordinary construction. Moreover, the temperature conditions are such that a track rail, for instance, of standard length and section, rolled from such an ingot will be delivered from the final pass of the rolls at a temperature which will permit it to be toughened by quenching without being first subjected to a reheating operation. The cooling temperature of 1100 deg. C. and the reheating temperature of 1125 deg. C. are suitable for manganese steel containing 1.10 per cent carbon or less. When the carbon is higher, say 1.20 per cent or more, somewhat lower cooling and reheating temperatures are employed.

The process can be changed in its later stages, as an alternative way of toughening the outer portions of the ingot is available instead of subjecting it to cooling in the soaking pit. The ingot may be taken from the soaking pit, in which it has been slowly and evenly heated to a temperature of from 1175 deg. C. to 1225 deg. C., depending upon the analysis of the metal, and is then subjected to light mechanical working over its periphery for the purpose of producing a wrought thick skin thereon. The lower temperature (1175 deg. C. or thereabout) is preferred when the amount of carbon present in the ingot is above 1.20 per cent; and the higher temperature (1225 deg. C. or thereabout) is preferred when the amount of carbon present is 1.10 per cent or less, it being assumed in both of these instances that any phosphorus present is low in amount. Let us assume that the manganese steel ingot is a 20-in. x 20-in. square section. The light preliminary rolling would then be effected by acting first simultaneously upon two horizontal sides and then turning the ingot and acting simultaneously upon the two remaining sides. Where the analysis of the manganese steel of the ingot shows it to be low in carbon (say, containing 1.10 per cent of carbon, or somewhat less than that amount), the reduction in the first one or two passes of the preliminary light rolling may be about 1 in. in each pass. This first reduction along the two horizontal sides of the ingot renders the metal of those sides capable of greater elongation, and consequently when the ingot is turned over and rolled along its opposite sides, a somewhat greater reduction can take place, say 1½ in. or 1¾ in. These two preliminary reducing passes will in general suffice for the light preliminary rolling intended. When the manganese steel has a harder analysis, i. e., when the percentage of carbon is as high as 1.20 per cent or higher, it will usually be advisable to limit the first reductions of the series of light preliminary passes to from ½ in. to ¾ in. This welding or consolidating of the outer portions of wrought skin or steel is of particular importance where the reduction contemplated in the rolling mill is considerable for the initial passes. It is carried out quickly, so that there may be no material drop in temperature and the ingot is then delivered to the rolling mill and rapidly reduced in the usual manner. (968,601, Aug. 30, 1910.)

Alloy Steels.—For the manufacture of that class of alloy generally known as high-speed steel, Mr. Jesse M. Darke, of Lynn, Mass., has patented a formula and procedure which has

been assigned to the General Electric Company. The components of the alloy are Swedish muck-bar, ferro-tungsten, ferro-chromium and manganese. The first three form the basis of the steel and are melted in a crucible in the usual manner. The ferro-tungsten and ferro-chromium are of the electric furnace variety and contain about 67 per cent tungsten and 68 per cent chromium, respectively. Sufficient quantity of each is used to give the finished alloy about 13.5 per cent tungsten and less than 0.5 per cent chromium. If these components are too dry to form a slag some slag is added and when the steel is nearly ready for pouring sufficient manganese is added to give the finished steel about 3 per cent manganese. The manganese is added in a special manner so that it will not combine with the slag which has a strong affinity for the same. A steel or iron tube is inserted into the molten mass in the crucible and the manganese inserted through this tube. The manganese being in lumps the size of a walnut, is soon dissolved in the alloy, and in about five

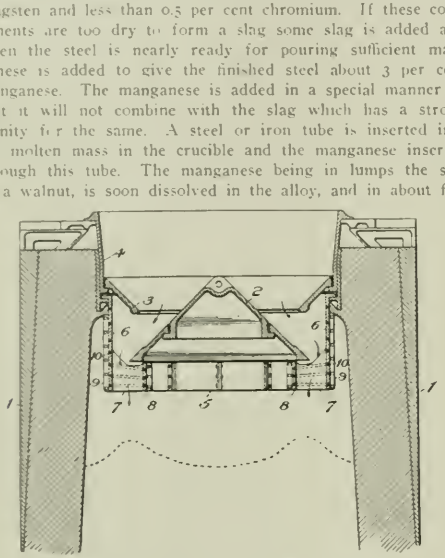


FIG. 1.—BLAST-FURNACE CHARGING APPARATUS.

minutes the charge may be poured. For certain purposes a small amount of vanadium may be also added. (967,335, Aug. 16, 1910.)

For Making Steel Castings from Steel Scrap in an ordinary foundry cupola Mr. HUGH F. JONES, of Los Angeles, Cal., charges the cupola with alternate layers of coke and flux and steel scrap, using the following ratio: for 48-hour coke, 7 lb. of steel to 1 lb. of coke; for 60-hour coke, 9 lb. of steel to 1 lb. of coke; for 72-hour coke, 10 lb. of steel to 1 lb. of coke. The smallest pieces of scrap are placed on the coke and the largest pieces are reserved for the top layer. Coarse coke is used and the pieces of scrap are preferably not larger than 25 lbs. If the castings are to be machined, open-hearth steel is used, but if no machine work is required, 75 per cent of Bessemer scrap and 25 per cent of open hearth scrap can be used. The flux used is known as Kirk's Compound, consisting principally of silica and iron oxide. It is used in the ratio of 10 lbs. of the compound to each ton of scrap when the coke contains 2 per cent sulphur. For each 1 per cent of sulphur over or under 2 per cent, 1 lb. of flux is added or deducted. The blast used on the furnace depends on the percentage of fixed carbon in the coke; for 70 per cent fixed carbon, an 8 oz. pressure is used and for each 5 per cent fixed carbon above or below 70 per cent the blast is reduced or increased 1 oz. The molten steel is tapped into a ladle containing a small quantity of aluminum and ferro-manganese (967,780, Aug. 16, 1910.)

Blast Furnace Charging Apparatus, designed to afford a more even distribution of the stock in the furnace, has been patented by Mr. GEORGE L. COLLORD, of Birmingham, Ala. Any distribution of stock which allows the fine material to flow to the furnace walls prevents uniform action throughout the charge and many disadvantages result. According to Mr. Collord's device, which is shown in one form in Fig. 1, the charge is caused to pass between deflecting walls suspended in the furnace below the bell so that the charge falls substantially in an annular body and gives a double wave toward the center and sides of the furnace, as shown in the dotted line. In the furnace 1 the bell 2 closes the opening in the charging extension 3 of the hopper 4. The stock distributer 5 is supported in the furnace, as shown by straps 6. The inner deflector is a concentric wall 8 supported by radial webs from the outer deflector and both deflectors and webs are water cooled. The inventor describes a number of forms of his invention which are not shown here. (967,328, 967,329, Aug. 16, 1910.)

Metallurgical Heating.—For the purpose of heating steel and steel alloys to high temperatures, even melting them, without oxidation of the valuable constituents, a type of furnace has been devised by Mr. BYRON E. ELDRED, of Bronxville, N. Y. (An article by Mr. Carleton Ellis on the methods of flame regulation and combustion, known collectively as the Eldred process, was given in our vol. II, p. 895.) The furnace is of the regenerative type in which gas and air previously heated may be mixed and burned over the hearth of the furnace. In the present invention the process is modified so that a stratum of burning gas and air is produced along the roof of the furnace, while another stratum of gas only is passed along the hearth of the furnace, spacing away the burning stratum from the material being heated, so that oxidation is prevented. The furnace has twin regenerative chambers on either side of the hearth. A current of gas entering from one hot regenerator is passed over the hearth on which are placed steel billets or other steel articles to be heated. A current of air entering from a similar hot checker work is divided into two fractions, one being sent along the roof of the furnace, where it burns a portion of the gas, forming a flaming stratum hugging the roof of the furnace; the other portion being by-passed around the furnace and completing the combustion of the gas issuing therefrom in the regenerator, which is in heating-up phase. Thus the material on the hearth of the furnace is heated by radiation through a protecting and reducing atmosphere of gas.

The process is susceptible of the reverse action, in case an oxidation effect should be desired on the material on the hearth, such as in the roasting of sulphide ores. In such a case the hot air would be passed over the hearth and the hot gas would be divided into portions, one being sent along the roof of the furnace where it would burn with a portion of the air, and the other being by-passed around the furnace to burn with the balance of the air in the regenerative chambers. Thus the material on the hearth of the furnace would have a hot layer of oxidizing material next it while being spaced away from the burning layer of gas with its products of combustion.

The inventor also calls attention to the fact that by using a gas layer next the hearth, iron ore may be reduced to form iron and steel direct. The necessary carbon or metallic alloy may be added without fear of oxidation and the iron can be reduced in contact with a gas containing the minimum amount of carbon dioxide, such as in producer gas, while being subjected to as high a temperature as desired by the flaming stratum above. (968,238, Aug. 23, 1910.)

Gas Scrubber.—In a patent of ALFRED STEINHART, of Pittsburgh, Pa., the latest mechanical construction of his well-known gas-scrubber tower for cleaning blast-furnace gas is described. The scrubber consists of a metal shell, 50 ft. or 60 ft. high, through which the gas to be cleaned passes upward, while water passes downward. The water is discharged from a nozzle at the lower end of a pipe at the top of the tower. When the nozzle is stationary the water is discharged in a fan like sheet, but when the water pipe and nozzle are rotated the fan like sheet assumes a conical shape. This conical flow of water impinges on a diaphragm built up of concentric rings, in the annular spaces between which thin metal strips or hoops are placed

so spaced as to allow the passage of a thin sheet of water between them. The effect of the diaphragm is to assure an approximately even flow of water through all its portions and to divide it up into uniform fine rain drops. The construction is such that the operation is absolutely continuous and that any repairs of the nozzle may be made without removing the nozzle or providing an opening for the escape of the gas from the scrubber. For the many carefully worked out details of construction the reader must be referred to the patent specification, which goes into the matter quite in detail. (965,248, July 26, 1910.)

Electric Arc Welding.—Articles of iron and steel are welded or repaired, according to H. L. J. Siemund, of New York, by establishing an electric arc between the article and a "welding rod," ⅜ in. in diameter, of soft iron coated by or, if hollow, containing the welding composition. This is made up as follows: One part of boracic acid, one part of pumice stone, one-half part of bicarbonate of sodium, one-half part of lampblack or of powdered charcoal, one-quarter part of chloride of calcium. These ingredients are mixed in the dry state and made into a thin paste by the addition of water so that it can be applied by a brush to the welding rod. (967,580, Aug. 16, 1910.)

Gold and Silver.

Charcoal Precipitation of cyanide or chlorine gold solutions is not extensively practiced save in Australia, where it is used in some large plants. The idea has been condemned on the score of being slow, cumbersome and costly as compared with precipitation on zinc. With a view to improving the process and making it faster, more convenient and cheap, Mr. Thomas Edwards, of Ballarat, Victoria, Australia, has designed apparatus shown in side elevation in the accompanying Fig. 2. It consists of a partitioned box similar to zinc precipitation boxes with provision for the flow of solution from one compartment to the next, passing alternately over

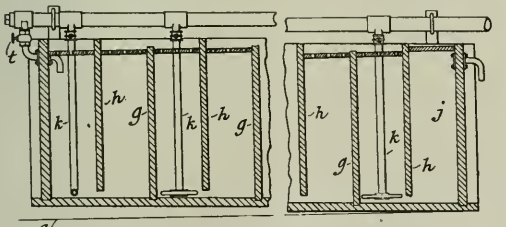


FIG. 2.—APPARATUS FOR CHARCOAL PRECIPITATION.

and under the partitions. Partitions are indicated in the drawing by *h* and *g*, the inlet of solution at *t* and the outlet at *n*. Perforated covers are provided for each compartment so that the solution may flow without carrying particles of charcoal. One of the novel features claimed by the inventor is the application of air through pipes *k* in some of the compartments, the aeration being claimed to be beneficial to precipitation. Removable gratings or cages may be provided for the expeditious removal or handling of the charcoal in each compartment. (967,744, Aug. 16, 1910.)

Precipitation of Aluminium.—The use of this metal to replace zinc as a precipitant for gold and silver in cyanide solution has been suggested from time to time by several writers, and is used in laboratory schemes for analysis of cyanide solutions. Stafford F. Kirkpatrick, of Kingston, Ontario, Canada, has patented the process for commercial application in an apparatus specially devised for the purpose. The most efficient form in which aluminium could be used for this purpose is as a powder, but powdered aluminium does not easily become wet by water and does not mix with solutions. In order to overcome this difficulty the inventor makes use of an agitating device to engulf the metal in the solution and thus causes it to react with the precious metal cyanide solutions. If the precipitation is carried out in a round tank with a conical bottom, a suitable agitator can be made by attaching propeller blades onto a vertical shaft revolving in the center of the tank. This will produce a vortex into which the aluminium will be drawn and thoroughly mixed with the solution. Sodium or potassium hydroxide is added to the solution, and the inventor gives the following equation as representing the reaction which takes place:

$$6NaAG(SN)_2 + 6NaOH + 2Al = 6Ag + 6NaCN + Al_2O_3 + 2H_2O.$$ (959,757, May 31, 1910.)

Zinc.

The Treatment of Zinc Ores containing zinc oxide with a solution of sulphurous acid so as to obtain a solution of bisulphite of zinc, is the subject of a patent granted to H. T. Durant, H. L. Sulman and W. Hommel, of London, England. The main features of the process consist in agritating a mixture of the ore and water with sulphurous acid gas in suitable apparatus, subsequently breaking up the zinc bisulphite formed by the injection of steam or hot gases to precipitate insoluble zinc sulphite, which may be heated to form zinc oxide and sulphurous acid gas, the latter being used in treating more ore. The agitation is carried on in an apparatus, one modification of which appears in Fig. 3. *K* is the mixing vat into which the injector nozzle *L* projects for the purpose of forcing gas or a solution of gas through the ore pulp. The agitated mixture rises through the pipe *M* into the chamber M^2, from which conduits M^3 conduct the mixture back into the vat. The operation is conducted under a slightly increased pressure to insure perfect action of the gases, but a gauge and safety valve are provided as shown so that the constantly accumulating pressure can be released when too great. When the agitation is complete the mixture can be drawn off at *N* and the solution treated as desired. The inventors also call attention to the application of their system to copper ores (966,389, Aug. 2, 1910.)

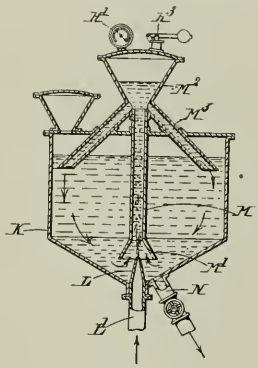


FIG. 3.—ZINC BISULPHATE APPARATUS.

Rare Metals.

Pyrophoric Alloys of the rare earth metals, such as cerium, lanthanum, didymium, etc., may have their pyrophoric properties greatly increased by subjecting them to the action of hydrogen gas at high temperatures, according to a patent recently issued to Adolf Huber, of Berlin, Germany. The proportions of the constituents of the alloy play an unimportant part as compared with the temperature required properly to incorporate the hydrogen gas therein. The requisite temperature must be determined for each alloy. For example, it has been determined that for an alloy of 15 per cent magnesium and 85 per cent cerium the right temperature is between 500 and 600° C. Furthermore, the duration of the exposure of the alloy to the hydrogen depends upon the alloy and must be determined for each composition. If the temperature is not properly chosen the alloy will crumble to pieces. The operation may be carried out in a muffle in which bars of the alloys are placed. The muffle is heated to the requisite degree of temperature and the gas is passed through as long as absorption is indicated. The action is complete when a fracture shows an even gray color throughout the alloy. The resulting product has a remarkable pyrophoric

power, giving off sparks in profusion and on the application of the slightest friction. (967,775, Aug. 16, 1910.)

Metallurgical Furnaces.

An Improvement in Reverberatory Smelting Furnaces is claimed for the design shown in Fig. 4, which embodies the ideas recently patented by Mr. F. W. Winkler, of Hancock, Mich. The object is to increase the area of the heating portion of the furnace chamber, to reduce the disintegrating action of the slag on the walls, to provide for ready repairing of the furnace walls and to increase the strength and durability of the furnace by providing adjustable compensation for

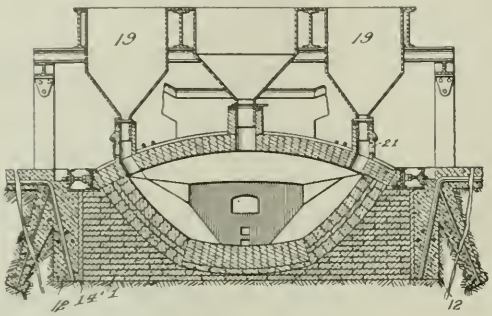


FIG. 4.—REVERBERATORY FURNACE.

expansion and contraction. It will be observed by reference to the cut that the bottom of the furnace and its side walls are curved on suitable arcs, giving a surface which the inventor claims will shed the slag to a great extent and thus reduce the well-known disintegrating action at the slag line. The furnace walls are supported by concrete walls 12, in which are embedded angle metal stays, as shown, inclining from the walls of the furnace. Between the furnace wall proper and the bracing wall is placed a cushioning layer of asbestos or magnesia 14 to permit of expansion and contraction without disturbing the walls of the furnace. The arrangement of the feeding hoppers 19 is such as to allow the material to slide down the furnace walls and thus reduce the action of the slag on the walls. Each inlet is also provided with peep holes 21 through which the action of the furnace may be viewed and through which plastic material and tools may be introduced for relining the walls when occasion requires. (966,285, Aug. 2, 1910.)

A Mechanical Roasting Furnace designed to simulate the work of the old-style hand-rabbled reverberatory is the basis of a patent granted to Cyrus Robinson, of New York. The invention relates to that type of roasting furnace in which the ore is rabbled on a hearth by means of a rabble moved through the furnace over the hearth. In most furnaces of this type the rabbles move over the entire length of the hearth in a uniform manner and return over another hearth or outside of the furnace. According to the present invention, however, provision is made to operate the rabbles in such a manner that ore in any part of the hearth may be rabbled as needed and may be moved forward toward the fire by means of hoes which are distinct from the rabble fingers. Thus, the hearth being covered with ore in various stages of roasting a section nearest the discharge or fire end may be discharged and successive sections of the ore moved forward, while a new charge is placed on the hearth at the charging end. The rabble carriage does not move uniformly throughout the hearth, but is moved forward and backward as needed. In this way the ore which has been roasted may be sintered before being moved from the hearth so as to be in better condition for blast furnace smelting than the usual fine roasted ore produced by most furnaces. The agitation of the ore is quite separate and distinct from advancing it, the hoes used for advancing the ore being moved out of active position during the rabbling or agitating operation. (968,265, Aug. 23, 1910.)

Chemical Engineering.

Cement from Slag.—United States letters patent have been granted to Hermann Passow, of Blankanese, Germany, for apparatus devised to prepare furnace slag for cement making. The patent is assigned by mesne assignments to the Atlas Portland Cement Company, of New York. The invention relates to the means used to treat slag, producing at will two varieties, which the inventor classes as chemically inert and chemically active. The inert slag is glassy and when ground and mixed with water gives little or no rise of temperature when heated with carbonic acid gas; the active slag is spongy, blistered and friable, and when ground and mixed with water shows a considerable rise of temperature when heated with carbonic acid gas. The two varieties of slag are produced by different methods of disintegrating and cooling the slag as it issues from the furnace.

In order to produce the inert slag the disintegrating means must be so regulated as to break the molten slag as far as possible into spherical particles, or thin sheets, and to cool it as rapidly as possible below a visible heat, without using any water in direct contact. The chemically active slag is produced by breaking the molten slag into irregular masses of spongy nature, cooled as quickly as possible to a condition of plastic consistency, and from this state the cooling must proceed more slowly than for the inert variety, until below visible temperature. In some cases it is advisable to use steam or water with the blast which acts as the disintegrating medium.

These two products may be made separately or together by providing the necessary means of properly cooling to make the two varieties. By mixing the two varieties of slag they are ready for the immediate production of cement. In making the varieties separately it is only necessary to provide proper temperatures of the blast which disintegrates the slag, and the cooling wall on which it strikes or the cooling conveyor on which it falls. If the two varieties are to be made together some inequality in the flow of slag must be provided, so that different portions of it receive different treatment, or an inequality in the disintegrating blast of air or steam may be provided, whereby different portions of the slag stream are blown to different cooling surfaces. (964,805, July 19, 1910.)

Glue from Leather.—Hy. G. Halloran, of Brighton, Mass., patents an electrical method of detanning mineral or chrome tanned leather. The leather in a finely divided state, such as leather scraps, skivings, etc., is placed in a tank in which a sodium chloride solution is electrolyzed, yielding chlorine and caustic soda. The latter acts upon the leather to soften and swell it and open the fibers. The scraps are then removed from this tank, the excess of alkali squeezed out, and the leather subjected to a thorough washing in clear water to remove the remaining caustic soda. The leather is then subjected to the action of chlorine water or chlorine gas to dissolve the tanning agent, after which it is again squeezed and washed and is then converted into glue by the usual method of boiling. By this method leather may be detanned in a few hours instead of several days. The tanning agent is subsequently recovered by mixing the caustic soda and chlorine water to neutralize each other, the latter containing the chrome in solution, thereby precipitating the chrome which may then be collected and used again. The remaining liquor is a mixture of sodium chloride and sodium hypochlorite solution and may be used over again in the detanning process. (967,762, Aug. 16, 1910.)

Electric Furnaces.

Calcium Silicide.—For the refining of steel, calcium silicide has become in recent years an important desulphurizing, dephosphorizing and deoxidizing agent. Messrs T. D Willson and M. M. Haff, of Ottawa, Ontario, produce an alloy of silicon with calcium and one or more other metals (aluminium,

magnesium, iron, tungsten, etc.) by smelting in an electric arc furnace a mixture of calcium carbide and compounds of silicon and of the other desired metals and carbon as reducing agent. The following example is given: A mixture of calcium carbide (CaC_2) containing probably a certain amount of a compound of magnesium, ground sandstone containing oxides of aluminium, silicon and magnesium, and iron and coke were fed to an electric arc furnace and subjected to a current of comparatively low voltage and high amperage (40 volts and 6000 amp). The coke and the carbon of the carbide apparently combined with the oxygen of the several oxides, leaving a silicide which including the slag of unreduced materials gave the following analysis: Silicon, 37.2 per cent; calcium, 23.3 per cent; iron, 4.4 per cent; aluminium, 2.74 per cent; magnesium, 4.96 per cent; silica, 16.80 per cent; carbide, 10.9 per cent.

The sand in this experiment contained 96.55 per cent silica, 0.77 lime, 0.75 iron oxide, 0.82 alumina and 0.28 magnesia. The other metals desired in the alloy may be secured by properly selecting substitutes for or additions to the sand. For example, a higher percentage of magnesium in the product may be secured by the addition of magnesia to the sand in the form of magnesite. To introduce more aluminium an ordinary clay containing both silica and alumina may be substituted for the sand. (963,345, July 5, 1910.)

Purification of Carborundum.—The purification of chemically inert electric-furnace products, such as the carbides of silicon, boron, titanium, etc., is the object of a patent of Mr. Francis W. Higgins, of the Carborundum Company. For instance, in the case of carborundum, where it is desired to remove the impurities from the surface of the grains after crushing, the chief impurities are free silicon and iron and aluminium in the form of carbides and silicides. These are removed by placing the crushed carborundum in a bath composed of aqueous solutions of sulphuric and hydrofluoric acids and permanganate of potash or other oxidizing agent, and digesting for several hours at a temperature of 150 deg. Fahr. to 200 deg. Fahr. (966,399, Aug. 2, 1910.)

Combined Arc and Pole-Plate Electric Furnace.—The Roechling-Rodenhauser combination furnace is in one respect an induction furnace; that is, the charge forms the secondary of a transformer and is heated by Joulean heat produced by the electric currents induced in it from the primary. But besides this, pole-plates are inserted in the walls, opposite to each other. These pole-plates are of metal and are separated from the charge by refractory material which is a non-conductor at ordinary temperatures and becomes a conductor when hot. These pole-plates are connected to an external source of current. When the furnace is started it acts as a pure induction furnace. The charge gets heated up and the refractory material in contact with the pole-plates, by also getting hot, becomes a conductor. Current now passes also directly from one pole-plate to the other through the charge, and the furnace acts as a combined induction and resistance furnace. Mr. Johannes Harden, of the Gröndal-Kjellin Company of London (the owners of the European patents for the induction and Roechling-Rodenhauser combination furnace) now proposes a combination of resistance heating by means of pole-plates with arc heating by means of electrodes above the charge. The furnace may, therefore, be described as a Héroult arc furnace provided with pole-plates or terminal plates. These are faced with refractory material which becomes a conductor when hot. The terminal plates are provided with cavities so that they can be air-cooled. In a three-phase furnace described in the specification three terminal plates are shown arranged triangularly and three electrodes at the top each of them being placed in an intermediate position between two terminal plates. The upper electrodes are operated independently of the current passing through the lower terminal plates. (967,908, Aug. 23, 1910.)

Calcium Carbide.—To reduce the electrical energy consumption in the production of calcium carbide in the electric furnace, Mr. Herman L. Hartenstein proposes to utilize both that heat which is "rendered latent in the formation of carbon monoxide" and that which goes off with the gases as sensible heat, the patent being assigned to the Continental Investment Company, of Duluth, Minn. For this purpose the hot combustible gas (rich in carbon monoxide) coming from the electric furnaces is directly burned in contact with limestone or other form of calcium carbonate to form the lime to be used in the electric furnace. The hot gases coming from the furnace are burned directly in contact with the material to be calcined, freeing it of carbon dioxide and heating it up, and the heated calcined material is delivered to the electric furnace. Both the sensible heat and the heat of combustion are utilized. The limestone is best treated in the powdered or granulated form in an inclined rotary kiln, where it is treated in the form of a tumbling, agitated, relatively thin traveling stream. In order to supplement the heat derived from the kiln gases, producer gas is employed coming hot from a producer. The waste heat from the rotary kiln is employed to heat the air required for combustion in it, and a portion of this waste heat may also be used to heat the coke or carbon supplied to the electric furnace.. (966,542, Aug. 9, 1910.)

Plating.

Rubbing Powders for Nickel and Zinc Plating.—A method of providing articles with a metallic coating by simply rubbing the surface with a plating powder has been the subject of considerable discussion recently in England. Details have now become known by the grant of two patents to Mr. Augustus Rosenberg, of London. A wet rag is simply dipped into the plating powder mixture and then rubbed on the surface to be coated. The principle is that if this surface is metallic, or at least electrically conducting, electrolytic action is set up in the wet mixture which is rubbed on and the desired metal is deposited on the surface. A suitable powder mixture for nickel plating contains

60 parts of nickel ammonium sulphate,
3 " " metallic magnesium,
30 " " chalk,
7 " " talc powder.

When this mixture is wetted by water the nickel ammonium sulphate dissolves and forms the electrolyte. The magnesium is a highly electropositive metal and will therefore form the anode, while the surface to be coated forms the cathode. We have, therefore, an innumerable number of short-circuited electrolytic cells with the particles of magnesium as anodes and the surface to be coated as cathodes. Nickel will, therefore, be deposited on this surface. The process is in principle an electrolytic deposition or electroplating process, but it differs from ordinary electroplating processes, as no external source of electric current is required.

An ammoniacal salt as electrolyte is specially advantageous when the surface to be plated is greasy, as the reaction then helps to dissolve the grease and facilitates the plating operation. The talc powder in the formula prevents the ammonium sulphate from acting upon the other ingredients in consequence of the absorption of moisture from the atmosphere. The mixture may, therefore, be stored or exposed to the atmosphere for lengthy periods without substantial deterioration. In this respect, however, it is also important to protect the metallic magnesium particles, as they would otherwise be oxidized. This is best done by coating the magnesium particles with a protective layer of waxes and resins. For this purpose they are moistened with a solution of wax or resin in gasoline, and then the solvent is dried out, after which the coated magnesium metal particles are mixed with the other ingredients. When the powder is rubbed on the surface the friction quickly removes the wax layer.

For plating zinc on iron or copper a powder of the following composition may be used: 45 parts commercial zinc dust, 15 ammoninm sulphate, 3 magnesium, 30 chalk and 7 talc powder. The magnesium particles are again coated with a wax layer as described above. (970,755 and 970,852, Sept. 20, 1910.)

The Washing of Filtering Residue.

By Oscar Nagel, Ph D.

For filtering purposes the filter press as well as the centrifugal separator have largely their own specific fields of usefulness, while in some cases either apparatus is applicable. The residue of filtration must be washed in such cases where all particles of filtrate adhering to the solid have to be recovered

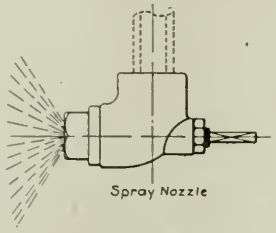


FIG. 1.—SPRAY NOZZLE.

or removed from the residue and, furthermore, where the solid is to be obtained in a certain degree of purity to be effected by the washing.

The process of washing the cakes in the filter press consists practically in pumping through the filter press a washing liquid instead of the material to be filtered.

The process for washing the residue in centrifugals is more difficult, though it is of importance in the manufacture of various organic preparations, in sugar plants, in calcium phosphate plants, etc. In this case it is essential to bring all particles of the residue in contact with the finely divided washing liquid. A very convenient method to accomplish this end is the application of the well-known Koertung spray nozzle, Fig. 1, which for this purpose is connected to the centrifugal separator as shown in Fig. 2 for washing sugar. According to the size of the nozzle and the pressure of the washing liquid, any effect desired can be obtained

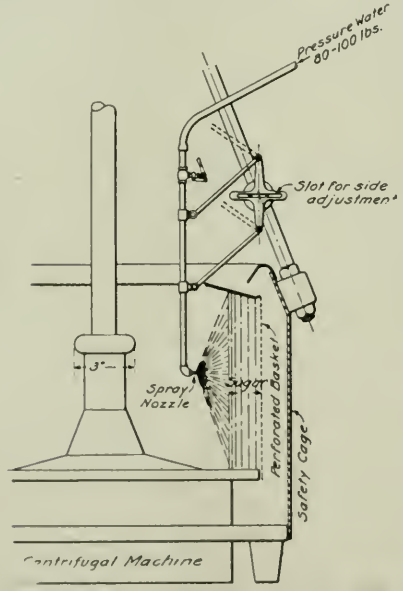


FIG. 2.—ARRANGEMENT OF CENTRIFUGAL SPRAY NOZZLE FOR WASHING SUGAR.

Improvements in Pickling Machines.

The Mesta pickling machine has become a well-known and necessary part of the large tin plate works of this country.

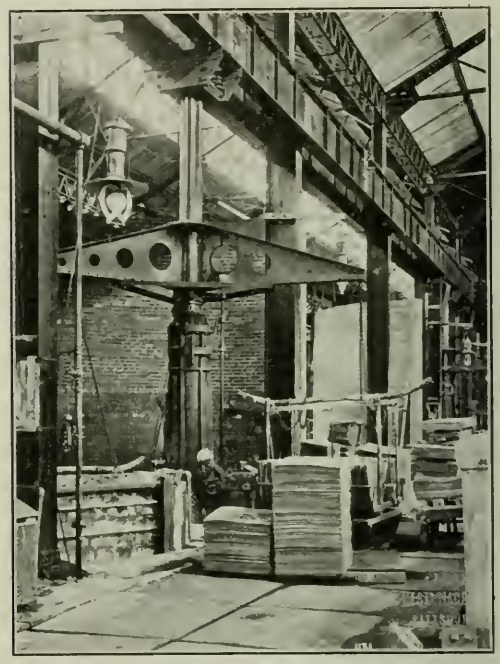


FIG. 1.—PICKLING MACHINES AT MCKEESPORT.

At present there are about 150 of these machines in operation in the United States and 95 per cent of all the tin plate made is pickled by these machines. It is in use for pickling sheets

FIG. 2.—NEW TYPE OF PICKLING MACHINE.

up to 144 in. in length and 48 in. wide, and is also employed for pickling miscellaneous other material, such as steel pipe, wire coils, automobile parts, etc. Fig. 1 shows two of a num-

ber of Mesta pickling machines installed at the plant of the McKeesport Tin Plate Company, at McKeesport Pa.

This machine has recently been improved in some noteworthy respects. As shown in Fig. 2 the latest type requires no top support, the plunger extending down through the base of the machine instead of the top support, as shown in illustration No. 3. This design also contains some other improvements in the details of construction, and is also very much heavier in all parts.

This absence of a top support permits the machine to be placed in any location in the building and is an important feature in modern plants with electric power transmission, as it permits electric traveling cranes to pass over the machine and carry material to and from it. Three of these new type machines have been recently installed in the new tin plate works of the Jones & Loughlin Steel Company, two in the plant of the Phillips Sheet & Tin Plate Company, one on the plant of the Hydraulic Pressed Steel Company for pickling material used for making automobile frames, and one in the plant of the Morris & Bailey Steel Company for pickling steel strips.

The machine is usually built with three arms and two vats, one acid vat and one water vat for washing, but for special purposes, where it is necessary to have three vats, the machine is built with four arms, as shown in Fig. 2.

The machine can be operated by either steam or compressed air. It takes steam, or air, only on its up stroke, while the down stroke is accomplished by gravity. It is regulated by means of a floating piston valve, which controls the exhaust as well as the intake. As the machine is single acting and takes steam, or air, only on the up stroke, the consumption is necessarily small. With 70 lb. of steam the machine will handle 2000 lb. of plates in each of the three crates. It is entirely automatic, not requiring a skilled mechanic to operate it. The crates for holding the plates are made of acid-proof metal, and vary in size and design, depending upon the material to be pickled.

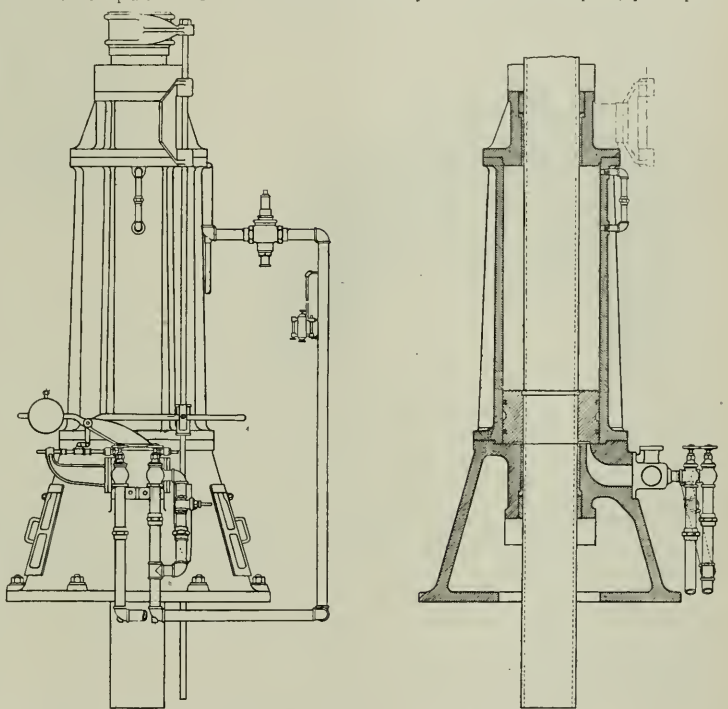


FIG. 3.—DIAGRAM OF MESTA PICKLING MACHINE.

The machine, designed for tin plate works and for pickling sheets up to 60 in. x 36 in., requires a floor space of about 25 ft. square, and has a height of about 18 ft. The inside dimensions of the two vats are 6 ft. 3 in. wide, 7 ft. long and 5 ft. deep. The size of the crates is 61½ in. square. The diameter of the main cylinder is 18½ in.; total lift of arms, 4 ft. 3 in.; diameter of steam pipe, 2 in., and diameter of exhaust pipe, 2 in.

This machine pickles 16,000 lb. of black plate per hour with 140 lb. of acid per ton of black plates. The maximum load of each crate is 2000 lb.

The floor space required for machines arranged for pickling sheets up to 144 in. in length and 48 in. in width is about 40 ft. square; the height about 22 ft. The inside dimensions of the vats are 4 ft. 4 in. wide, 14 ft. long and 6 ft. deep. The crates are 37 in. wide and 128 in. long. The diameter of the main cylinder is 18½ in.; the total lift of the arms, 5 ft. 3 in.; the diameter of the steam pipe, 2 in.; the diameter of the exhaust pipe, 2 in. This size of machine pickles from 12,000 lb. to 15,000 lb. of long sheets per hour. The maximum load of each crate is 2000 lb.

The floor space required and capacity of a machine for pickling wrought-iron pipe or other material depends upon the size and kind of material to be pickled. The Mesta pickling machine is built by the Mesta Machine Company, of Pittsburgh.

Crushing in Stages.

By H. W. Hardinge.

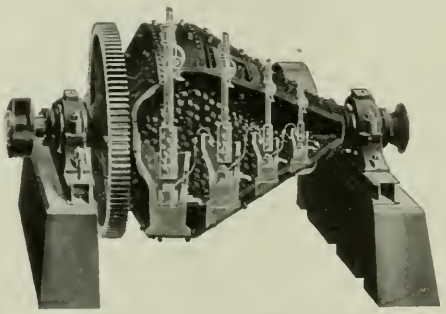
If it was possible to set aside individual interests and view the matter from an entirely disinterested standpoint, yet explain my ideas in another way than by reference to the Hardinge conical mill, it would be better to do so. But as such practices as I advance are based upon my work with both stamps and ball mills I can relieve the reader from unnecessary explanations by referring to the illustration, on page 600, of the conical mill made of two cones, over which I have outlined in shadow serial stamps of different sizes and weights, thus representing the step reduction obtained in the principle and action of the grinding bodies within the conical mill.

A great deal of interest is being taken by metallurgists as to the best method of fine reduction. In South Africa the stamp is being constantly made heavier until it now ranges from 1000 lb. to 2000 lb., the latter apparently having a preference. At the same time it is conceded that with this enlargement only increased capacity is obtained at a sacrifice of fineness. I am of the opinion that this practice should be reversed unless the heavier stamp is considered as a coarse crusher with proportionately increased screen aperture.

After having watched comminution, both in my own prac-

tice and that of others for a great many years, using stamps, Chili and Huntington mills, rolls and numerous other devices for fine comminution, I have come to the conclusion that maximum efficiency for power expended depends upon the exposed crushing surfaces, rather than upon the force or energy behind the surface when it is clogged with the result of the previous charge of material undergoing division.

To illustrate, take a 1000-lb. stamp with an 8-in. shoe dropping 100 times per minute. Considering both shoe and die as each individual surfaces, we would have approximately 100 sq. in. of surface being acted upon by 1000 lb., or 10 lb. of weight for each square inch of crushing surface. This stamp dropping upon a 2-in. cube of ore, weighing approximately ¾ lb., if reduced to a mass averaging ½-in. cubes, would have a relation of 1:64. If the stamp is fairly proportioned to the work, the next stamp should be 1000/64 in., or about 16 lb. This would supply in 16-lb. balls, provided the ratio of weight was alone required. This 4-in. ball has a surface exposure of approximately 50 sq. in., or 4 sq. in. for each pound of weight, or 40 times the surface relation to weight as that of the 1000-lb. stamp shoe and die to its 2-in. cube.



COMPARISON OF CONICAL MILL WITH STAMP MILLS.

My diagrammatic illustration of serial or stamp step reduction in weight of stamps might at first appear to be greatly out of proportion to the work required and it might seem that succeeding stamps should be larger than indicated, while, in fact, they are each ten times too heavy for the successive divisional work to be performed.

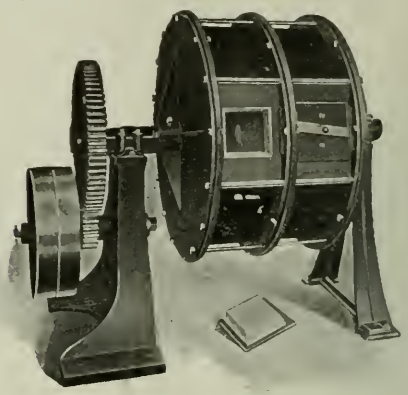
If the metallurgist will cast back over his work, consider the cause and remedies by which he has essayed to better effect a more economic comminution, I am inclined to think he will agree with me that the theory at least appears tenable. Resistance to subdivision under similar conditions is proportional to the mass. Do not waste energy in endeavoring to crush a rock by pounding it upon a more finely divided mass. It is equivalent to trying to break it upon a springy board platform. A 1000-lb. stamp uses energy sufficient to crush 50 tons per day if the crushing faces between shoe and dies are clean. The same principle applies to any other system of dividing. The reason the jaw crusher has such an enormous crushing capacity is owing to the fact of its gravimetrically clearing itself. This is a subject worthy of consideration and an effort to remedy the defect.

Tumbling Barrel for Burnishing with Steel Balls.

The adjoining illustration shows a tumbling barrel, built by the Globe Machine & Stamping Company of Cleveland, Ohio, specially suitable for the polishing and burnishing of small articles by means of steel balls. By the use of this burnishing process all buffing is eliminated on small metal parts before and after plating. The process of burnishing a great many small pieces at once is thus greatly facilitated.

The burnishing tumbling barrel of the Globe Machine & Stamping Company is of the horizontal type, the barrel having an octagonal cross-section. It weighs empty 1800 lb. and occupies a floor space 42 in. x 62 in. The proper pulley speed is 60 r.p.m. and the barrel speed is 15 r.p.m.

The burnishing process itself is exceedingly simple, if some ordinary precautions are taken. There should always be enough steel balls in the barrel so that the articles to be burnished do not rub one another. A safe rule is to use two pails of balls for each pail of material. These steel balls are made especially for this process by the Abbott Ball Company, of Hartford, Conn., in six sizes ranging from 1/16 in. to ¼ in. in diameter.



TUMBLING BARREL.

For each pail of material a pail of hot water is needed. Before adding the water about 6 oz. of chips of ivory soap or some other mild and pure soap are dissolved in the water. Instead of these soap chips Hascoline burnishing compound, or burnishing chips, made by the Haas Brothers Company, of Newark, N. J., may be advantageously used. In ordinary practice 6 oz. of Hascoline chips are dissolved in 2 gal. of water.

This process of burnishing large quantities of small articles in one operation saves considerable time and reduces the cost of finishing the plated articles very considerably.

Combined Agitation and Filtration of Cyanide Solutions.

Since the introduction of the cyanide process for gold extraction there has been practically little change or improvement in the chemistry involved, but inventors have given the mechanical side of the process a great deal of attention. This was necessary on account of the greater number of mechanical difficulties met with as the variety of ores treated increased and the scope of the process extended. The greatest problem to be solved was the rapid dissolution of the gold and silver in the ore, and the subsequent separation of the solution from the ore. Of the many types of mechanical devices intended to accomplish the desired end, those of the Hendryx Cyanide Machinery Company, formerly of Denver, Colo., and now of New York City, are among the latest. They comprise a combination agitator and filter, and a tailings dewaterer

Combined Agitator and Filter.

The Hendryx combination agitator and filter has been designed for the purpose of effecting in one compact machine the dissolution of the precious metals and filtration of the cyanide solution. It consists essentially of four parts: (1) A cylindrical tank with conical bottom, (2) a mechanical agitating pump located in a central well in the tank, (3) a series of filter cells

of the leaf type, and (4) a system of coils of pipe for heating the cyanide solution during agitation.

The construction of the machine will be understood by reference to the accompanying drawing. The tank is made of sheet steel and has a 45-deg. conical bottom. Over the inside of the latter is fastened a canvas sheet which serves to distribute evenly the inflowing wash solution when displacing the gold cyanide solution. This will be referred to later in the description of the operation of the machine.

The agitating pump consists of a circular well within which is a shaft carrying a driving pulley near the top and several screw propellers at different points along its axis. At the top of the well is a circular apron which slopes gradually toward its periphery. The revolution of the propellers causes a strong upward current of solution through the well, finally flowing over the apron, where it is constantly aerated. The charge of ore and solution is kept in rapid circulation and slimes and sands are agitated together without difficulty. No trouble is experienced from the settling of the solid particles of the pulp, even after operations are closed down for many hours.

The filter cells are located beneath the distributing apron and stand vertically in the tank. Each cell consists of a wooden frame with iron tees and elbows, and a piece of cocoa matting which fits inside the frame. This is completely enveloped in a

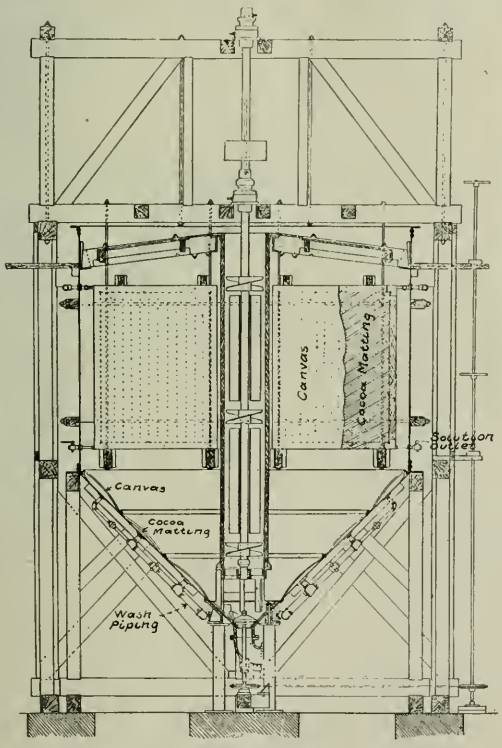


FIG. 1.—COMBINATION AGITATOR AND FILTER.

sheet of canvas, and the matting and canvas are quilted together at intervals. The usual means are provided for the outflow of the filtered solution and the inflow of a wash solution to clean the cell. The coils of heating pipes are suitably disposed in the tank so that the solution is readily heated.

The operation of the machine is varied to meet special conditions, but in general is as follows: The pulp, composed of about one part ore to two parts solution, is run into the agitator from a storage tank until the agitating tank is filled to the bottom of the distributing and aerating apron. The pump is then started and agitation continued until the dissolution of the metals is complete. At the same time the solution is heated to about 90 deg. F. by means of hot water or steam circulated through the heating pipes. When this process is finished the filter cell outlets are opened, allowing the cyanide solution to filter off by gravity to tanks, where it is precipitated by zinc dust. Wash solution is simultaneously run in through the canvas sheet on the bottom of the tank to displace the gold solution in the pulp settling below the filter cells. The charge is filtered until its level is nearly to the bottom of the cells. This

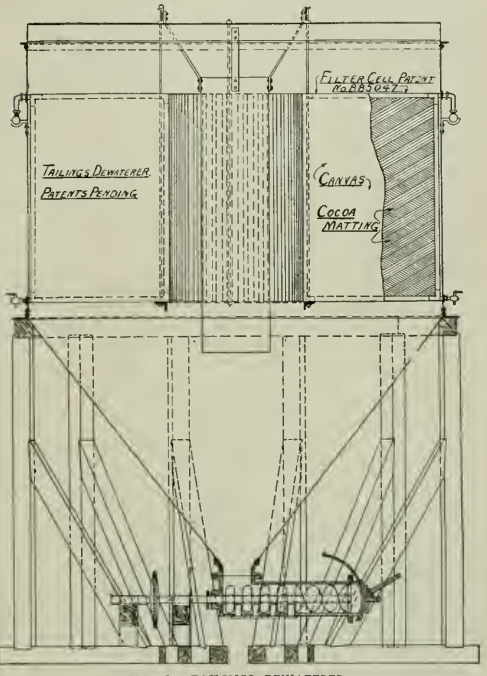


FIG. 2.—TAILINGS DEWATERER.

filters off about 80 per cent of the solution and dissolved values. Should a slimy ore reduce the rate of filtration, the filters may be cleaned by a reverse current of solution when they are apparently clogged. At the close of the first filtration a barren wash solution is run in through the canvas in the bottom of the machine until the charge again rises to the top of the filter cells. The filters are thoroughly cleaned by a reverse current of wash solution and agitation proceeds as before for a short time. This wash is then filtered and another barren wash applied if necessary. If not, fresh water is run through the canvas in the bottom, and through the inside of the filter cells, in sufficient quantity to compensate for the moisture in the tailings. The agitator is then started, and when the charge is thoroughly mixed, a discharge gate is opened at the bottom of the tank and the pulp is discharged.

Tailings Dewaterer.

Where it is necessary to conserve water, or where it becomes necessary to stack the tailings in a dewatered condition, the Hendryx tailings dewaterer is installed to receive the discharge from the combined agitator and filter. This dewaterer consists of a round tank with a conical bottom within which are placed vertical filter cells to clarify the solution. At the bottom of the cone is an outlet in which is a screw connected to a shaft

by a sprocket. As the tailings settle they are removed from the conical bottom and forced through the outlet by the revolving screw in a continuous uniform stream.

The Hendryx company gives the following figures relative to these machines. In the 16-ft. combined agitator and filter the filter cells are 6 ft. high and 7 ft. long. A machine of the size mentioned and operated as described has approximately 6000 sq. ft. of filtering canvas and will yield as high as a ton of clear solution per minute. The time necessary to treat a charge ranges from two to four hours, depending on the character of the ore. A machine capable of handling 100 tons of ore per day requires 6 hp. The reason for the small power consumption is that there is only an 18-in. lift, as the pump casing and tank are normally filled to the same level.

The tailings dewaterer will dewater 200-mesh tailings down to 12 per cent to 15 per cent moisture. An 18-ft. machine has a capacity of 200 tons of dewatered tailings and 400 tons to 600 tons of water or solution per 24 hours. The Hendryx company has records of many tests showing the efficiency of the machines described.

Concentrating Table.

The two adjoining illustrations show a new and interesting concentrating table, designed by Mr. C. W. Eccleston and built by the Eccleston Machinery Company, of Los Angeles, Cal.

The table top is placed in a frame and suitable means arranged on each side of the table to place the table under heavy tension transversely; several clevices are attached to the table on either edge, into which are hooked springs of suitable strength, which are connected to the frame by the hook bolt provided with a threaded end, and a wing nut for regulating the tension on each spring independently.

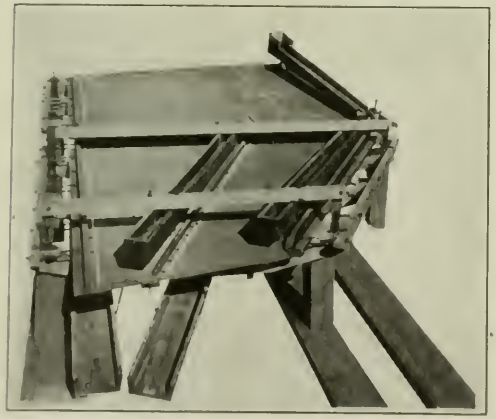


FIG. 1.—CONCENTRATING TABLE.

The table is made in two sections, the first section, 8 ft. x 8 ft. is provided with a feed box, and is so constructed that the first 3 ft. below the feed box have a gentle slope, the remaining 5 ft. are almost flat, forming a pool of 5 ft. x 8 ft. The heavy sands and sulphides are precipitated on the first 3 ft. and are carried forward to the second deck, the lighter particles such as slime are precipitated in the pool and are also carried forward to the second deck.

The second deck is placed directly in front of the first deck and below it, so that the material being driven forward off the first deck is caught by the second deck. The second deck is 8 ft. wide at one end, 7½ ft. long and 3 ft. wide at the concentrates discharge end.

This deck is divided in two sections and is provided with two wash-water boxes. The heavier sands and sulphides precipitated on the upper side of the first deck are received on the upper side of the second deck; the fine sands and mixed slime that has been precipitated in the pool are delivered to the lower section of this second deck. The two products are thus kept entirely separate and the wash water is regulated to suit the two products. By this means a classification is made of the feed and it is dressed separately. Provision is made to take off the two classes of tailings; the sulphides are also delivered separately.

The second deck is adjustable independent of the first deck. After the first deck is adjusted to any requirement the second deck can be adjusted to dress the product delivered to it from the first deck. The adjustments cover any range of work and are positive.

One head motion operates the two decks. The combined weight of the two decks is 240 lb. There is over 100 sq. ft. of concentrating surface.

The objects in placing the table tops under tension transversely are many. It is said that a much more delicate movement can be secured in this way; further, that the method of attaching the tensional means to the edge of the table places the entire supporting frame and all the weight of the machine in the plane of the table tops to absorb vibrations and control momentum. By using this tensional means and supporting the table in this way a very light table top can be used one that can be stretched into shape and always held in shape.

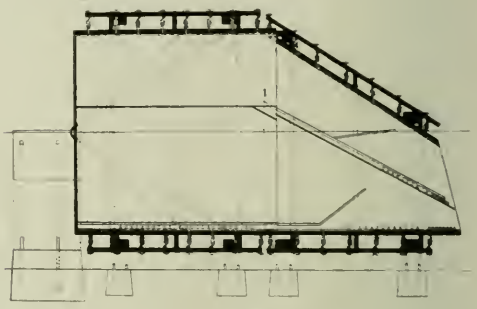


FIG. 2.—DIAGRAM OF CONCENTRATING TABLE.

By having a light table top under tension it can be run at a very high speed, viz.: 400 r.p.m. to 500 r.p.m., giving a delicate vibration entirely void of shocks and jars.

It is well known that a classification of the feed to a concentrator is absolutely necessary to get the best results. This machine takes the classified product and makes still another classification of it and dresses the products separately.

The machines are built for the different products viz: roughing table for very coarse material, sand table for material from 15 mesh to 100 mesh, sand table for very fine and overflow slimes.

From differential indicator diagrams plotted by the builders they conclude that a very great range of the differential can be obtained on this machine. A change in the differential can readily be made without changing the stroke.

The deflection of the table tops in this machine is peculiar, as the transverse tensional means provide for an increase and decrease of the transverse and longitudinal tension as the table is vibrated by the head motion.

As the table travels forward on the impelling stroke the speed increases rapidly, due to the differential mechanism, at the same time the transverse tension on the table tops is decreasing. Consequently, when the table arrives at the termination of the forward or impelling stroke it is traveling at its greatest speed and is under the least tension transversely and

longitudinally. The table tops are brought to a sudden stop at this point and start on the return stroke; the table tops at this point drop below the plane of vibration, due to a body taking the line of least resistance on being brought to a sudden stop. The tables are now returning, the speed is decreasing, and the tension transversely and longitudinally is increasing, consequently when the table tops arrive at the point of starting they are traveling at their slowest speed and are under the greatest tension, so that they do not leave their plane of vibration.

This drop in the table tops is very efficient, as it relieves the friction between the table tops and the pulp to a certain extent, allowing the pulp to travel forward further and faster than it would otherwise do. The drop or deflection can be controlled by the transverse tension. A very small drop is all that is permitted—being from 1/64 in. to 1/100 in.

Metallurgical Microscope.

In our August issue, page 503, we described the new micro-metallograph of Ernst Leitz, Wetzlar, Germany. While this elaborate instrument has proven very successful there is also a demand for a less complete and less expensive metallurgical microscope sufficient for all the usual work which the metallographist has to perform.

The adjoining illustrations show such a new metallurgical

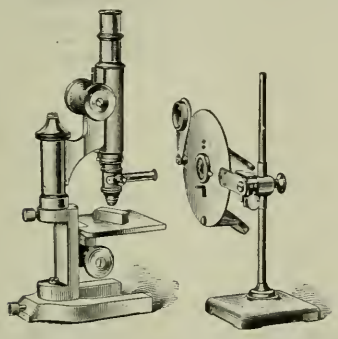


FIG. 1.—METALLURGICAL MICROSCOPE WITH ILLUMINATOR.

microscope developed by the firm of Ernst Leitz, with the aid of Dr. Wm. Campbell, the well-known metallographical expert and professor of metallurgy at Columbia University.

Fig. 1 shows the microscope mounted for laboratory research work with a new opaque illuminator in connection with a specially designed illuminating stand which insures an even illuminated field with the most powerful objectives. The stage may be raised and lowered by rack and pinion so as not to disturb the relative position of the vertical illuminator and the source of light. This rock and pinion arrangement fulfills at the same time the work of the coarse adjustment of the microscope. The fine adjustment is effected by the delicate micrometer screw on the top of the column.

To overcome the great difficulty of having two absolutely parallel surfaces of the specimen to be examined the stage is also provided with an ingenious device for inclination to insure in all cases absolute parallelism between the surface to be examined and the front lens of the objective.

Fig. 2 illustrates the microscope arranged for examination of large metal surfaces and castings at the works. The stage of the microscope with rack and pinion is omitted and the tube with coarse and fine adjustment is attached directly to the horseshoe case. In this instance the coarse adjustment is made by rack and pinion movement of the tube.

The works of Ernst Leitz have branches in this country at 30 East Eighteenth Street, New York City, and at 1923 Ogden Avenue, Chicago, Ill.

NOTES

The **American Foundrymen's Association** will hold its convention next year in Pittsburgh from May 21 to 23.

The Goldschmidt Thermit Company, of New York, announces an important change in its management. Commencing Oct. 1 Mr. E. Stütz, vice-president and general manager, retires from the direction of the company, which passes under the management of Mr. William C. Cuntz. Mr. Cuntz brings to his position a thorough knowledge of the steel business and a wide acquaintance with the railway and street railway officials of this country, having been connected for 18 years with the Pennsylvania Steel Company.

The **Rockwell Furnace Company**, of New York, announces the opening of a Chicago office in the Fisher Building, rooms 718 and 719, which will be in charge of Mr. A. L. Stevens.

Phosphate lands to the extent of 2,500,000 acres have been withdrawn from entry in Idaho, Wyoming and Utah in accordance with the Govenment's policy of conservation. It is believed by the Government experts that these phosphate fields may prove to be the most extensive in the world.

Sulitelma Mines.—It is reported from the Sulitelma Company's mines in Norway that for the month of August 645 tons of concentrates containing 7.38 per cent copper were produced by the Elmore process.

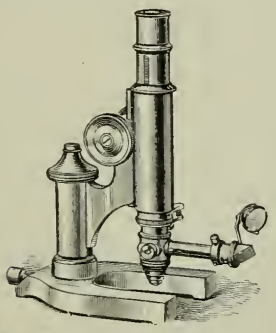


FIG. 2.—EXAMINATION OF LARGE METAL SURFACES AND CASTINGS.

The **Thwing Instrument Company** has been organized to manufacture the line of pyrometers which has heretofore been manufactured by Dr. C. B. Thwing, the inventor of the Thwing multiple-record recorder and of the Thwing fixed-focus radiation pyrometer. He will remain as president of the company, the other officers being: Vice-president, Mr. E. C. Stover, assistant general manager of the Trenton Potteries Company; secretary-treasurer, Mr. L. R. Woodhull. The offices are at 445 North Fifth Street, Philadelphia.

The **Pittsburgh Testing Laboratories**, inspecting and consulting engineers and chemists, of 325 Water Street, Pittsburgh, Pa., announces the removal of their New York office from 1 Liberty Street to 50 Church Street, Hudson Terminal Building. The company's interests in New York and New England have been placed in the hands of the second vice-president of the company, Mr. William F. Zimmermann.

The Camp Bird, Ltd., Ouray, Col., treated during the month of July, 1910, 6818 tons of ore, yielding 9985 oz. of bullion and about 824 tons of concentrates. The financial statement for the same month shows about $186,000 applicable to profit.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in June, 1910, was 71. There were milled 1,814,195 tons of ore; 9870 stamps were in operation with an average duty of 7 tons per 24 hours; 178 tube mills were at work. Comparison with last year's

record for the same month shows an increase in tonnage milled, number and duty of stamps, and number of tube mills in operation. The yield for the month was 625 18t fine ounces gold.

The estimation of slime can be accomplished by the use of the ordinary laboratory centrifuge, using specially calibrated tubes. Mr. G. J Young recommends tubes calibrated with known weights of pyrite. By using a mixture of 3 or 4 of water to 1 of slime and revolving the same in the centrifuge the component parts of the pulp will settle to the bottom of the tube according to their specific gravity, and a line of demarcation between slime and fine sand will be plainly noticeable. Much coarse sand renders the results unsatisfactory, but for quite clean slime the method is said to be good.

Water Softening and Filtration Systems is the title of a new trade pamphlet recently issued by the Booth Apparatus Company, Syracuse, N. Y. A brief description is given of the essentials of water softening plants, also some illustrations of plants varying from 250 hp to 10,000 hp. Mr. Wm. Booth is the well-known chemical engineer and water expert of Syracuse, whose contributions to chemical engineering literature have been repeatedly noticed in these columns.

The Goldschmidt Thermit Company, New York, has issued a very neat bulletin on thermit butt-welded pipe joints, illustrating and describing this kind of work. The process, which is well-known to the readers of this journal, has found much favor, especially for ammonia pipes, etc., where any leak would mean a considerable loss.

The Bonnot pulverizer is a machine adopted from the cement industry for fine metallurgical grinding. The machine is said to be efficient as a fine grinder and also as an agitator during the process of grinding. The Bonnot Company, of Canton, Ohio, has issued a bulletin describing and illustrating the machine.

Cleaning Metals.—A small folder has been received from the New York Chemical Works, H. M. Anthony Company, New York, describing the uses of "Kalye," a cleaning compound, to be used in the preparation of metals for electroplating or lacquering. This substance, which is dissolved to make a hot-water bath, does not saponify oil or grease, and is said to give good results with little manipulation. It has been upon the market for over a quarter of a century. Sample cans, enough for 2 gal. of cleaning solution, sent upon request.

Valves.—A handsome 220-page catalog comes to us from the Nelson Valve Company, Philadelphia. It is divided into three sections illustrating and briefly describing the products of the company. Section I deals with bronze valves; globe and angle, check, swing and gate valves in standard sizes from ¼ in. to 3-in. pipe and for working pressures; standard 125 lb., medium 175 lb., and extra heavy 300 lb. Special bronze gate valves have been built up to 5000-lb. service. The Nelson double-disk tapered-seat valve for all gate valves is strongly featured. Section II deals with iron-body valves bronze mounted for various services and in sizes from 2 in. to 30 in. One of the specialties of the company is an extra heavy gate valve for superheated steam up to working pressure 400 lb. at temperature 800° Fahr. The body of the valve is of steel and the disks, seat rings and bushings of monel metal, which admirably withstands corrosive conditions. Gate valves are arranged for operation by hand or from a distance through gearing or by hydraulic or electric means. Section III illustrates accessories and fittings.

Boiler-Cleaning Devices.—The General Specialty Company, of Buffalo, N. Y., have recently issued "Pictorial Proof," a neat booklet giving illustrations, tests and reproductions of testimonial letters regarding their boiler-tube cleaning devices. The "Demon" is designed to remove the scale from water-tube boilers, and consists of a triplex cutting head driven by a small high-power water motor, to which it is attached. The cutter and cutter arms are set in a non-yielding manner to the net inside diameter of a clean tube, which means that all the scale will be removed. The "Torpedo" cleaner is used for fire-tube boilers. It consists of a head with centering lugs and a vibrator driven by steam or compressed air to give 11,000 oscillations a minute. The short, light stroke knocks off the non-elastic scale without injuring the tube.

Mechanical Draft for Boilers.—"More Power for the Same Money or the Same Power for Less Money" is the heading of a neat booklet by the Turbo Blower Company, New York. After a few pages on the advantage and economy of mechanical draft over natural chimney draft follows an illustrated description of a somewhat unusual type of blower. The "turbo blower" is self-contained and occupies no space outside the boiler setting, the entire length of the case being masoned into this setting. It consists of a five-bladed fan with an enlarged hub, providing a chamber containing the vanes of a miniature steam turbine of the impulse type. The spider which supports the bearing also has an enlarged hub, which forms the other half of the chamber and accommodates three or four stationary nozzles which direct boiler steam against the vanes. The spider is bolted to the cast-iron cylindrical casing imbedded in the furnace wall, and the revolving fan forces air into the ash pit under the grate and thus through the furnace to the stack. The apparatus is of rugged design, as well as compact, for the smallest size, 15-in. diameter fan is intended to serve boilers up to 150 hp. The device is simple and ought to be successful provided that the turbine arrangement offers a fair economy in steam consumption.

The Utah Copper Company has issued its ninth quarterly report, under date of Aug. 4, 1910, for the second quarter of 1910. Each of the company's three concentrating plants was in operation at about full capacity throughout the quarter. Alterations at the Magna plant have progressed nearly to completion and the capacity of the plant has been increased; while only one-third of the plant is operating in its remodeled form, the alterations for the balance of the plant will be put into service in a short time. The alterations at the Arthur plant have not been made, as the necessary material has not yet arrived. The ore treated in this quarter contained about 1 lb. of copper per ton more than that treated in the first quarter. The total copper contained in the concentrate produced and sold was 25,124,052 lb. The gross production by months was as follows:

April	7,902,643 lb.
May	8,862,913 "
June	8,358,684 "
Average	8,374,684 "

Net profits from mining and milling	$1,184,199.05
Additional miscellaneous income	8,351.94
Income from Nevada Cons. dividend	356,428.50
Total net profit for the quarter	$1,584,979.49
Dividend paid	1,155,243.75
Net surplus for the quarter	393,735.74

Average price of copper on which above earnings are based was 12.5 cents per pound as compared with 13.33 cents per pound for the previous quarter. The average cost per pound of net copper produced, after deduction of smelter allowances, was 7.53 cents. During May, when slightly richer ore was mined, the cost of copper per net pound was 7.17 cents. During the same month the cost at the Magna plant was 7.06 cents. The railway which the Utah Copper Company is interested in building between Bingham and Garfield, where the mines and mills are located, is progressing rapidly and it is expected that the open grading work will be completed before cold weather sets in. The railroad will probably be completed and in operation within a year and will serve to relieve the unsatisfactory transportation conditions that now exist.

Water Purifying Devices.—From the Hoppes Manufacturing Company, Springfield, Ohio, come a couple of catalogs showing their feed-water heaters, steam separators, etc. The live-steam water heater and purifier uses a number of dished basins or trays one above the other in a cylindrical steel shell. Water is fed into the top pan, fills it and trickles over following the curved bottom to the lowest point, from which it drops into the next pan for a similar operation. Solid matter in suspension settles in the trays and matter in solution is precipitated on the bottom surface of the trays by the heat of steam, which always comes into direct contact with it. When live steam is used in the purifier no chemicals are necessary to purify the water as the high temperature of the boiler pressure does this effectively. An exhaust-steam feed-water heater and purifier is made on the same plan with the addition of a baffle plate to remove the oil. Four types are made for various requirements, one of them having a cast-iron bottom and pans and particularly adapted to handling water of a corrosive nature or containing acids. The sizes range up to 40,000 hp. The Hoppes steam separator shows a somewhat novel design in that baffle plates and various obstructions are avoided, the steam gradually expanding into a large chamber where the water is intercepted by a cone with water-sealed edges to prevent the escape of entrained moisture. The separator is made in various types and an oil eliminator, also an exhaust head, are described in the booklet.

PERSONAL

Mr. **H. B. Abbott,** formerly Western manager of the *Electric Railway Journal,* has accepted a position on the advertising staff of Metallurgical and Chemical Engineering.

Mr. **John H. Banks,** of Ricketts & Banks, New York, has been in Idaho on professional business.

Mr. **H. K. Benson,** Professor of Industrial Chemistry in the University of Washington, Seattle, has been engaged in field work for the Soil Survey during the past season.

Mr. **Henry D. Boddington,** formerly engaged in professional work in the Southwest and Mexico, has gone to Philadelphia, where he will spend some time.

Mr. **D. W. Brunton,** of Denver, attended the sessions of the American Mining Congress at Los Angeles.

Mr. **R. B. Carnahan, Jr.,** of the American Rolling Mill Company, of Middletown, Ohio, the inventor of the process of making "American ingot iron," has returned to this country on October 1 from a six months' trip to Europe, where he studied the latest practice in rolling mill methods.

Prof. **F. G. Cottrell,** of the University of California, has organized an association of students and professors for the purpose of studying the various phases of the smelter fume problem. There will be co-operation on the part of the College of Agriculture and the College of Mining.

Mr. **A. C. Dart,** general manager of the Rambler Copper & Platinum Company, Holmes, Wyo., spent a week in Denver last month on business for his company.

Mr. **E. L. Fletcher,** formerly of Tonopah, Nev., has been placed in charge of the tungsten mines at Osceola, Nev.

Mr. **Charles S. Herzig,** of Constant & Herzig, New York, has been appointed consulting engineer for the Jumbo Extension Mining Company, Goldfield, Nev.

Mr. **George T. Holloway,** consulting metallurgist, chemist and assayer, has removed his laboratories and testing plant from Chancery Lane to 9-13 Emmett Street, Limehouse, London, England.

Mr. **W. H. Seamon,** Chihuahua, Mexico, has recently disposed of his laboratory to Wilding & Keller, of Parral, who will conduct it as a branch office.

Mr. **James A. Snedaker** has removed his offices in Denver to 818-819 First National Bank Building.

Mr. **A. H. Wethey,** manager of Senator Clark's interests in Butte, returned last month from his trip to Europe.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

Electric Furnaces (Continued).

No. 679,432, July 30, 1901, George D. Burton, of Boston, Massachusetts.

Resistance type, especially for ore roasting. The furnace consists of a rotary cylinder of steel or fire-clay, through which pass resisters consisting of hollow open-ended steel tubes in conductive connection with hollow axles, through which gases may be introduced and exhausted through suitable condensers and receivers. The furnace is either horizontal or inclined.

No. 679,973, August 6, 1901, Denis Lance, Raphael Louis, Emmanuel de Bourgade and Leon Schmitz, of Paris, France.

Resistance type, particularly for effecting chemical reactions in which gases are used as in making cyanides. The furnace is narrow and either straight or annular; has walls of porcelain heated externally by wire resistors. The charge is placed in the lower part of the furnace and covered with fragments of porcelain through which gas is introduced, passes through the charge and out at the bottom. The resistor wires may be connected at one end to a common terminal and at the other to contacts over which revolves a contact arm.

679,973, Aug. 6, 1901. D. Lance, R. L. E. de Bourgade & L. Schmitz, Paris, France.

Resistance type, for reacting on solids with gases, e.g., for producing cyanides. The heating chamber is an annular or straight porcelain crucible, heated by windings of wire. In the annular type, the windings may be external and internal helices; or both types may have vertical parallel resistance wires, connected to bars at their opposite ends. The current may be passed successively through the wires by a distributor. The crucible and windings are imbedded in sand and asbestos. The crucible has a gas inlet and perforated diffuser at the top and an outlet at the bottom. The solid to be heated, e.g., carbon, occupies the lower half of the crucible, which is filled above it with fragments of porcelain to divide the gases and give uniform pressure.

681,107, Aug. 20, 1901. A. H. Cowles, Cleveland, Ohio.

The arc type of furnace, with porous carbon walls, shown in prior patents 676,576, etc. In smelting a mixture of sodium aluminate and carbon, sodium and aluminium carbide result, the sodium vapor escaping through the walls and being condensed to a liquid in a chamber kept above 96° C. If iron, copper or tin be added to this mixture, the aluminium alloys therewith. If zinc be added, an alloy of sodium and zinc volatilizes. By mixing sodium and calcium compounds and proper amounts of carbon, sodium and calcium carbide may be produced.

682,088, Sept. 3, 1901. F. A. Kjellin, Gysinge, Sweden.

Induction resistance type, for smelting or heating ores and metals. The heating chamber is a horizontal annular body of refractory material, having a circular groove, with tapholes and a forehearth at one side. The inducing winding is a helix, within this body and inclosing one portion of the iron core, which is shown as a vertical (laminated) ring linked through the body.

683,089, Sept. 24, 1901. K. C. Wideen, New York City.

Arc type, for reducing metallic compounds, especially oxides, without the addition of carbon or other reducing agent. By means of mirrors, the heat of several arcs, or of the sun, is focused upon the compound, dissociating it. A charge of alumina is reduced by supporting it in a cavity in the top of a mound of alumina, or a block of asbestos, within a chamber having a roof of two transparent plates. The chamber is initially connected to a vacuum pump and the heating rays are concentrated downward upon the charge, raising it to about

10,000° Fahr. Aluminium of a purity of 99.23 per cent has thus been produced, also pure silicon, as stated. A reducing agent, e.g., a hydrocarbon gas, may be used.

683,107, Sept. 24, 1901. L. S. Dumoulin, Flushing, N. Y.

Resistance type, for smelting ore. The chamber has fire-brick walls, with openings at opposite sides receiving the resistor. This consists of a carbon bar with large ends, the thinner middle portion being covered with fireclay, applied plastic. The whole may be slipped into a loosely-fitting fire-clay tube for further protection. A furnace for smelting one ton of copper ore per day should have two horizontal resistors, placed 2 in. above the bottom, each consisting of a carbon bar 12 ft. long, their ends having a diameter of 2 in. and their intermediate portion of $\frac{3}{4}$ in. A current of 5000 watts is supplied.

684,296, Oct. 8, 1901. W. Nernst, Göttingen, and L. Glaser, Coburg, Germany.

Resistance type. The resistor is a "dry electrolyte," carrying preferably an alternating current. A mixture of magnesia and calcium carbonate, alumina, silica or kaelin is preferred, magnesia alone having too high resistivity. The resistor is shown as a vertical tube, surrounded by a non-heat-conductor, e.g., loose magnesia, lime or alumina, inclosed in a jacket. This tube is preferably lined with pure magnesia, one or both sides of which lining may be covered with fine graphite to increase conductivity. Connection is preferably made to the ends of the resistor by bodies of compressed ferric oxide and metal clamps. The resistor is preferably preheated to the temperature to pass current by an internal carbon bar resistor, which is then withdrawn. The charge is then introduced, e.g., in a crucible pushed up from below. Preheating may be effected by a carbon or platinum resistor; or by an initial resistor-layer of powdered carbon or silicon, which burns away; or by burning coal-gas or a burning mixture of a solid combustible and a nitrate.

OBITUARY.

Mr. Gabriel L. Feldman, a leading expert in the electroplating industry, died on Sept. 16 at the age of 54 years, following an operation for appendicitis. Mr. Feldman was a native of Philadelphia, but became connected early in his life with the Hanson & Van Winkle Company, of Newark, N. J. During the 35 years of his connection with this firm he advanced through various positions to that of secretary of the company, which office he held at the time of his death. He leaves a widow, four daughters and three sons.

NEW BOOKS.

Becker, O. M. High-Speed Steel. The development, nature, treatment and use of high-speed steels, together with some suggestions as to the problems involved in their use. 300 pages, many illustrations. Price, $4 net. New York: McGraw-Hill Book Company.

Chwolson, O. D. Traite de Physique. Traduit par E. Davaux. Vol. III, part 2: General thermodynamics. Fusion. Vaporization. 340 pages, 206 illustrations. Price, in paper cover, 11 francs (retail price in New York, $3.65). Paris: A. Hermann & fils.

Fulton, Charles Herman. Principles of Metallurgy. An introduction to the metallurgy of the metals. 544 pages, 188 illustrations. Price, $5 net. New York: McGraw-Hill Book Company.

Klar, M. Technologie der Holzverkohlung. Second edition. 430 pages, 49 illustrations. Berlin: Julius Springer.

Treadwell, F. P. Analytical Chemistry. Vol. II. Quantitative Analysis. Translated by Wm. T. Hall. Second edition. 787 pages, illustrated. Price, $4. New York: John Wiley & Sons.

Van Nostrand's Chemical Annual, 1909. A handbook of useful data. Second edition. Edited by John C. Olsen. 580 pages. Price $2.50 net. New York: D. Van Nostrand Company.

Bulletin No. 15, Illinois State Geological Survey. Geography of the Middle Illinois Valley by Harlan H. Barrows. 128 pages. Urbana: University of Illinois.

Iron Ore Resources of the World. Summary composition upon the initiative of the executive committee of the Eleventh International Geological Congress, Stockholm, 1910, with the assistance of geological surveys and mining geologists of different countries. Edited by the general secretary of the congress. Accompanied by an atlas of 42 maps. Price $20. New York: Lemcke & Buechner.

Summary Report of the Mines Branch for 1909. 181 pages. Ottawa: Department of Mines.

BOOK REVIEWS.

A Text Book of Organic Chemistry. By William A. Noyes, professor of chemistry in the University of Illinois. 537 pages. Price, $1.50 net. New York: Henry Holt & Company.

Professor Noyes' text-book of organic chemistry appeared first in 1903. In the meantime a German translation was issued and in its introduction no less an authority than Wilhelm Ostwald expressed high praise on the great independence and originality shown by Professor Noyes.

The changes and additions which had been prepared for the German edition of the book have been included in the second American edition. The chapter on compounds of interest in physiology and pathology has been rewritten on the basis of the classification of proteins recently adopted by the American Society of Biological Chemists and the American Physiological Society.

The Construction of Graphical Charts. By John B. Peddle, professor of machine design, Rose Polytechnic Institute. 109 pages, 58 illustrations. Price, $1.50 net. New York: McGraw-Hill Book Company.

"Much of the work of calculation done by the engineer or designer is in the repeated application of a limited number of formulas to a variety of different conditions, which involves merely the substitution of different variables in identical equations.

"Any mechanical means for performing this operation expeditiously will not only lead to a saving of time and mental wear and tear, but will also minimize the chances for error.

"Such a device is the calculating chart, or nomogram, and the increasing frequency with which it is employed in the more recent technical publications is a good evidence of the growing recognition of its value.

"Many excellent examples of these charts have appeared of late years and are available for use, but it is evident that to realize their full value as useful instruments the engineer should have a sufficient acquaintance with their underlying principles to construct charts suited to his individual needs.

"Some of the chart forms employed to-day have been known and used for many years, but it is only within recent times that any systematic study has been made of the subject as a whole or any attempt to properly classify and correlate the different types.

"In this work the French have been pioneers, and it is to one of them, Maurice d'Ocagne, that we owe what is probably the most thorough and comprehensive text on the subject, his Traite de Nomographie."

Professor Peddle has done useful work in preparing for the use of the American engineer a simple and brief volume on the theory and practice of the construction of graphical charts. We believe it is the first of its kind in the English language. All mathematical explanations are made as simple as possible and numerical examples are added.

There are seven chapters dealing with the following subjects respectively: Charts plotted on rectangular co-ordinates; the alinement chart, alinement charts for more than three variables, the hexagonal index chart; proportional charts; empirical equations; stereographic charts and solid models.

Metallurgical and Chemical Engineering

A Consolidation of Electrochemical and Metallurgical Industry and Iron and Steel Magazine.

Vol. VIII. New York, November, 1910 No. 11

PUBLISHED MONTHLY BY THE

Electrochemical Publishing Company

239 West 39th Street, New York.

Pittsburgh Office, 201 Oliver Building.
Western Office, 1510 Court Place, Denver, Colo.
European Office, Hastings House, Norfolk St., Strand, London, Eng.

J. W. Richards, Ph.D., President.
Curtis E. Whittlesey, Treasurer. E. F. Roeber, Secretary.
Telephone Call: 4700 Bryant.

E. F. Roeber, Ph.D., Editor.
H. C. Parmelee, Western Editor.

Yearly subscription price for United States, Mexico and United States dependencies, $2.00; for all other countries, $2.50 (European exchange, 10 shillings, 10 marks, 12.50 francs).

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Entered as Second-Class Matter at the Post Office at New York, N. Y., under the Act of Congress, March 3, 1879

New York, November, 1910.

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The U. S. Steel Corporation and Electric Steel Refining.

While going to press, we are informed that the United States Steel Corporation has secured complete control of the Héroult patents for electric steel making and refining in this country. From a commercial standpoint this news appears to be of momentous importance. As well known to the readers of this journal, the Steel Corporation has had two 15-ton Héroult furnaces in commercial operation for some time, one at South Chicago, Ill., the other at Worcester, Mass. At South Chicago the electric furnace is used in connection with the Bessemer converter to make large-tonnage steel, such as axle steel, rail steel, etc. At Worcester the electric furnace is used in connection with the open-hearth to make high-grade steel for wire, etc. The operation at the South Chicago plant was described at some length in an article in our April issue (page 179). As to the further technical development of the Héroult process in the hands of the United States Steel Corporation, it may be surmised that two patents of Mr. William R. Walker, of the Steel Corporation, will come into great prominence. One recorded in our October isssue, 1909 (Vol. 7, page 450), refers to the division of the different metallurgical reactions between acid-lined converter and basic-lined electric furnace, with a view of reducing electric power consumption and avoiding losses of metal. The other patent of Mr. Walker, recorded in our September, 1910, issue, page 546, gives a prominent place to the mixer in the Bessemer converter-electric furnace combination process.

Screen Analysis.

Considering its value and importance in metallurgy and ore dressing the screen analysis is not as extensively used as it should be. As a first aid in adjusting a disordered mill it is of prime importance, and in planning a new system of ore treatment it is indispensable and equal in importance to the chemical analysis which should accompany it in some instances. Economy should result in the occasional screen and chemical analysis of mill pulp at various stages in its treatment. In this way the efficiency of any step could be ascertained, and the proper adjustment of the machines determined. Loss of valuable material could be prevented, or perhaps a barren portion of the pulp could be eliminated from further treatment. Certain steps in the process might be found useless, and could be discarded, or, on the other hand, additional refinements might be indicated which could be installed with a resulting profit.

The basis of good screen analysis is reliable screen cloth. Little good can be accomplished by the use of screening which is irregular in size of aperture or gage of wire, giving a variable number of meshes per linear inch. Conversely, the greatest good results from the use of a series of standard laboratory screens suitable to the process used in the mill, comparable in every way with the standard mill screens, and with a definite relation between the various screens in the

series, as to gage of wire and size of aperture. Such screens are obtainable as a result of the co-operation of manufacturers and specialists in ore treatment. They are not to be compared with screens made of commercial-grade metal cloth, which is notoriously irregular. The results obtained from standard screens are proportionately superior to those from ordinary screens, and are the only reliable guide for accurate work.

Metallurgical Evolution.

It frequently has been observed that old mining districts rarely become completely worked out. Their history is usually written in short epochs marked by unusual prosperity at the beginning, gradually waning to a period of depression, at which time a superficial judgment of "dead" is pronounced upon them. The development of metallurgy has a great deal to do with these periodical revivals. At first the rich ores and simple processes are used, and in the height of prosperity which follows little attention is given to the possibility of better methods of metallurgy or to the probability of encountering more complex and refractory ores. Indeed, unless extensive exploration of the mineral area is undertaken the latter point cannot be determined in time to anticipate more appropriate methods of ore treatment, so that operations are not always continued without interruption. Hence, years sometimes elapse between revivals of prosperity.

Smelting is one of the oldest of the metallurgical arts. It is likewise of limited scope, and in the development of our Western country it has followed the lines of least resistance. To-day it is a declining industry, and in some sections has been wholly supplanted by other methods. The cyanidation and chlorination of Cripple Creek ores alone have encroached on smelting to such an extent that little support is given the smelting industry from that district. In this instance the change in process has not been due to a change in the ore, except perhaps in grade, but to the fact that cyanidation offers a better return than smelting. As matters stand now it would seem that Cripple Creek would have profited more if metallurgical research and experiment had begun years ago, before the decreasing grade of the ore made such a step advisable. This would have resulted in fewer huge dumps of low-grade ore which now demand attention, and would have eliminated the spectacular feature of high-grade shipments, but in the end the progress might have been steadier and the profit greater. With the increasing use of cyanide Cripple Creek has a prospect of 50 years of steady prosperity, whereas, depending on smelting alone, it might soon become almost an abandoned district.

Another element contributing to the rejuvenation of old mining districts is the progress in ore dressing. This applies chiefly to base ores which are beneficiated to a degree which warrants metallurgical treatment, usually by smelting. The so-called porphyry copper properties which have come into such great prominence in the last few years are the best examples of the support which concentration lends to smelting, making that process available where otherwise it would not be considered. The preparation of mixed base ores for smelting offers a more complicated problem which, however, is being successfully solved. Wet dressing and electromagnetic or electrostatic separation are proving to be the salvation of many mines and districts supposed to be in the abandoned list. In Colorado this kind of treatment is receiving more than its share of attention to-day, and bids fair to renew interest in mining, especially in the older districts.

There are enough examples of the profitable treatment of ore worth only from $2 to $4 per ton to warrant the belief that as long as any ore exists in a mine or district some method will be found to extract its metals profitably. The rôle of metallurgy is one of refinement, and there are indications that the application of electrometallurgical methods to the non-ferrous metals will increase that refinement to a degree that may eliminate the necessity for mechanically enriching low-grade ores. Each step in a process involves certain unavoidable losses, and the fewer the steps the greater the saving, as a general rule. Hence the advisability of considering the applicability of new methods before the necessity for using them arises.

Cyanamide and Electrolysis versus Cyanide in Gold Metallurgy.

The process of Mr. John Collins Clancy, described at length in this issue, represents a noteworthy departure from present cyanide practice. Since cyaniding was introduced into commercial gold metallurgy by MacArthur and Forrest some twenty years ago, many changes and improvements have been made in the cyanide process. But all of them have been more or less of a mechanical nature, in the way of arranging the apparatus, while the chemistry of the process has been little changed in its principles. In this respect Mr. Clancy's process means apparently a radical departure. He proposes to substitute cheap calcium cyanamide for part or all of the sodium cyanide and potassium cyanide now used; this means a considerable saving. On the other hand, he combines the use of cyanamide with electrolysis, and this looks at first glance like a certain complication. However, the electrolytic part of the Clancy process is absolutely different from other electrolytic cyanide processes formerly proposed or used; it may especially be pointed out here that the Clancy process has to do essentially with the solution of the gold, not with its precipitation from solution. The mechanical simplicity of the electrolytic apparatus is noteworthy. The ordinary agitating tank is used, provided with graphite and iron electrodes. While nothing can yet be said about the length of the life of the electrodes in practice, yet it would seem that mechanically there is little complication involved and that the construction of the apparatus may be as robust as desired.

The keynote of the process would seem to be the use of cyanamide. As this is a substance hardly well known to practical metallurgists, a few words may be said on the interesting and in parts amusing history of the discovery of cheap cyanamide manufacture in the electric furnace. It is an example of the fact that chemical reactions do not always occur according to the chemical equations written down on paper, and that conclusions by analogy do not always hold good. Some ten years ago or so Frank and Caro set out to make cyanides from carbides. Barium carbide reacts with nitrogen (obtained from atmospheric air) at an elevated temperature to form barium

cyanide: $BaC_2 + 2N = Ba(CN)_2$. Frank and Caro concluded by analogy that they could make calcium cyanide in the same way from calcium carbide, which is, as is well known, a commercial electric furnace product made from lime and coke. They worked out their "cyanide process" and it was discovered only afterwards that they did not have cyanide, but had found something else quite worth while finding, calcium cyanamide. The reaction is here $CaC_2 + 2N = CaCN_2 + C$. Methods of making cyanide from cyanamide were worked out, but when it was found that cyanamide could be used directly as an effective fertilizer, this side of the problem was commercially developed and the manufacture of cyanide was laid aside. There are now quite a number of cyanamide factories in operation in Europe and one in Niagara Falls, on the Canadian side. In all of them the product is a fertilizer, which has been introduced into agriculture under the trade names kalkstickstoff, chaux azotée, lime nitrogen.

It would now seem that Mr. Clancy has stepped in where Frank and Caro had stopped—with one important difference. Mr. Clancy is not out to make cyanide as a commercial final product. Being directly interested only in providing an effective solvent for the precious metals, he uses essentially a calcium cyanamide solution and with the aid of the electric current brings it into the form of an effective solvent. Spent cyanate solutions he changes back into cyanide. To characterize the process briefly, he uses a comparatively inexpensive solution to start with and produces in it by electrolysis "nacent cyanide" as an effective solvent for gold and silver. This must be understood *cum grana salis* as the reactions in the solution are somewhat complicated and both the cathodic and the anodic reactions should be separately considered to understand the process in all its phases. At any rate, Mr. Clancy has devised a most interesting new process out of the ordinary ways, and the introduction of the process into large-scale metallurgical practice deserves close attention.

A Word on Bakelite, Ions, and Beauty.

In many people's mind there is a great gulf betweeen science and art. Yet the successful work of the scientist, whether it be a chemical synthesis of commercial value, as Dr. Baekeland's bakelite, or a new feat of pure science, as Professor Millikan's isolation of the ion and his absolute measurement of the ionic charge, elicits the same feeling of highest pleasure which we get from a work of art. There is beauty in science as in art. Even more, the methods of the scientist and the artist are essentially the same. We are only liable to forget this when the product of the research work of the scientist passes from the laboratory into the realm of commerce. Thus with bakelite now being manufactured on a large scale and becoming available for commercial use, we are thinking of its immense practical possibilities; and we are liable to forget that the synthesis of bakelite is an instance of a laboratory reaction running apparently along lines parallel to those of the subtle physiological changes which in the Japanese lacquer tree engender the substance known under the name of Japanese lacquer. But the laboratory process is quick and inexpensive and bakelite, the artificial product of the laboratory, is decidedly more resisting to chemical and physical influences than the product of Nature. The same is true of Dr. Acheson's artificial graphite which, produced in the electric furnace, is purer than graphite found in Nature and therefore superior to the latter for many practical purposes. Thus bakelite, like artificial graphite, are two examples showing that for the purpose of man the product of human endeavor may be superior to the product of Nature.

Of course, man cannot really improve on Nature's processes or methods. Nothing more is possible than to imitate or copy Nature's processes and to combine them for a certain end. But because a certain product is the goal kept in view it is possible to select the directest process of production and to produce only what is wanted. Nature does not set out to make pure graphite. Man can. Selection of those starting materials which are needed and nothing else and selection of only those processes of Nature which are essential for his purpose—these are the methods followed by the chemist in his synthesis. Now, this is exactly what the artist does—the sculptor, the painter, the poet. The artist must take Nature as it is. But for his work of art he takes only what is essential to the idea he has in mind. It is this selective copying of the essential from Nature and this absolute restriction to nothing but the essential that make a work of science or a work of art beautiful. It is for the same reason that Professor Millikan's lecture on the last day of the recent Chicago meeting of the American Electrochemical Society on his method of isolating ions and measuring their charge became one of the big features of this very successful convention. It is true, Professor Millikan's method is in the last analysis a modification of Wilson's cloud method, but by the details in which he modified it, to the final substitution of oil drops for water drops, he has selected the most direct imaginable way of getting from Nature an answer to one of the most fundamental problems of physics—the corpuscular nature of electricity and the value of the ionic charge. The method is now simplicity itself, free from hypothesis, free from anything that is not essential. And this beauty of the method is equaled by the accuracy and fundamental character of its results.

It has been truly said that while proper equipment of a research laboratory is important, yet the primary essential requirement is the chemist himself with a true spirit of research. Just as in the sculptor's workshop the sculptor is the one essential element, and marble and model and chisel are secondary. What counts in science, in industry, in art, in all progress of the world, is, after all, the man as man. Because this is universally true, it will not be out of place to quote here a few sentences on the essential elements in man from a book which has nothing to do with chemistry or engineering, but which has considerable to do with the seeking after truth—Gustav Frenssen's Hilligenlei: "We really do not require anything superhuman. We do not require men to go out to seek an empire. But while they are running after money and titles and other errors we require them to have a hope that they might find on the next meadow instead of a herd of sheep a gathering of angels; and we require them to have a restlessness expecting that they might meet at the next corner of the way, under the oak tree, the eternal being that holds quietly in holy hands the thousand miracles of all worlds and might give the solution of a few of them. This we require. For this is essential in a real man."

International Congress of Applied Chemistry.

At a meeting held on October 8, in the hall of the Chemists' Club in New York City, the presidents and vice-presidents of most of the sections of the International Congress of Applied Chemistry, to be held in this country in 1912, were elected. A few elections were held over. For the electrochemical section, Dr. Wm. H. Walker, of the Massachusetts Institute of Technology, was elected president; Prof. Chas. F. Burgess, of the University of Wisconsin, vice-president; for the metallurgical section, Dr. Jos. W. Richards, of Lehigh University, was elected president; Mr. J. B. F. Herreshoff, of the Nichols Copper Company, vice-president. When the organization of all the sections is completed a full account will be given in this journal.

* * *

Dr. Bernhard C. Hesse, who is the Secretary of the Congress, presented a suggestive address at the October meeting of the New York Section of the American Chemical Society on the relation of the American Chemical Society and its local sections to the International Congress.

He first emphasized that the utility of the Eighth International Congress of Applied Chemistry will be judged, determined and measured largely by the printed record of its deliberations and conclusions. It is essential that the printed record shall truthfully and accurately reflect the activities of the Eighth Congress.

That these activities of the Congress shall properly and completely represent the then condition of applied chemistry over the whole world is perhaps the main task of the Eighth Congress.

It is necessary that everything pertaining to applied chemistry in the United States which can be properly reported at the Congress should be reported.

In the last analysis it is the individual member who must carefully search his mind and his surroundings and ascertain whether or not there is something, no matter how small, which would be material to a correct reflection of the actual condition of applied chemistry in the United States and to report the same so that it may be properly considered.

The chemists interested in fats and oils have already taken steps to be properly represented. "Let the metallurgists, let all the chemists interested in the manufacture of heavy chemicals, and those interested in the manufacture of fine chemicals, and all the chemists of every other distinctive group of common interest come together and see to it that their interests are properly placed before the committees of the Congress having their interests in charge."

At the Congress a large number of interpreters will be needed with considerable chemical knowledge. There are four official languages used in the Proceedings and Transactions of the Congress, namely English, French, German and Italian. It is desirable to have each section and sub-section provided with chemists who would act as interpreters. Interpreters will also be needed at the Information Bureau.

Further, should it be feasible to print the papers in advance, a large number of proofreaders would be needed. These will also be needed for the daily bulletin and the minutes of the daily meetings of the various sections.

"The fact that the actual holding of the Congress is now almost two years off should not deter any one from at once offering his information and suggestions or volunteering his services whether as a source of information for a certain locality or as any other aid. A postal card notice will be sufficient. The more promptly the Organizing Committee is fully and completely informed as to what it has to provide and as to the individuals, societies and groups of men upon whom it can count and what each can or will do, the more expeditiously can that committee proceed with its work and the more closely will it approach to the complete realization of the objects and purposes of the Congress."

The offices of the Secretary of the Congress, Dr. B. C. Hesse, are at 25 Broad Street, New York City.

The Western Metallurgical and Chemical Field.

Cement Plaster.

One of the important chemical industries of Wyoming dealing with non-metallic minerals is that of the manufacture of cement plaster from gypsite. In the vicinity of Laramie, Wyo., there occur large deposits of gypsite, or earthy gypsum, being a mechanical deposit from the disintegration and erosion of rock gypsum. In the Laramie beds the gypsite occurs in a very finely pulverized condition. After stripping a slight overburden, the bed is worked with disk harrows to allow a part of the moisture to evaporate. It is then hauled in wheel scrapers to the storage sheds, taking only a comparatively thin layer of material so as to provide a uniform mixture for treatment.

The raw gypsite is charged directly into calcining kettles of 10 tons capacity. During the process of heating the material is kept in constant motion, and the temperature is gradually raised to about 400 deg. C. At about 220 deg. water begins to come off and the mass boils violently. Another period of slight boiling occurs at about 290 deg., after which the mass is comparatively quiet. The final temperature is between 340 and 396 deg., being determined by the appearance of the charge and the use of thermometers.

The kettles are discharged into storage hot pits, whence it is removed by spiral conveyors and elevators to the screen line, where about 1 or 2 per cent of coarse material is removed, reground and returned to the screen. The small quantity requiring grinding is due to the naturally fine state of the original gypsite. From the screen the plaster is conveyed to storage bins from which it is weighed in weighing hoppers, a weighed quantity of retarder added, and sacked. The finished mixed plaster is continually tested for setting time and quality. The operations described herewith represent the process at the plant of the Overland Cement Plaster Company, Laramie, Wyo.

Matte Smelting.

The smelter of the Colorado Gold Mining & Smelting Company, at Alma, Col., is the latest semi-pyritic plant blown in in this State. It is a plant of 150 tons daily capacity, with a modern matting furnace 42 x 120 in. at the tuyeres, there being nine 4-in. tuyeres on each side. The company owns the Kentucky Belle and Atlantic mines from which it receives about 30 tons of ore daily, and has leases on some other properties and dumps which, together with small lots of custom ore from the district, make up its total ore supply. The following average analyses of some of the ores show the nature of the material available.

	Ky. Belle	Atlantic	Moose	London	Old Slag
SiO_2%	45	20	20	77	35
Fe	15	45	2	4	33
Al_2O_3	20	—	—	11	—
CaO	—	—	20	1	10
MgO	—	—	12	—	—
S	8	22	—	—	—
Cu	1	0.2	—	—	—
Au, oz.....	0.6	0.2	—	0.7	—
Ag, oz.....	2	2	—	—	—

It will be seen that there is a variety of silicious, sulphide and lime ores from which to make the charge, which is made up as follows: Kentucky Belle, 600 pounds; Atlantic, 350; Moose and London, 250; slag, 600; flue dust, 200; matte, 50; limestone, 200; coke, 225.

The slag produced analyzes as follows: SiO_2, 38 per cent; FeO, 28; CaO, 14; Al_2O_3, 9. The matte carries 4 to 5 per cent copper, 4 to 5 oz. gold, 30 oz. silver, 60 per cent iron and 3 per cent silica.

Electric power generated at the plant is used throughout; limestone is obtained from quarries in the vicinity.

Consolidated Mercur.

In the annual report of the Consolidated Mercur Gold Mines Company, President John Dern makes a frank explanation of

the comparatively unfavorable situation in which the company finds itself after a year of exceptionally hard work to meet increasing difficulties. The company operates mines and cyanide mills at Mercur, Utah. Owing to the decreasing grade of ore it has been difficult to operate at a profit, and at times operations have been conducted at a loss. The average value of the ore treated this past year was about the same as last year, but there was a decreased tonnage in the hope of raising the value of the ore sufficiently to bring the profits up to normal.

The extraction was not as high as last year, due largely to the extra quantity of base ore treated, and to the refractory nature of the oxidized ore from the Brickyard mine. It has been impossible, even in the laboratory, to get as low tailings on this ore as formerly.

The report shows that 231,631 tons of ore were treated during the year, of an average total value of $3.59 per ton. The average of the tailing for the year is $0.94 per ton; expenses, $2.63 per ton. The total gold production was 29,900 oz. fine gold, the cost of production being $20.376 per oz., giving a net profit of 29.4c. per oz.

The Consolidated Mercur management has always been progressive in adopting the latest scientific achievements in the treatment of ore, and it is probable that some process will yet be worked out which will apply to its present conditions.

California Smelters.

The conflict between the smelting and agricultural interests in Shasta County, California, is proving disastrous to the former, and the output of the State in copper and silver will be materially reduced this year. The United States government is also engaged in actions against smelting operations where the resulting fumes are said to be damaging national forests. The Bully Hill Copper and Smelting Company has definitely closed its plant and curtailed mining operations on account of the government's activity. This is due directly to the government's ultimatum which provided that fume condensing devices must be installed before July 1, or an injunction would be sought to restrain smelting operations. As it was impossible to comply with these terms, the plant was closed when the date mentioned arrived.

The smelter at Coram is still out of commission on account of the determination of the agricultural element to prevent operation until the Cottrell system of electric condensation is completed, which will require several months. The Mammoth smelter of the United States Company is operating at reduced capacity with bag houses for fume condensation. This plant is interesting in view of the application of the bag house to the filtration of gases from copper furnaces. The bag house has long been operated in lead smelters, but it has not been successfully tried in copper plants until lately. The United States Company built a bag house which it was thought would suffice for its four furnaces, but it has been found that the capacity is sufficient for only two furnaces and this number only is in operation. The management is reported to be satisfied that with additional bag area the process will satisfy the requirements of the plant. The problem is largely mechanical in its nature and will be worked out.

Utah Copper Mills.

The Ohio Copper Company's mill at Lark, Utah, is handling a remarkable tonnage of ore, considering its condition and original estimated capacity. Owing to financial difficulties the mill has been equipped only in one section, which represents half the capacity of the plant. The estimated capacity of this section was 1000 tons of ore daily. Recently the tonnage treated in this section was a little more than the estimated capacity of the entire mill, or 2096 tons. When the mill is completed, as it is expected it will be this year, it will take its place among the large copper producers which have attracted so much attention during the last two years.

The plant of the South Utah Mines and Smelters Corporation, at Newhouse, Utah, is treating 600 tons of ore daily, and shipping one car of copper concentrates daily to the International Smelter at Tooele. It is expected that when the plant is in full operation under the improvements that are being made it will treat about 800 tons daily.

The Tooele smelting plant of the International company has been completed and four reverberatory furnaces are in operation. As these meet the present needs of the plant, the fifth will not be used for some time. It is generally believed that no copper smelting plant has been constructed and placed in commission with so few delays, and so nearly to the pre-arranged program as to time and cost as this one.

Utah Consolidated Tramway.

The tramway between the Highland Boy mine and the Tooele smelter is now completed and in operation. It is four miles in length with a difference in elevation of 1300 feet. Seventy-nine towers, varying in elevation from 10 to 90 ft.; support the line. Two hundred and twenty-five buckets of 8 cu. ft. capacity, weighing, with load, approximately 1150 lb., move at the rate of 600 ft. per minute. The line is operated in three sections, the first two being motor driven with 20 hp motor, and the last being operated by gravity, generating 75 hp, giving a surplus of power over that required in the first two sections. The cost of transportation is about 2c. per ton.

The Iron and Steel Market.

October, the last of the fall months in which a general buying movement in steel products could normally be expected, has brought no increase in buying except in wire products, and the wire activity is presumably due to belated buying because buyers did not take hold at the usual seasonable time.

The railroads have continued their position of being strictly out of the market except for absolutely necessary requirements. This position they will undoubtedly maintain until the freight-rate issue is settled, and this is expected next February. There have been reports that a private agreement has been reached by which the railroads will be given about half the advances they have asked for, but nothing official has, of course, been done, and if the report is correct, the Interstate Commerce Commission will have the great task of deciding which advances will be allowed and which not, of the great mass of advances for which notices have been filed and which have been suspended temporarily by the commission.

In Wall Street there has been a much better feeling and the steel trade is disposed to accept this as an augury of better things, but nothing is in sight nearer than the railroad buying expected after the rate controversy is settled. The better position in Wall Street has not materially encouraged business, such as bridge and building work, depending upon financial arrangements.

Production is slightly on the down grade, the recession being due to the gradual exhaustion of the accumulations of orders for various heavy materials, particularly rails, plates and shapes. In the lighter lines orders have been comparatively good, and, on the whole, substantially equal to the production. In wire products buying has run slightly ahead of production, so that occasionally mills have fallen somewhat behind in deliveries. In sheets, the next most active line, the enormous increase in productive capacity in the past few years tells very seriously, for there is no question that the demand in point of tonnage is well in excess of that in 1907. Since that year, however, new sheet mills have come in at the rate of about 30 per year, some as new plants and others as extensions to existing plants, largely by interests which make their steel and, therefore, feel in a strong position to meet any possible competition. The total increase in sheet rolling capacity, through the building of new mills and harder driving of ex-

istic mills, has been between 300,000 and 500,000 gross tons annually in the past three years.

Finished steel prices are being maintained quite well all along the line, but through the tacit understanding that there is no advantage in cutting rather than because the mills are comfortably filled with business.

Pig Iron.

Pig iron prices continue to waver at points and occasionally to drop. Basic iron has become quotable at $13, valley, or 50 cents less than at last report, and it is by no means certain that this price cannot be shaded. In all markets the tendency to demand a material advance for first quarter or first half over prompt prices is decreasing. A sale of 6000 tons of malleable iron, for instance, has been made for equal deliveries through 1911 at $14.90, delivered Pittsburgh. The valley market, 90 cents higher delivered Pittsburgh, is quotable at $14 for No. 2 foundry, $13.50 for malleable, $13 for basic and $15 for Bessemer, these prices being for early delivery.

Steel.

The market has been very quiet, but through offerings can be quoted 50 cents to $1 less than at last report, or at $23.50 for Bessemer billets, $24.50 for Bessemer sheet bars, $24.50 for open-hearth billets, and $25 for open-hearth sheet bars. Rods, however, are firmer at $28.50, Pittsburgh.

Finished Products.

Prices are substantially unchanged.

We quote the market as follows, Pittsburgh, unless otherwise noted

Standard steel rails, $28 at mill, except Colorado.

Plates, $1.40 for wide, $1.35 for narrow, tank quality, ¼ in. and heavier.

Shapes, $1.40 for beams and channels, 15 in. and under, zees and angles 2 in. x 3 in. and larger.

Steel bars, $1.40, base; iron bars, $1.45, Pittsburgh; $1.35, Chicago; $1.35 to $1.40, Philadelphia.

Wire nails, $1.70, base; plain wire, $1.50; painted barb wire, $1.70; galvanized barb wire, $2.

Black sheets, 28 gage, $2.15 to $2.20; galvanized, $3.15 to $3.20; blue annealed, 10 gage, $1.60 to $1.65.

Tin plates, $3.60 for 100-lb. cokes.

The Non-Ferrous Metal Market.

During the past month there has been little change in the status of the base metals. The reduction in domestic stock of copper has had a stimulating effect on the market for that metal. Tin has commanded a premium for spot deliveries, but the tone of the general market has not been strong, as evidenced by the reshipment of 200 tons to London. Lead has been very steady, but zinc was weak and the demand was so light that it is believed that stocks are accumulating in the hands of the smelters

Copper.—The figures of the Producers' Association for September showed a decrease of copper stocks of 20,087,531 lb., and this had the effect of stimulating action among the dealers and speculators Lake copper is quoted at 12¾ cents to 12⅞ cents, with electrolytic at 12.5 cents and 12.55 cents.

Lead.—The market has been steady at 4.40 cents, New York, with slightly lower prices ruling in the West on account of large Western offerings At St Louis the quotations are 4.25 cents to 4.27½ cents.

Tin.—The shipments of Straits tin to this country and other markets have showed a decrease under last year Business is most active among speculators and dealers, and spot tin has commanded a premium of about 1 cent October tin was available at about 36½ cents

Zinc.—Business has been fair, but not so voluminous as some time ago While the smelters may be holding accumulating stocks, the expectation for a better future market is strong and they are not urging immediate sales. The New York market was 5.52 cents to 5.55 cents, New York, and 5.37 cents to 5.40 cents, St. Louis.

Other Metals.—Business has been only fair in aluminium, antimony and mercury, with prices almost nominal. Aluminium is offered plentifully at 22 cents, and sales have been effected at a lower rate. Antimony prices are unchanged and range from 7¼ cents to 8⅜ cents for various grades.

CORRESPONDENCE

Determining Thermal Conductivity

To the Editor of Metallurgical and Chemical Engineering:

Sir:—In the advance copies of my American Electrochemical Society paper, presented at the Chicago meeting, there are some misprints in the formulas, which should be corrected by those who wish to use them.

Page 8, in the second formula, there should be decimal points in the places left for them, namely, to the right of the first figures. Page 9, in formula 5, the n should be an r, and in formula 6 there should be a square root sign before the n.

Philadelphia, Pa. Carl Hering.

Autogenous Welding of Thermo-Couples by Direct Fusion.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—In industrial thermo-electric pyrometer installations it is universally recognized that the fusion of the wires forming the hot junction of the thermo-couple is indispensable, as otherwise the prolonged heating which these couples have to undergo in permanent installations has the effect of causing the wires to become loosened, if they are merely twisted together, resulting in bad contacts and subsequently incorrect readings.

The ordinary method adopted for this fusion of couple wires used at high temperatures necessitates the use of compressed oxygen, or if the alternative method of soldering with palladium is used, compressed air is essential.

Fusing by the above methods is not always convenient, and occasionally circumstances arise where the adoption of either is impracticable by reason of either oxygen gas or palladium metal being difficult to obtain.

An alternative practical method for direct fusion adopted by the writer under these circumstances was to take advantage of the high temperature of the electric arc. To conduct the fusion the following method was adopted:

The couple (removed from its insulating and protecting sheaths) was clamped in a stand and the ends to be fused were twisted for a length of about 5 mm. Contact of the free ends of the couple were made to one terminal (A) of an electrical circuit containing a switch, while to the other terminal (B) was connected a short length (30 cm) of platinum wire subsequently used to form the arc.

To effect the fusion the switch was closed and the platinum wire attached to the B terminal was brought in contact with the twisted ends of the wires of the thermo-couple, being then instantly removed to a sufficient distance to strike the arc and then immediately withdrawn.

If the heating effect of the arc is allowed to continue too long, the twisted wires will fuse completely In this event the beads of metal should be removed and the operation repeated.

The operation of fusing is quite instantaneous if successfully performed A sufficient length of the twisted wires is fused by once striking an arc, and after a little practice the work can be effected without difficulty. The strength of current used was 1.8 amp at 440 volts pressure. The couple after annealing was calibrated, the results being entirely satisfactory.

By the help of borax as a flux platinum and copper leads may be fused by electric arc.

Ishapore, Calcutta, India Percy Broadbent Crossley

Co-operation and Business Stability.*

BY ELBERT H. GARY.

One of the most important business matters at this time, and perhaps at all times, is the maintenance of stable conditions in the iron and steel industry, and to this is attached the question of prices. For periods, at least in the past, unsatisfactory conditions have been the rule and not the exception. Violent, sudden and wide fluctuations in prices have occurred and they have been detrimental to every one connected with the business, and to the public generally. The steel and iron industry, more or less recognized as the barometer of trade, has had an influence on general conditions for good or bad, and much of the time for bad.

We do not, and never have, advocated fixed, unchangeable prices. If they are unreasonably high they should be lowered, and vice versa. If for any good reason at any time we are convinced that the rights and interests of ourselves, our employees or our customers demand decreases in selling prices or the readjustment of prices we should be prompt to make them. This does not, however, militate against the general proposition in favor of stability as opposed to demoralization. We know, as a matter of fact, from most careful observation and an abundance of testimony from those directly affected that the maintenance of fair prices and the prevention of sudden and wide fluctuations are desired by our customers themselves.

If you suggest that many members of this Institute are purchasers or consumers of the commodities produced by others, and, therefore, may object to the position taken, the obvious answer is that they do not object. Many of them purchase with the view of converting or refining and selling the product. Many have stocks of materials on hand from time to time and cannot afford, and do not desire, to have the values of inventories reduced. Besides, consumers like to feel that they can depend upon prices which they pay as being as low as the prices which their neighbors subsequently pay. We are all purchasers and consumers of one thing or another. We sell certain manufactured products, but we buy a greater variety of products; and we like to feel that we are at all times treated as our neighbors are treated.

If we go to the tailor, the butcher, the grocer, the farmer, the railroad company, the steamship line, the telegraph office, the newspaper office, or any others having for sale that which we need and buy, we are better satisfied if we are certain we are treated like our neighbors and that there will be no sudden and material decreases immediately after we have bought and before we have had opportunity to consume the product which we have purchased.

It is pleasing to note that in much of the business transacted in this country, commercial, financial and industrial, there is a tendency toward an even, steady and continuing basis of action. It would be better for us all, and better for the country, if conditions in all departments of business were more stable and less likely to be disturbed by agitation, doubt or distrust.

It has sometimes been urged that an effort to maintain reasonable prices for iron and steel is contrary to the natural laws of trade; that it is opposed to the law of supply and demand. This we do not admit. We do not undertake or advocate anything which is unreasonable, unnatural or unfair.

Those who oppose the maintenance of fair prices when the volume of business is low on the ground that it is contrary to the law of supply and demand make the mistake of supposing that capacity to produce creates supply. The law of supply and demand means the quantity which one possesses and is willing to dispose of, and the quantity which another desires to possess and is willing to buy. Capacity to furnish does not create supply, nor does ability to purchase create demand.

It is just as illogical to say that because one is able to furnish a given quantity that is evidence of the supply as it would be to say that because another can use and has money to pay for a certain quantity that constitutes a demand. The whole question should depend upon mutual consideration and decision. The seller should never take advantage of the necessities of the consumer to unreasonably advance prices, nor should advantage be taken of the necessities of the producer to secure this product at an unreasonably low price.

What then should the manufacturer do at times when the consumers or purchasers are not willing to pay the price demanded, however reasonable it may be? Many of us at least believe he should reduce the output to meet the demand, much or little, and wait until those in need are willing to pay what is fair and right. Remember this assertion is based on the assumption that prices at all times must be fair and reasonable and subject to reductions or changes whenever it is decided they ought to be made. Such reductions, however, should be made only after due consideration, and in an orderly way and not as the result of panic.

On the other hand, if prices are at any time higher than they ought to be, the consumer, whatever his pecuniary ability to pay may be, should wait until the prices are reduced. If the attitude of both seller and purchaser should be as suggested, it would not be long before they would get together on a proper basis, especially if what we advocate shall be carried into actual effect.

What has been said is outside of, and without reference to, any question of special privilege, or undue advantage, or improper control of any commodity which would enable the producer from motives of cupidity or otherwise to fix or maintain unreasonable prices, or to take advantage of the necessities of the consumer. The great aggregations of wealth make this question important, but it may not be appropriate to discuss the same at this time.

Now, how shall we bring about the conditions which have been suggested as ideal?

The existence of this Institute and the large attendance on this occasion furnish the answer.

Real, hearty, cheerful and continued co-operation on the part of the members will secure results which should be entirely satisfactory. Frank and friendly intercourse; full disclosure of his business by each to the others; recognition by all of the rights of each; a disposition to assist and benefit each other so far as practicable and proper; conduct founded on the belief that healthy competition is wiser and better than destructive competition; all these are desirable and necessary and will be effective.

Some of you may say that in your opinions, based on the past, a fair and just equilibrium cannot be maintained except as the result of some agreement on the part of those who are interested. It is a sufficient answer to make that you have no right to enter into such an agreement, express or implied, that the laws of this country are opposed to it.

But, as stated by the Attorney-General of the United States in his argument before the Supreme Court in the Standard Oil case, the law does not compel competition; it only prohibits an agreement not to compete. If competitors are in frequent communication and make full disclosure to each other in regard to their business, notifying one another of what they are doing, it will follow as a natural result that no one will take advantage of the information he thus receives to act unjustly or dishonorably toward his neighbor. You know by experience that this is so, and it will become more certain as time goes

*Extracts from the presidential address of Judge Gary, chairman of the board of directors of the U. S. Steel Corporation, before the first formal meeting of the American Iron and Steel Institute on Oct. 14, 1910. Judge Gary's address began with words of welcome to the members of the Institute and visitors from abroad. Statistical data was given on the growth of the American iron and steel industry, with a forecast of future growth. A sketch of the proposed program of the work of the American Iron and Steel Institute was then given, leading to the principal subject of the address—maintenance of stable conditions by means of "co-operation." Only this part of the address is given above, but this part is given in full.—EDITOR.

by and each of us realizes more fully his obligations to others and to this Institute.

Co-operation is broadening in its application to specific subjects growing out of the activities of life. We see it in many lines of business outside of our own. We see it in this country and in foreign countries. Especially have we observed it during the last few months in the consideration of some of the matters which have demanded the attention of the Interstate Commerce Commission and other official departments.

Co-operation can be, and should be, extended to our business relations with foreign manufacturers and producers. They will meet us half way. Indeed, the presence at this meeting of so many distinguished and representative men from Great Britain, Germany, Austria, France and Belgium, respectively, is positive proof that they believe in this Institute and are friendly to its members, and have high appreciation of the efforts we have been making in the direction of conciliation and co-operation.

Whenever and however we may have opportunity to meet these gentlemen and to legitimately co-operate with them in promoting and conserving their interests as well as our own, we should do so without hesitation. We must at all times evidence a disposition to reciprocate the friendship which these gentlemen have shown, and to demonstrate to their satisfaction that we believe thoroughly in the doctrine that we are more or less bound together by business connections, and that we enjoy prosperity when they enjoy it, and that we fail of success when they fail.

Many who are present will live to see the time when the spirit of co-operation shall extend even to the government of the nations of the earth.

Again, there is no better way of advancing our own interests than to conduct ourselves in such a way as to make it certain we are endeavoring to promote the welfare of others with whom we come in contact. Uppermost in our minds should be the proper consideration of the rights of our employees.

Members of this Institute represent corporations or firms who have in their employ hundreds of thousands of men. Many of these workmen are foreigners who are not familiar with our language or customs.

Some of them have been poor and have had few advantages in their native countries, and may not be fully able to protect themselves.

Whether this be true or not we should make it certain that all are so treated that they have the opportunity to be well housed, and well fed, and well clothed. So far as we can we should lend our assistance to the improvement of the conditions of our workmen, and, of course, the wages paid should always be fair and reasonable. All of us have discovered since this Institute was formed that there were many things which we could do for the betterment of the conditions of the workingman, and we have been prompt to do them. Particularly have we made great strides toward safeguarding our employees against accidental injury.

A continuation of the efforts we have been making in this direction will result in securing a larger measure of confidence on their part, and will bring to us the consciousness that not only have we benefited ourselves pecuniarily, but, what is more important, that we have done the right thing.

Also, we should extend the spirit of co-operation to our customers. We should be sure at all times that we are treating them right. If it is claimed on any occasion by a customer that our prices are too high with respect to any commodity, we should promptly take up with him the question for the purpose and with the intention of arriving at a just conclusion. In all cases we advocate the doctrine of co-operation as opposed to independent action. We have found, particularly during the last two or three years, that we are succeeding better as we are brought into closer relations with our customers, giving them our confidence and informing them frankly of our conditions and intentions.

Copper Precipitation in the Butte District.

Editorial Correspondence.

There are precipitated each month in the Butte district about 700,000 lb. of copper from mine and tailing waters. Of this quantity about 550,000 lb. are recovered from copper-bearing mine water, and 150,000 lb. from water percolating old tailing dumps.

FIG. 1—SERIES OF TERRACED PONDS USED IN LEACHING OLD TAILING DUMP

The adjoining illustrations, Figs. 1 to 5, show the manner of leaching and precipitating at the Red Metal precipitating plant on the tailing dump of the Montana Ore Purchasing Company's old concentrating mill.

The original tailing carried about 1 per cent copper and was deposited on a bed of slime which provides a very effective impervious stratum preventing the loss of leaching water.

The dump is divided into two sections and has a total area of about 10 acres. The surface of each section is arranged in terraced ponds, as shown in Fig. 1, into which the leaching water is pumped by means of Morris phosphor-bronze centrifugal pumps. On the larger section, one 20-hp pump elevates about 700 gal. per minute to a height of 30 ft.; and on the smaller, a 15-hp pump elevates 400 gal. per minute to a height of 25 ft. Both pumps are motor driven and require but little attention. The leaching water used at this plant is mine water

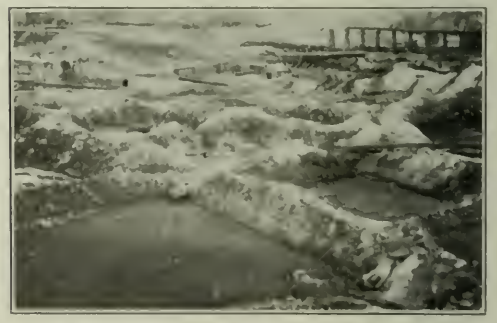

FIG. 2—PONDS DRIED TO REMOVE ACCUMULATED IRON SALTS

which has been treated previously in a precipitation plant where about 95 per cent of its copper content has been recovered, leaving from 0.003 per cent to 0.004 per cent copper.

ANALYSIS OF MINE WATER USED IN LEACHING.

Constituent.	Grams per Liter.
SiO_2	0.0092
CuO	0.0225
Al_2O_3	0.1241
Fe_2O_3	0.1541

FeO	1.5082
MnO	0.0502
ZnO	0.4257
CaO	0.3936
MgO	0.2172
SO_3	3.5400
Cl	0.0234

The percolating water is collected in tunnels extending through the bottom of the dump, and is finally discharged into a sump whence it is pumped to the precipitating launders. On the larger dump the tunnels are about 200 ft. apart and

FIG. 3—BUTTE MINING DISTRICT. COPPER PRECIPITATING LAUNDERS IN FOREGROUND

have an aggregate length of 1200 ft. They are 5 ft. high, 3 ft. wide on the bottom and 2 ft. wide at the top, and timbered throughout. On the smaller dump there are but 800 ft. of tunnels. The number and length of tunnels is dependent on the nature of the ground on which the dump is formed. Porous ground which would permit heavy loss of water by seepage would require more collecting tunnels than an impervious foundation stratum. At this plant the loss by seepage amounts to about 20 per cent.

As shown in Fig. 2 a section of the dump surface is allowed to dry out periodically, so that the accumulation of precipitated iron compounds may be removed and the dump material allowed to oxidize before further leaching. The rapid accumulation of iron compounds is due to the fact that the mine

FIG. 4—PRECIPITATING LAUNDERS, GRADE ½-INCH PER FOOT

water used carries considerable iron from previous treatment, and on exposure to the atmosphere deposits peroxide and basic sulphate of iron. This accumulation prevents leaching and is occasionally scraped from the surface of the pond. By this treatment the dump yields a constant supply of copper, this particular dump having been leached continuously for four years with no apparent diminution in yield.

The leaching water collected through the tunnels carries from 0.015 per cent to 0.03 per cent copper. This is not as high grade as some of the mine waters, but is profitable. Figs. 3 and 4 show the launder system now preferred for precipitation rather than the tower system as shown in Fig. 6. The advantages lie in convenience of operation, particularly in charging the launders with scrap iron, and in cleaning up the copper precipitate. The launders are 4 ft. wide and 1 ft. deep, and preferably have a fall of ½ in. to the foot. Launders of this size and from 600 ft. to 700 ft. long will serve to precipitate successfully 400 gal. of copper-bearing water per minute. At the Red Metal plant the launders are 700 ft. and

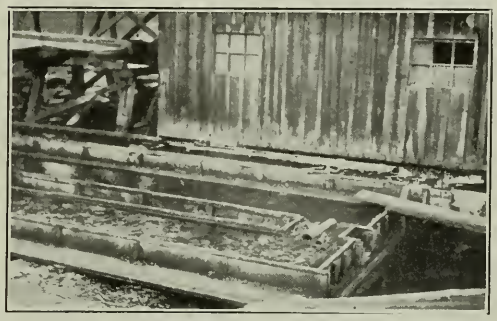

FIG. 5—COLLECTING SUMP PLACED AT THE END OF LAUNDERS

512 ft. long, respectively, on the larger and smaller dumps.

Scrap iron and tin cans are used for precipitation. Paper must be removed from the cans, and if greasy they must be burned before use. The Red Metal plant has about 75 tons of scrap in constant use, the launders are replenished with about 1000 lb. daily. Iron is consumed at the rate of 1½ lb. for each pound of cement copper produced. The precipitated copper is swept from the iron every two hours during the day, and is carried to the sump (Fig. 5) by the stream of water flowing through the launders. Thus, no handling of the iron is necessary after it has been placed in the launders. From the sump the copper is shoveled onto a drying platform and is shipped to the refinery with about 15 per cent moisture.

The recovered precipitate assays about 75 per cent copper

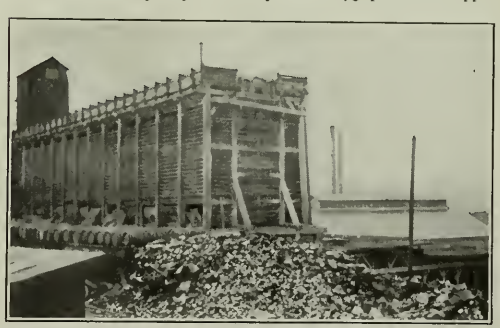

FIG. 6—OLD-STYLE TOWER PRECIPITATING PLANT AND PILE OF TIN CANS AND SCRAP IRON

and 8 per cent iron, with smaller percentages of other impurities. About 90 per cent of the copper in solution is precipitated in the system described. The richer the solution the more efficient the precipitation. Scrap iron is paid for at the rate of $8 per ton, although as high as $12 has been paid when copper was selling at 25 cents. Payment for the precipitates is made on the wet assay, less 1.3 per cent, and at 4 cents less than the market price for copper.

Power costs $6 per hp-month. Labor cost is low, as two day laborers and one night watchman only are required at this plant, in addition to the superintendence of Mr. J. M. Gillie, to whom we are indebted for the data given herewith.

Analysis of Titaniferous Iron Ores.

By W. D. Brown.

For a long time titanic acid has been the bugbear of the furnace manager and of the chemist. However, any detriment to the furnace working due to titanic acid is largely overcome if the amount present is accurately known and the necessary precautions taken. Rossi and others have shown that titanic acid may be used in a blast furnace and fluxed as readily as silicic acid.

However, the presence of titanic acid cannot be disregarded in calculations for slags and in their analysis. Many methods are in existence for the determination of titanic acid. However, the writer has seen but little literature treating of the complete analysis of an iron ore containing titanic acid. Titanic acid if disregarded in an analysis remains with the silica or is precipitated with the alumina.

The writer has used the following method, analogous to the ordinary phosphate method, for the determination of the alumina and titanic acid on the same sample. In this method titanic acid is precipitated by a modification of the well-known method of Baskerville (*Journal*, American Chemical Society, XVI, page 427). However, it was found that this precipitate contains some alumina. This is recovered, and the total alumina precipitated as phosphate.

This method applies to the so-called non-titaniferous as well as to the titaniferous iron ores of which there are vast deposits which have not come into use because of their high content of titanic acid, notwithstanding the fact that they have a high content of iron. Probably none of our commercial iron ores are entirely free from titanic acid. The lake ores investigated, and in daily use, contain from 0.05 per cent to 0.30 per cent of titanic acid. As the percentage of iron in the ore decreases and the amount of gaunge increases, we may look for an increase in the titanic acid present, as it is present in clays to a considerable extent.

The Method of Analysis.

Silica.—One gram of the ore is digested in a porcelain dish in 25 c.c. of strong hydrochloric acid, and the solution filtered into another porcelain dish. The residue is fused, the fusion returned to the dish containing the filtrate from the insoluble residue, and the solution is evaporated to dryness on the steam bath. The residue is moistened with dilute hydrochloric acid (equal parts acid and water), water is added and the solution is brought to a boil and filtered into an 0.5 beaker. The precipitate is washed until free from chlorine, all the washings being caught in the beaker. The washings may appear cloudy, due to the running through of the titanic acid. This is disregarded as it is retained by the precipitation subsequently to be made. The silica on the paper contains a part of the titanic acid and after burning and weighing is volatilized with hydrofluoric and sulphuric acids. The crucible is ignited and weighed, the residue being preserved for the recovery of the titanic acid. The difference between the first and second weights of the crucible is silica.

Manganese, Lime and Magnesia.—A basic acetate precipitation is made in the filtrate from the silica. The titanium present is completely precipitated with the iron and alumina, the writer having found none in the filtrate. The basic acetate precipitate is dissolved by washing with hot dilute hydrochloric acid and hot water. Titanic acid dissolves with difficulty from the paper, the writer having found as high as 0.05 per cent remaining with the filter paper; this is recovered by burning the paper with the impure precipitate of titanic acid subsequently obtained. If the manganese is above 1 per cent in the ore, a second basic acetate precipitation is made in the hydrochloric acid solution of the first precipitate. The manganese lime and magnesia are determined in the filtrate or combined filtrates, as described in "The Methods of the United States Steel Corporation for the Commercial Sampling and Analysis of Iron Ores" (this journal, Vol. VII, page 65).

Alumina.—The hydrochloric acid solution of the basic acetate precipitate is diluted to 300 c.c. and ammonium hydroxide added until a slight permanent precipitate is formed. One and one-half c.c. of strong hydrochloric acid are now added, then 50 c.c. of a 20 per cent sodium thiosulphate solution, followed by 10 c.c. of acetic acid. The solution is boiled 10 minutes, the titanic acid being completely precipitated, accompanied, however, by a part of the alumina and phosphoric acid and a small amount of iron. The amount of alumina varies with the phosphorus and inversely with the acidity, as high as 0.75 per cent having been found in the precipitate from an ore containing ½ per cent of phosphorus and 5 per cent of titanic acid. After allowing a few minutes for the precipitate to settle, the solution is filtered into a No. 5 beaker, and the precipitate washed with a hot 2 per cent acetic acid solution.

To the filtrate 30 c.c. of a 10 per cent solution of ammonium phosphate and 1½ c.c. of strong hydrochloric acid are added at the same time, then 30 c.c. of thiosulphate solution are added. The solution is heated to boiling, at which point 15 c.c. of a 20 per cent solution of ammonium acetate are added, and the solution is boiled 10 minutes. The precipitate of aluminium phosphate is allowed to settle, filtered off and washed slightly with hot water. It is dissolved on the paper with very hot dilute hydrochloric acid into the beaker in which the precipitation was effected. This solution is reserved for a second precipitation after the addition of the alumina recovered from the preliminary titanic acid precipitate as obtained above.

This impure titanic acid precipitate is burned in the crucible containing the residue from the silica volatilization. The contents of the crucible consisting of all the titanic acid a small quantity of the alumina, part of the phosphorus and some iron are now fused with 2 grams of sodium carbonate. The fusion is disintegrated in 50 c.c. of hot water in a No. 2 beaker and the solution is heated to boiling temperature for 10 minutes, allowed to settle, and filtered through a close filter from the insoluble sodium titanate and ferric oxide into a No. 3 beaker, the residue being washed with a 1 per cent sodium carbonate solution. It may be necessary to refilter the solution as the sodium titanate is liable to run through. To test the completeness of the separation of alumina and titanic acid by the above method the residue on the paper was ignited and again fused with sodium carbonate, digested and filtered as before. No alumina was found in the filtrate in several tests, although the alumina in the filtrate from the first fusion was as high as 2.50 per cent.

The clear filtrate from the sodium titanate is added to the solution of the first aluminium precipitate. To this solution containing all the alumina to c.c. of ammonium phosphate solution are added and then ammonium hydroxide is added until a flocculent precipitate is formed, the solution being acid to litmus. One and a half c.c. of strong hydrochloric acid are now added, followed by the addition of 30 c.c of sodium thiosulphate solution to the clear solution. The solution is heated to boiling, at which point 15 c.c. of ammonium acetate solution and 8 c.c. of acetic acid are added, and the boiling continued 10 minutes. The solution is filtered and the precipitate washed 10 times with hot water, burnt and weighed as aluminium phosphate.

Titanic Acid.—The titanic acid may be determined colorimetrically or gravimetrically. In the former case a sulphuric acid solution of the titanium is required, while in the latter a hydrochloric acid solution is preferable. The filter paper containing the sodium titanate is transferred to the No. 2 beaker in which the fusion was disintegrated, the crucible rinsed with hot acid and water into the beaker and to c.c. sulphuric acid

(equal parts acid and water) or strong hydrochloric acid added. On warming all the titanium enters the solution as sulphate or chloride, as the case may be. The solution is filtered from the paper pulp and the titanium determined colorimetrically or gravimetrically in the filtrate.

When the titanic acid is determined by the latter method, the solution of titanium chloride ($TiCl_4$), containing some ferric chloride, is diluted to 300 c.c. in a No. 5 beaker. Ammonium hydroxide is added until a flocculent precipitate is formed and the solution is yet acid to litmus. One and a half c.c. of strong hydrochloric acid are now added, followed by the addition of 30 c.c. thiosulphate solution to the clear solution. The solution is now boiled 10 minutes and the precipitate allowed to settle, filtered and washed 10 times with hot 2 per cent acetic acid solution, burnt and weighed as titanic acid (TiO_2), 0.6005 of which is titanium (Ti).

The titanic acid may be estimated colorimetrically by comparing the solution of titanium sulphate, $Ti(SO_4)_2$, obtained above, or an aliquot part of it, with a standard solution of the sulphate.

To this end the standard titanium sulphate solution may be prepared as follows: One-quarter of a gram of ignited titanic acid (Merck) is fused with 6 grams of sodium carbonate, the fusion disintegrated in 50 c.c. water and the crucible and lid placed therein also. On warming for an hour or more with 100 c.c. dilute sulphuric acid, solution is complete. This solution is transferred to 250 c.c. flask and diluted to the mark with water.

One c.c. of this should contain 1 mg titanic acid. However, the titanic acid obtainable is seldom 100 per cent pure. Its purity may be found and the solution standardized by precipitating 50 c.c. of the solution with thiosulphate in a slightly acid solution, as shown under the gravimetric method.

The color comparison may be made by adding the standard titanium sulphate solution from a burette to a tube containing 5 per cent sulphuric acid and 5 c.c. hydrogen peroxide, the volume being the same as that of the test, until the color matches that of the test solution contained in a tube of the same bore and to which 5 c.c. of hydrogen peroxide have likewise been added. The number of cubic centimeters of standard solution used indicates the titanic acid present. W. F. Hillebrand ("The Analysis of Carbonate and Silicate Rocks," Bulletin 422 of United States Geological Survey) states that some hydrogen peroxide contains fluorine, which bleaches the yellow color, and gives method for testing for it.

Phosphorus and Sulphur.—These elements are determined as described in "The Methods of the United States Steel Corporation for the Commercial Sampling and Analysis of Iron Ores," previously mentioned. However, in the determination of sulphur in an ore containing titanic acid, sodium nitrate should be used in the fusion of the ore in place of the more common potassium salt as the potassium titanate is soluble.

The alumina and titanic acid in the Iron Mountain Standard Ore of the United States Steel Corporation and in the Crescent Standard Ore of the Bureau of Standards were determined by the foregoing method with the following results:

	Alumina.		Titanic Acid.	
Iron Mountain	1.25	1.26%	0.10	0.10%
Crescent	1.00	1.01	0.07	0.07

A test was also made adding titanium chloride solution to the solution of the foregoing ores and determining alumina and titanic acid in the mixture. One-half gram of ignited chemically pure (?) titanic acid was fused with sodium carbonate, disintegrated in hot water and dissolved in 100 c.c. strong hydrochloric acid. The solution was transferred to a flask and made up to 500 c.c. and the titanic acid determined on 50 c.c. portions.

Alumina and titanic acid were also determined in the mixture of 50 c.c. of the above solution of titanium chloride with the solution of one gram of standard ore. Results were as follows:

	Al_2O_3 calculated from $AlPO_4$	TiO_2 as weighed	TiO_2 in Ore	TiO_2
	grams.	grams.	grams.	grams.
Precipitated by NH_4OH	....	.0435	....	....
Fe_2O_3 present	....	.0007	....	....
Difference	....	....	....	.0478
Precipitated by $Na_2S_2O_3$	....	....	....	.0478
Precipitated by $Na_2S_2O_3$	....	....	....	.0479
With Sol. of 1 gram Iron Mountain Ore	.0129	.0488	.0010	.0478
With Sol. of 1 gram Iron Mountain Ore	.0130	.0488	.0010	.0478
With Sol. of 1 gram Crescent Ore	.0101	.0486	.0007	.0479
With Sol. of 1 gram Crescent Ore	.0103	.0488	.0007	.0481

The Bureau of Standards report on their Crescent Standard Ore, Alumina 1.01 per cent, Titanic Acid .07 per cent.

Laboratory of Duquesne Steel Works, Duquesne, Pa.

The Equilibrium of the System Consisting of Calcium Carbide, Calcium Cyanamide Carbon and Nitrogen.

BY M. DE KAY THOMPSON AND ROBERT H. LOMBARD.*

Since the discovery by Frank and Carol[1] that pure nitrogen is absorbed by impure calcium carbide at high temperature according to the reversible reaction:

$$CaC_2 + N_2 \rightleftarrows CaCN_2 + C,$$

calcium carbide has been employed on a commercial scale for the fixation of atmospheric nitrogen. This method yields four times as much nitrogen for a given amount of power as the direct oxidation in an electric arc. Calcium cyanamide can be used directly as a fertilizer or it may be the starting point for the manufacture of other nitrogen compounds. In consequence of the importance of this reaction, we were led to undertake the determination of the equilibrium pressure of the nitrogen

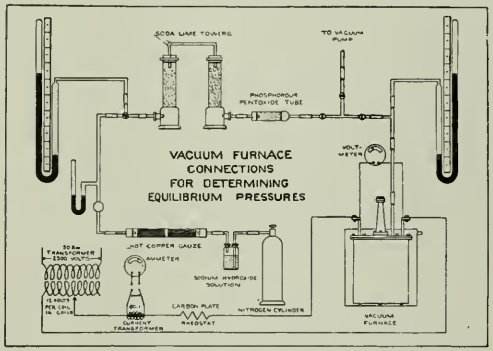

FIG. 1.—ARRANGEMENT OF APPARATUS FOR MEASURING EQUILIBRIUM PRESSURE.

in the above reaction for a number of different temperatures. In this system there are three components and four phases; consequently, according to the phase rule there will be one degree of freedom, or for every temperature a corresponding pressure at which equilibrium will exist.

The method of carrying out this investigation was to heat a graphite crucible containing graphite, calcium carbide and calcium cyanamide in an Arsem vacuum furnace[2] in an atmosphere of nitrogen. The heating current was taken from a transformer in steps of 12 volts, and fine adjustment of the current was obtained by a carbon-plate rheostat. The pressure was indicated by a mercury manometer, one arm of which was connected with the furnace, the other was evacuated and sealed off.

The arrangement of the apparatus is shown in Fig. 1. It

*The following research was carried out under a grant from the Rumford Fund of the American Academy of Arts and Sciences. Grateful acknowledgment is hereby made to the trustees of this fund.

[1] *Z. f.angew. Chem.* 19,853 (1906).

[2] *Trans. Am. Electroch. Soc.* 9,163 (1906).

was intended to measure the temperature of the crucible by means of a thermoelectric junction entering the furnace through the tower projecting above the water in which the furnace is immersed, and two or three such runs were made, in which the wires of the couple were insulated from each other by different kinds of tubes. One wire was covered with a tube 1 mm in diameter, and this tube and the other wire were placed in a larger tube sealed at the end which covered the junction.

These tubes were long enough to reach up into the tower when their lower end was resting in the crucible. The hot end of the tube surrounding the wires of the thermoelectric couple was covered with a short graphite tube to protect it from the material in the crucible.

No tubes that were available, however, would stand a temperature above 1200° C. Even Berlin porcelain was spoiled after a few hours' heating. Carbon seemed to have penetrated the whole mass of the porcelain where it was surrounded by the graphite spiral heater, and as much of the couple as was heated was also spoiled.

It was, therefore, necessary to give up the direct measurement by the thermocouple and use a Wanner optical pyrometer. The instrument used in this investigation had just been calibrated by the Bureau of Standards, with results that were not materially different from the calibration furnished by the makers. The amyl acetate lamp was not constant, however, and, therefore, adjustment of the current in the small electric lamp of the Wanner pyrometer was made as follows:

A furnace, shown in Fig. 2, was made by wrapping platinum foil ¼ in. wide and 0.001 in. thick about a Royal Meissen porcelain tube of 2.54 cm inside diameter. A cylindrical piece of graphite was placed in the furnace, and a small porcelain tube was passed through the larger one and through a hole in the graphite. The smaller tube was to hold the thermoelectric couple, the junction of which was in the same plane as one of the faces of the carbon. The temperature of this face of the carbon was, therefore, given directly by the thermoelectric couple.

In order to adjust the current in the Wanner pyrometer the reading of the Wanner was set to agree with that of the couple, a piece of glass cut from the same plate as that in the tower of the vacuum furnace was placed in the groove in the pyrometer provided for holding the smoked glass sometimes used for increasing the range of the instrument, and the pyrometer was sighted on the graphite cylinder. The current was then adjusted until the two fields in the pyrometer had the same appearance. The small furnace was kept heated during each run, and before measuring the temperature of the vacuum furnace a comparison was always made with the thermoelectric couple.

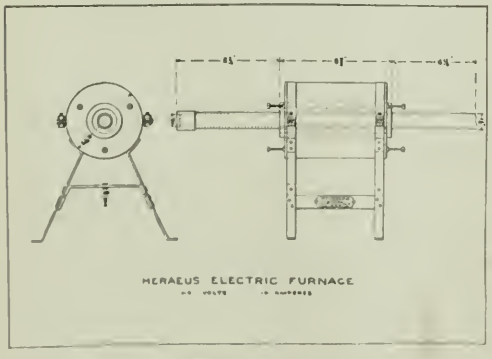

FIG. 2.—ELECTRIC FURNACE

It was found that by this means the current in the lamp as adjusted from day to day was much more constant than when the amyl acetate lamp was used as a standard, but it will be seen from the tables below that the current was not absolutely constant. This must have been due to a variation in the electric lamp itself. The following table shows the effect of a change of the current in the lamp on the apparent temperature

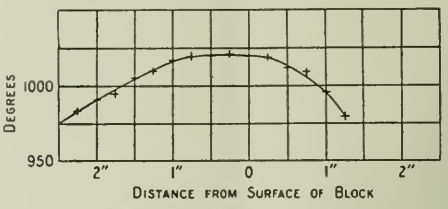

FIG. 3.—NON-UNIFORMITY OF TEMPERATURE.

as measured by the pyrometer. The temperature of the body measured was held constant in the vacuum furnace. The current is given in arbitrary units equal approximately to tenths of an ampere.

TABLE 1.

Current in Wanner pyrometer.	Temperature.
6.0	1304
6.1	1295
6.2	1282
6.3	1263
6.4	1263
6.5	1241
6.6	1226

The current could not be set better than 0.1 of a unit, corresponding on the average to 10°.

In order to see how the temperature varied in the auxiliary furnace the temperature was taken with the junction at different distances from the surface of the carbon block, which was placed near the center of the tube. The results are given in Fig. 3, which shows that no appreciable error would be introduced if the junction had not been exactly in the same plane as the surface of the graphite block.

The calcium carbide used in the following experiments was

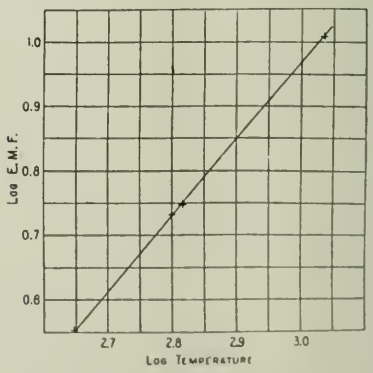

FIG. 4.—CALIBRATION OF THERMOCOUPLE.

all made at one time in an arc furnace from Merck's lime and turnings from Acheson graphite electrodes. It was analyzed by the loss in weight method and found 78¼ per cent pure, having been ground up and passed through a sieve with 36 meshes to the linear inch before the analyses were made.

The method of carrying out the experiments was as follows: The crucible was filled nearly to the top with the carbide, was

covered with a graphite lid with a hole in it, and then placed in the furnace. The furnace was pumped out and the charge heated to the temperature at which the experiment was to be made (or higher) until all gases were driven off. Nitrogen, made by the Linde process and 90 per cent pure, was let into the furnace after passing it over hot copper, soda lime and phosphorus pentoxide.

After allowing the nitrogen to enter the furnace until the pressure was 2 cm or 3 cm, it was pumped out to about 1 mm or 2 mm. This was repeated once or twice to remove all other gases. The furnace was filled with nitrogen to a pressure that was certainly greater than the equilibrium pressure. The furnace was generally hot when filled. The furnace was often filled in the afternoon and the run started on the following morning. On heating with an excess of nitrogen some of it would be absorbed, producing calcium cyanamide and carbon for the carbide, so that all three solid phases were then present. After equilibrium had been reached from this side, nitrogen was pumped out, and the equilibrium was approached from the other side. The mean of the two results was taken as the best value of the equilibrium obtainable.

Experimental Data.

A Siemens & Halske millivoltmeter of 471 ohms resistance was used for measuring the temperature with a platinum, platinum-rhodium junction. The millivoltmeter was calibrated and the following table contains the corrected value of the scale reading with the corresponding temperature:

TABLE 2.—CALIBRATION OF THERMOELECTRIC JUNCTION.

Temperature Cent. of melting point.	Millivolts.	Log of temperature.	Log of millivolts.
Sulphur 447.7	3.56	2,6481	0.5515
Antimony 630.7	5.38	2.7998	0.7307
Aluminium ... 656.	5.59	2.8169	0.7474
Copper .....1083.	10.19	3.0346	1.0081

The logarithms of the temperatures and pressures are seen from Fig. 4 to lie on a straight line.

The pressures given in the following tables are millimeters of mercury and the temperatures in centigrade degrees. The numbers of the experiments are those recorded in the laboratory notebook. These tables contain only an abbreviated record of the experiments. Under the heading "Kilowatts" is given the product of current and amperes divided by 1000, not taking into account the power-factor of the furnace.

EXPERIMENT 4.

The following experiment was carried out before the method of standardizing the Wanner pyrometer described above had been adopted. Temperature measurements of the charge were, therefore, made with different values of the current in the lamp of the Wanner pyrometer, and the correct value, with its corresponding temperature, was determined subsequently from Table 1.

	Time.	Kilowatts.	Pressure.
April 28.	1:23 p. m.	8.36	317.8
	1:50 " "	8.26	276.3

Some nitrogen was pumped out in order not to use up all the carbide.

	1:52 p. m.	8.52	264.0
	4:45 " "	8.52	267.4
	Time.	Kilowatts.	Pressure.
April 29.	12:35 p. m.	8.52	267.8
	2:22 " "	8.53	270.0

Here the pressure had become nearly constant, so nitrogen was admitted.

	2:25 p. m.	8.48	277.0
	4:45 " "	8.52	273.3

The equilibrium pressure is, therefore, between 270 mm and 273 mm, or 271.5 mm of mercury. The temperature, determined as explained above, was 1263° C.

EXPERIMENT 6.

A fresh charge of 33.7 grams of carbide was used in the following experiment:

	Time.	Kilowatts.	Pressure.
May 11.	2:55 p. m.	7.65	281.7
	4:05 " "	7.66	224.0
	4:37 " "	7.66	223.6

The pressure had become nearly constant, so nitrogen was admitted.

	4:45 p. m.	7.66	226.3
	5:10 " "	7.66	225.1
May 12.	12:40 " "	7.70	218.5
	1:00 " "	7.00	222.7
	2:22 " "	7.58	223.0

Nitrogen was again pumped out.

	2:23 p. m.	7.58	214.1
	4:30 " "	7.52	224.7
May 13.	1:45 " "	7.56	224.6
	2:45 " "	7.61	223.5

The pressure had become nearly constant at 224 mm, so nitrogen was admitted.

	Time.	Kilowatts.	Pressure.
	2:55 p. m.	7.61	236.0
	5:05 " "	7.40	224.7

The mean temperature was 1223 with the current in the lamp of the Wanner pyrometer 6.4 units. The equilibrium pressure was determined twice from each direction and was found to be 224 mm.

EXPERIMENT 7.

The following experiment was made with the same charge as in 6:

	Time.	Kilowatts.	Pressure.
May 17.	2:35 p. m.	8.98	275.3
	3:45 " "	8.98	282.9

The pressure was increasing very slowly, possibly all calcium cyanamide having been decomposed. Then free nitrogen was admitted.

	3:50 p. m.	8.98	316.5
	4:55 " "	8.98	304.2
May 18.	1:30 " "	8.92	305.6
	3:00 " "	8.78	305.1

The pressure had become nearly constant. Nitrogen was, therefore admitted.

	3:02 p. m.	8.78	329.2
	4:45 " "	8.71	311.8
	5:00 " "	8.71	313.0
May 19.	12.30 " "	8.78	312.5
	1:13 " "	8.71	312.9

The pressure was nearly constant.

Some nitrogen was then pumped out.

	Time.	Kilowatts.	Pressure.
	1:18 p. m.	8.71	293.8
	3:35 " "	8.71	307.0

The pressure had become nearly constant. Some nitrogen was, therefore, admitted.

	3:38 p. m.	8.71	317.4
	4:20 " "	8.71	313.0

Absorption was taking place very slowly. In order to see whether 313 or 305 was nearer the true value, nitrogen was pumped out.

	4:34 p. m.	8.71	309.0
	4:55 " "	8.71	310.3

The results of this experiment are somewhat inconsistent, as the pressure was decreasing at 305 mm and increasing at 310 mm. The mean of these is 308 mm, and the mean temperature was 1297° C. The current in the lamp of the Wanner pyrometer was from 6.3 to 6.5 units.

EXPERIMENT 8.

	Time.	Kilowatts.	Pressure.	Current in Wanner pyrometer.	Temperature.
June 14	1:55 p. m.	9.86	444.0	5.80	1371
	2:15 " "	9.86	384.8	...	....
	2:45 " "	9.69	373.8	5.83	1379
	3:15 " "	9.73	375.3	5.88	1373
	3:45 " "	9.73	375.4	5.82	1371
	4:45 " "	9.65	376.1	5.80	1377
June 15.	8:45 a. m.	9.67	....	...	....
	9:15 " "	9.71	374.2	...	....
	10:30 " "	9.68	376.5	5.80	1380
	Nitrogen was pumped out.				
	11:13 a. m.	9.64	366.6	...	....
	11:30 " "	9.65	370.2	5.88	1374
	12:30 " "	9.62	373.4	...	....
	1:30 p. m.	9.65	377.3	5.84	1377
	4:30 " "	9.55	379.3	..	....

It is seen that the pressure decreased until it became constant at about 376 millimeters, and after pumping out to 367 it increased to 379. The average mean is 378 millimeters and the average temperature 1378. The interior of the furnace after these runs was found covered with a thick white deposit.

The charge in the crucible in each case gave tests for carbide and calcium cyanamide. The latter was tested for by adding ammonia and silver nitrate to the aqueous solution. A yellow precipitate of silver cyanamide showed the presence of calcium cyanamide. The white powder taken from the walls of the furnace also gave a yellow precipitate of silver cyanamide on adding ammonia and silver nitrate to a solution of this substance.

EXPERIMENT 9.

In the following run an attempt was made to measure an equilibrium at about 1440. It was not successful, however, as will be seen from the following results:

Time.	Kilowatts.	Pressure.	Current in Wanner pyrometer.	Temperature.
10:15 a. m.	10.6	....	...	....
10:20 " "	10.88	650.0	5.75	1433
10:30 " "	10.94	587.0	...	....
11:30 " "	10.94	513.3	...	....
12:55 p. m.	10.94	514.0	5.80	1420
Some nitrogen pumped out.				
1:00 p. m.	10.94	458.0	..	....
1:15 " "	10.94	468.0		....
2:00 " "	10.94	460.0		
Some nitrogen pumped out.				
2:05 p. m.	10.94	460.0		....
2:45 " "	10.94	461.0		....
More nitrogen pumped out.				
2:50 p. m.	10.94	427.0		
3:45 " "	10.94	432.7	5.92	1448

It is evident that though the reaction goes in both directions at this temperature, it is so slow that it is difficult to tell whether an equilibrium has been reached or not. The reason for this may have been that the charge had become very hard and could not be dug out of the crucible and appeared as though it had been somewhat fused together. This would make it difficult for the nitrogen to penetrate its entire mass. The average temperature of this run was 1437° C. A later attempt at 1448° was more successful and is given in Experiment 15.

(To be concluded.)

Zinc Ore Dressing in Colorado--II.

By H. C. Parmelee.

The Wellington Mines Company.

The property of the Wellington Mines Company is in Summit County, Colorado, about two and one-half miles up French Gulch from the town of Breckenridge. It comprises about 150 acres of mineral land and a concentrating mill of 80 tons daily capacity. The history of the mine is similar to that of many others in the region. Located over 20 years ago it yielded some carbonate ore from surface deposits, but deeper working developed the complex zinc-lead-iron sulphides which are now characteristic of the district, and which have remained undeveloped while more easily treated ores have been sought. It is mainly in the successful milling of the mixed sulphides that the Wellington now lays claim to attention.

The nature of the crude ore is shown in the following tabulation of analyses of several samples. The lime and silver given in some of the samples may be taken as typical of those from which they are omitted.

WELLINGTON CRUDE ORE.

Zn %	17.4	17.1	15.5	15.3
Pb %	9.2	9.0	7.6	7.7
Fe %	20.6	20.5	19.8	28.0
SiO_2 %	18.4	22.2	20.1	....
CaO %	...	...	...	1.2
Ag. oz.	3.56	3.40	....	....

At present the Wellington mill uses a system of wet dressing only, but is preparing to install roasters and magnetic separators to treat a low grade zinc-iron middling which is now accumulating.

Jigs are used, commencing concentration at 4 + 6 mesh and finishing with slime. A peculiar feature in the construction of the Wellington mill is that the rolls are set in the upper part of the building. At first sight this plan would be condemned, but an inspection of the flow sheet given herewith will probably bring out an advantage, viz., that the elevators handle only coarse material from which the fine has been screened, and therefore operate easily, giving no trouble on account of the load sticking in the buckets and requiring a water spray for its complete release. The work described herewith represents merely the present state of progress toward a finished system which the present management has planned.

Crushing and Screening.

The ore is delivered by electric tramway to crude ore bins of 80 tons capacity. Passing over a grizzly (1-in. slots), the oversize is fed to a 10-in. x 15-in. Blake crusher, reducing the ore to 1¼-in. size. Crushed ore and undersize of the grizzly are combined and elevated by (No. 1) 12-in B. and B. elevator to crushed ore bins of 80 tons capacity. A Challenge feeder discharges ore from these bins to (No. 2) 10-in B and B. elevator (where water is added), which delivers to a trommel fitted with ⅜-in. punched screen. The oversize passes to No. 1 14-in. x 27-in. horizontal rolls and thence back to (No. 2) elevator. The undersize is delivered to the screen line of three trommels fitted with 4-mesh, 6 mesh and 12-mesh wire screen of No. 12 gage, 14 gage and 18 gage respectively.

The oversize of the 4-mesh trommel is delivered by (No. 3) 10-in. B. and B. elevator to No. 2 14-in x 27-in horizontal rolls and thence back to the screen line. The undersize of the 4-mesh trommel passes to the 6-mesh trommel, the oversize of the latter (being through 4 mesh and on 6-mesh) is sent to jigs Nos. 1 and 2. The undersize of the 6-mesh trommel passes to the 12-mesh, the oversize of which (being through 6 mesh and on 12-mesh), passes to jigs Nos. 3 and 4. The undersize of the 12-mesh trommel is laundered to a thickening cone, preliminary to classification for treatment on Nos. 5 and 6 jigs, and on the tables.

Jigging.

The jig equipment consists of three double 6-compartment Harz jigs, or six in all.

The details of jigging given in the present article apply to general practice which, it will be understood, may be changed occasionally as the mill feed varies in mineral composition. All compartments of all jigs make hutch products, and from some compartments an overflow discharge is also taken off through the side gates. The jig products are lead concentrate, shipped to the lead smelter; low-grade zinc-lead-iron middling, stored for future magnetic separation; middling for regrinding and further concentration; high-grade zinc-iron middling, shipped to pigment works; and zinc concentrate, shipped to zinc smelter.

The source of these various jig products can be understood better from the following diagram of the jigs, in which hutch and gate-and-dam products are indicated.

The coarse jigs are represented at the top of the diagram, and the finer below.

The flow of ore is from left to right. The terms used in the diagram, and their meanings are, lead, galena concentrate; iron, zinc-lead-iron middling for magnetic separation; regrind, middling which is reground and further concentrated; middling, high-grade zinc-iron middling used for pigment; zinc, zinc concentrate.

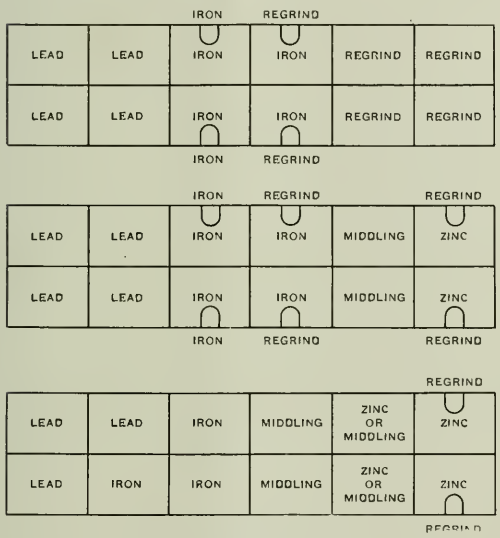

FIG. 1.—DIAGRAM OF JIGS.

DIAGRAM OF JIG PRODUCTS.

The following tabulation gives some details of the jigging practice at the Wellington mill:

MESH OF JIG SCREEN.

Jig No.	—Compartment Number— 1	2	3	4	5	6
1.....	4	5	6	6	7	8
2.....	4	5	6	6	7	8
3.....	6	7	7	7	7	7
4.....	6	7	7	7	7	7
5.....	10	12	10	10	10	10
6.....	10	12	10	10	12	12

The gage of wire used in the various screens is as follows: 4-mesh, No. 14; 5-mesh, No. 15; 6-mesh, No. 16; 7-mesh, No. 17; 8-mesh, No. 18; 10-mesh, No. 19; 12-mesh, No. 20.

SPEED (R.P.M.) AND LENGTH OF STROKE (INCHES) OF JIG PLUNGERS

Jig No.	r.p.m.	—Compartment Number— 1	2	3	4	5	6
1	175	7/8	5/8	1	3/4	3/4	1
2	175	7/8	5/8	1	3/4	3/4	1
3	235	5/8	1/2	3/4	1/2	1/2	3/4
4	235	5/8	1/2	5/8	1/2	1/2	3/4
5	272	5/8	1/2	5/8	1/2	5/8	3/4
6	294	1/2	3/8	1/2	3/8	3/8	1/2

In bedding the jigs, clean galena crushed to ½-in. mesh is used for the first two compartments of the two coarse jigs. The finer jigs are bedded, as required, with the lead or iron concentrates of the coarse jigs.

Classification.

As stated above, the undersize of the last (12-mesh) trommel is classified into suitable products for the fine (Nos. 5 and 6) jigs and the tables. The hydraulic classifier is one designed by general manager Robert M. Henderson, and is somewhat similar in outline and action to the Richards pulsator classifier although the hydraulic water is without pulsations. As illustrated herewith the box has the general outline of an inverted right angled triangle, and is divided vertically into compartments decreasing in size from the first to the fifth. The feed flows down the inclined bottom of the box, and as it passes beneath the various compartments is hydraulically classified by water introduced at the proper points. The first compartment receives no hydraulic stream and gives an overflow of the finest material. The second, third and fourth compartments yield increasingly coarser products. The fifth compartment gives two products, one from a spigot placed in the descending line of spigots from the other compartments, and the other from a vortex classifier placed at the bottom of the compartment. The 5-compartment classifier thus yields six products. The appearance of the concentrating tables attests the excellence of the classifier.

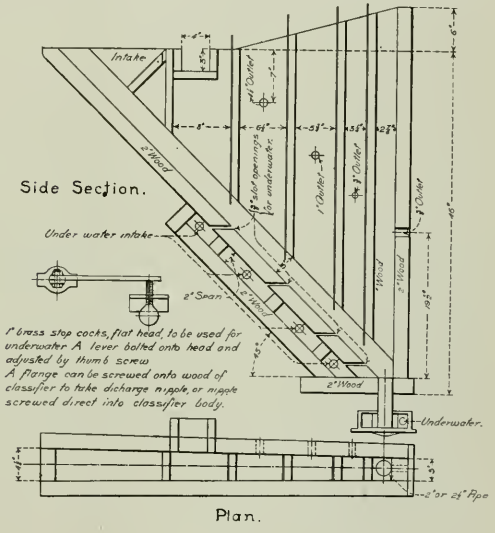

FIG. 2.—HENDERSON'S CLASSIFIER.

The disposition of the classified products is as follows: Fifth compartment (spigot and vortex), to Nos. 5 and 6 jigs; the other four compartments to concentrating tables.

Table Concentration.

The table equipment consists of three No. 4 and one No. 5 Wilfley tables, and one Card. The Wilfley tables treat pulp

from the first four compartments of the classifier, and the Card receives thickened slime from the cone system. The details of table operation are tabulated below.

No.	Table	Feed	Strokes per min.	Length of Stroke
1	Wilfley	Coarse	240	15 16 in.
2	"	Medium	240	7/8 in.
3	"	"	280	1 in.
4	"	Fine	280	3/4 in.
5	Card	Slime	290	3/8 in.

Three shipping products are recovered from each table, viz., lead concentrate, high-grade zinc-iron middling and zinc concentrate. The iron is divided between the lead and zinc to make suitable shipping products according to smelter contracts. Tailings from tables, jigs, and the overflow from the cone system all go to waste. They are automatically sampled by a Scobey tailing sampler taking a sample every 2 minutes. The accumulation is assayed every two days.

Products.

The following assays are typical of the products now being made.

GENERAL ZINC CONCENTRATE

Au	oz.	0.02	0.015	0.02
Ag	oz.	3.00	2.90	2.20
Pb	%	1.8	Trace	1.7
Fe	%	16.4	16.1	15.3
Zn	%	35.2	39.6	37.5
SiO_2	%	7.0	6.0	6.4
Cu	%	0.26	..	0.3
S	%	28.6	25.9	31.0
CaO	%	0.6	1.2	...

The above assays represent the mixed jig and table concentrates just as shipped. The average of the jig zinc is about the same as that of table zinc, although different machines in both classes vary in the quality of their products. The order of the jigs in the quality of zinc concentrate made is 3, 4, 5, 6, 1, 2 (No. 1, coarse and No. 6, fine). The order of the tables in the quality of their zinc concentrate is 3, 2, 4, 5, 1 (No. 1, coarse and No. 5, slime), varying from 43 to 35% zinc respectively.

The mixed jig and table galena concentrate as shipped, assays as follows:

GENERAL LEAD CONCENTRATE

Au	oz.	0.03	0.03
Ag	oz.	11.20	10.60
Pb	%	46.4	44.8
Fe	%	15.9	17.0
Zn	%	...	5.4
SiO_2	%	2.0	2.5

The zinc-iron middlings are of two grades; one, a high-grade middling shipped to pigment works, and the other, a low-grade middling now being saved for roasting and magnetic separation.

MIDDLINGS

		High-grade	Low-grade
Zn	%	28.3	13.2
Fe	%	24.3	32.5
Pb	%	3.1	5.2

The tailings, combined as described before, assay as follows

GENERAL TAILING

Zn	%	0.4	8.5	10.7	0.0	0.4	8.7
Pb	%	2.0	1.0	2.2	1.2	1.3	1.9
Fe	%	..	..	..	..	..	9.8
CaO	%	3.0	2.6	..	..	..	..
SiO_2	%	..	..	..	..	..	47.3

Power, Water and Labor.

The mill is electrically operated. Current is supplied by the Central Colorado Power Company at 13,000 volts, which is stepped down to 440 volts at the mill transformer house. Two motors suffice for the operation of the mill machinery. One 75-hp motor drives the crusher, rolls, elevators, screens, jigs and water pump; and one 10-hp motor operates the tables, tailing sampler and slime pump.

Water is secured from springs and conducted to the mill by flume, and raised by a centrifugal pump to a tank placed in the upper part of the building. Arrangements are also made to reuse a part of the mill water when the natural supply is short. The jig tailings can be dewatered and the clear water pumped back for further use.

Four men, in addition to the foremen or shift boss, are required for each of two shifts. As crushing is carried on for but two shifts, one man is dispensed with on the third shift.

Proposed Improvements.

It is intended immediately to install additional jigs and tables. Instead of directly regrinding the oversize from the 4-mesh trommel, it will be jigged first, as it contains a great deal of clean mineral which does not require regrinding. Also, instead of returning the reground jig middling to the screen line at the 6-mesh trommel, it will be sized and jigged separately. An additional Card table will be installed for slime treatment, as the present slime feed is too heavy for one table, and another Wilfley will be added to treat the reground middling mentioned.

When a sufficiently large quantity of zinc-iron middling has accumulated to warrant the erection of a magnetic separation plant, it will be installed. In all probability the management will favor the multiple hearth type of roaster, but the type of magnetic separator has not been decided upon. Additional table equipment also will be provided to treat the zinc-lead tailing from the magnetic separators, as there is sufficient lead present to warrant this step. The magnetic iron removed will be without value.

Following is a condensed statement of the Wellington mill practice:

Crude ore by electric tram to
1 Bins of 80 tons capacity, over grizzly (1-in. slots) to
2 10 x 15 Blake crusher reducing to 1¼ in.
Undersize of grizzly and product of crusher to
3 12-in. B. and B. elevator to
4 Bins of 80 tons capacity, by Challenge feeder to
5 10-in. B. and B. elevator to
6 3/8-in. punched screen trommel; oversize to 7; undersize to 8.
7 14 x 27 horizontal rolls, back to 5.
8 4-mesh wire screen trommel; oversize to 9; undersize to 11.
9 10-in. B. and B. elevator to
10 14 x 27 horizontal rolls, back to 8.
11 6-mesh wire screen trommel; oversize to 13; undersize to
12 12-mesh wire screen trommel oversize to 14; undersize to 15.
13 Nos. 1 and 2 jigs.
14 Nos. 3 and 4 jigs.
15 Thickening cone; underflow to 16; overflow to 21.
16 5-compartment classifier; 5th, compartment to 17; first four compartments to 20.
17 Nos. 5 and 6 jigs.
Jig middling or regrinding to
18 10-in. B. and B. elevator to
19 12 x 20 horizontal rolls, back to 11.
20 Four Wilfley tables.
21 Thickening cone; overflow to 22; underflow to 23.
22 Thickening cone; overflow to waste; underflow to 23
23 Card table.

Silver Production in Mexico.—The July production of the Rio Plata Mining Company, Chihuahua, Mex., amounted to 82,000 oz. of silver. This is the largest monthly production on record, the best previous showing having been made in January, when 71,164 oz were produced. Last month's shipment of bullion to Monterey brought $40,340

Modified Cyanide Process.

Reference has already been made in our abstracts of recent chemical and metallurgical patents to improvements in the cyanide process, due to Mr. JOHN COLLINS CLANCY, chief metallurgist of the Moore Filter Company, of New York City.

In view of the great interest which this invention is attracting, a summary of the process will be welcome.

The process will be found particularly interesting from an electrometallurgical standpoint. Calcium cyanamide, which plays a very important part in Mr. Clancy's process, is an electric-furnace product obtained by the action of nitrogen gas (obtained from air, for instance, by liquefaction) upon calcium carbide, the latter being also an electric-furnace product. There are various calcium cyanamide plants in operation in Europe and one, that of the American Cyanamide Company, at Niagara Falls, Ontario, Canada. While heretofore calcium cyanamide has been used only as a fertilizer, Mr. Clancy's invention opens a new and important field for it in the metallurgy of gold and silver.

But the relation of the Clancy process to electrometallurgy is still more intimate, since, as will be seen, electrolysis plays an important part in the process.

The following quotations are taken from a foreign patent specification of Mr. Clancy:

Chief Principles.

"I have discovered that a cyanogen-bearing compound which is not a solvent of the precious metals can be made a solvent of the same by the addition of an amidine or amide compound in the presence of an electric current. As a result of this discovery I am able to use cyanogen compounds, heretofore not available in the cyanide process, and also I am able to regenerate spent cyanide solutions, and I am also able to use in the cyanide process oxidizing agents without loss of available cyanide.

"I have also discovered that when the pulverized ore is converted into a pulp by mixing the said ore with a substantially non-acid solution of a thiocyanate containing a halogen salt, and electrolyzing this mixture, then the precious metals only will be dissolved to the substantial exclusion of the baser metals, and that the above solution may be used with advantage in conjunction with the ordinary cyanide solution, and that this process may be successfully applied to either rebellious or non-rebellious ore without preliminary roasting, even though such ores contain reducing agents or tellurium or both.

Use of Cyanamide with Electrolysis.

"In the practice of my invention I may use the crude cyanamide of commerce, its isomerides and polymerides, as a solvent for the precious metals and other metals contained in ores; or I may use a soluble cyanamide in conjunction with the cyanide process in the treatment of ores, and as a means of regenerating cyanide solution or non-available cyanogen-bearing solution.

"When a soluble amidine or amide compound (such, for example, as dicyandiamidine, urea, guanidine, guanidyl-gluanidine, guanidine carbonate, formamide, oxamide, cyanamide, dicyanamide, or the various polymeric and isomeric modifications of the last-mentioned compounds) is added to a cyanogen-bearing solution and the mixture electrolyzed in contact with ore, the consumption of cyanide is very strikingly reduced; and also solutions containing cyanates, before, after or during the treatment of ore (together with ferrocyanides, ferri-cyanides, thiocyanates, or other such non-available cyanogen compounds formed in the treatment of ores) may be converted into compounds which readily dissolve the precious metals contained in ores; and most remarkable is the regeneration of cyanide from cyanates, such as alkaline cyanates, isocyanates and the various isomeric and polymeric compounds, or the oxygen derivatives of cyanogen, their isomerides and poiymerides.

"I have described the addition of a soluble amidine or amide compound, which may also include any of the above mentioned compounds; but I use preferably, when working my process, a soluble cyanamide (such, for example, as calcium cyanamide, as it is at present the cheapest source of an amidine or amide compound). A solution of calcium cyanamide, or its isomerides and polymerides, when subjected to electrolysis, is, I have discovered, capable of dissolving the precious metals, and other metals, without any other admixture of cyanogen-bearing material. From cyanamide all the amide or amidogen compounds I have mentioned are very easily prepared. For instance, by the action of sulphuric acid or hydrochloric acid upon cyanamide, water is absorbed and it becomes urea. B_2S converts it into thio-urea. NH_3 converts it into guanidine, while substituted guanidines are produced upon introducing the hydrochlorides or primary amines; and numerous other amide compounds may be prepared by simply inexpensive chemical treatment of cyanamide. It will, therefore, be obvious that my claim for a soluble amidine or amide compound includes all the polymeric and isomeric modifications and transpositions of soluble cyanamides.

"To show the action of these amide compounds I may take, for example, urea (carbamide) and add this to a solution of a cyanate (such, for example, as potassium cyanate) and electrolyze this mixture between inert electrodes. After a few minutes the said mixture will be transformed into one which will dissolve the precious metal in alkaline solution. It is well known that cyanates have absolutely no dissolving effect upon the precious metals, and that the formation of cyanate in the cyanide solution represents the chief source of loss of cyanide in the cyanide process, since when cyanates are allowed to stand for any considerable time, or are retained in solution, they are converted into ammonia and potassium carbonates and are transposed by the lime used in the cyanide process into insoluble carbonates, or by the action of sulphuric acid in the ore into sulphates, so that the original cyanide finds itself eventually in the residue dumps in the form of calcium sulphates or insoluble carbonates. Any oxidation means, when used in the cyanide solution to hasten the dissolution of the precious metals, increases the cyanide consumption, due to the formation of cyanate, which is often very great; and up to the present no means for regeneration or recovery of cyanide from cyanate in aqueous solutions has ever been discovered, so far as I am aware; but by my discovery that a soluble amidine or amide compound (such, for example, as calcium cyanamide) added to the cyanide-bearing solution, in conjunction with the electrolysis of said solution, opens up a cheap way whereby oxidation means other than atmospheric oxygen (such as treatment of cyanide-bearing solutions with oxidizing agents and soluble halogen compounds, or other oxidation means) may be successfully employed for readily dissolving and extracting precious metals from refractory ores, such as tellurides, arsenides, sulphides and other ores containing reducing agents. In general, this process may be successfully applied to either rebellious or non-rebellious ores without preliminary roasting, even though such ores contain the refractory compounds mentioned.

Apparatus.

"The apparatus for carrying my process into effect is simply the ordinary agitating tank furnished with electrodes spaced about from 1½ in. to 2 in. apart. These electrodes may be made of carbon, or one of iron and another or carbon; that is to say, the positive electrode undergoes no deterioration, as only nascent hydrogen is produced at the said electrode; therefore, iron answers the purpose. The negative electrode or anode is made preferably of a hard graphite or other such inert material.

"The circulation caused by the agitation keeps the whole mixture constantly circulating between the said electrodes. This circulation is maintained for a period of from 8 to 12 hours to obtain the desired result, which is described in the

examples. The said agitating tank with the electrodes is equivalent to the electrolyzer described in the examples.

Use of Halogen Compounds.

"Different ores require variations in the composition of the solution and in the current. For example, in treating ores containing tellurides or arsenides we may add a soluble halogen compound to the cyanamide treatment solution, or to the cyanogen-bearing solution, or to the non-available cyanogen-bearing solution, in the manner shown in the examples following.

"The function of a halogen compound—such, for example, as potassium iodide, when added to the cyanogen-bearing solution—is, that during electrolysis, the iodine liberated at the anode acts as an oxidizing agent. It is clear, therefore, that the presence of a halogen compound, such, for example, as potassium iodide, in said solutions, insures complete absorption of any nascent oxygen which might escape without doing oxidation work. The halogens also form addition compounds with cyanamide, these addition compounds in themselves acting as oxidizing agents.

"It is, therefore, obvious that in any of the examples given the function of the halogen is analogous.

Examples.

"In the following examples I use a solution of a soluble cyanamide alone or in conjunction with the cyanide solution, or in conjunction with a non-available cyanogen-bearing solution By non-available cyaogen-bearing solution I mean one that contains cyanogen in a different combination from that in which it exists in what is known as simple alkaline cyanides. For example, a solution after being used in the treatment of ore on testing shows no available cyanide. The cyanide in such solution is said to have been consumed, and is unfit for further treatment of ore until restrengthened by the addition of more cyanide. It is clear, therefore, that my process aims at, besides its other features mentioned, the substitution of a cheap amidine or amide compound (such, for example, as calcium cyanamide) in the place of cyanide or together with only a small proportion of cyanide.

"*Example 1.*—Two thousand pounds of pulverized ore is suspended in a solution of 4 lb. calcium cyanamide, 2 lb. caustic soda or lime (with or without the addition of 1 lb. potassium iodide) in 2000 lb. of water. The whole mixture, being in the proportion of two parts of said solution to one part of ore, is contained in an agitation tank. This mixture of ore and solution is continuously circulated for a period of 12 hours through the electrolyzer having a current density of about 50 amp per square foot of electrode surface. About 90 amp to 120 amp are used per ton of ore

"*Example 2.*—Two thousand pounds of pulverized ore is suspended in a solution of 1 lb. of potassium cyanide, 2 lb. calcium cyanamide, 2 lb. caustic soda or lime (with or without the addition of 1 lb. potassium iodide) in 2000 lb. of water, the whole mixture being in the proportion of two parts of said solution to one part of ore, is contained in an agitation tank. This mixture of ore and solution is continuously circulated for a period of 12 hours through the electrolyzer having a current density of about 50 amp per square foot of electrode surface. About 90 amp to 120 amp are used per ton of ore

"*Example 3.*—Two thousand pounds of pulverized ore is suspended in a solution of 2 lb. potassium ferrocyanide, 2 lb. calcium cyanamide, 2 lb. caustic soda or lime (with or without the addition of 1 lb potassium iodide) in 2000 lb. of water The whole mixture, being in the proportion of two parts of said solution to one part of ore, is contained in an agitation tank. This mixture of ore and solution is continuously circulated for a period of 12 hours through an electrolyzer having a current density of about 50 amp per square foot of electrode surface About 90 amp to 120 amp are used per ton of ore

"*Example 4.*—Two thousand pounds of pulverized ore is suspended in a solution of 2 lb potassium thiocyanate, 2 lb. calcium cyanamide, 5 lb pulverized calcium carbonate (with or without the addition of 1 lb. potassium iodide) in 2000 lb. of water. The whole mixture, being in the proportion of two parts of said solution to one part of ore, is contained in an agitation tank. This mixture of ore and solution is continuously circulated for a period of 12 hours through the electrolyzer having a current density of about 50 amp per square foot of electrode surface. About 90 amp to 120 amp are used per ton of ore.

"*Example 5.*—Two thousand pounds of pulverized ore is suspended in a solution of 1 lb. potassium cyanide, 2 lb. calcium cyanamide, 10 lb. pulverized calcium carbonate (with or without the addition of 1 lb. potassium iodide) in 2000 lb. of water. The whole mixture, being in the proportion of two parts of said solution to one part of ore, is contained in an agitation tank. This mixture of ore and solution is continuously circulated for a period of 12 hours through the electrolyzer having a current density of about 50 amp per square foot of electrode surface. About 90 amp to 120 amp are used per ton of ore.

"*Example 6.*—Two thousand pounds of pulverized ore is suspended in a solution of 1 lb. potassium cyanide, 1 lb. potassium thiocyanate, 1 lb. calcium cyanamide, 10 lb. calcium carbonate (with or without the addition of 1 lb. potassium bromide) in 2000 lb. of water. The whole mixture, being in the proportion of two parts of said solution to one part of ore, is contained in an agitation tank. This mixture of ore and solution is continuously circulated for a period of 12 hours through the electrolyzer having a current density of about 50 amp per square foot of electrode surface. About from 90 amp to 120 amp are used per ton of ore.

Regeneration of Cyanide.

"In this specification I have described my meaning of a non-available cyanogen-bearing solution, and I now give the following example as being a typical one for the regeneration of cyanide from spent cyanide solutions—that is, those solutions in which the cyanogen is non-available, such, for example, as cyanates, ferrocyanides, ferricyanides, sulphocyanides, etc.

"Example: Take 2000 lb. of solution which has an original strength of 2 lb. cyanide per ton and which has already been used for the treatment of ore, on testing shows only ¼ lb. of cyanide per ton of solution. Add 4 lb. of calcium cyanamide and electrolyze this mixture between inert electrodes for a period of two hours at a current density of 50 amp per square foot of electrode surface. On again testing the said solution it will be found that the available cyanide is increased, or rather gives a titration equal to 2 lb. of available cyanide. This solution is then ready for treating a fresh batch of ore. From the above it will be seen that ores which are readily leachable may be treated by percolation with the above-described electrolyzed solution.

"It will be seen that by my invention we generate solvents and continuously dissolve the precious metals which transform them into compounds soluble in the above solution. The function of the electric current in conjunction with a soluble amidine or amide compound (such, for example, as calcium cyanamide in the cyanogen-bearing solution) is one that is both reducing and oxidizing. It is clear that I make use of the anodic and cathodic reactions which occur at both electrodes in my solutions.

"From the above solutions or pulp the dissolved metals may easily be recovered by filtration and precipitation by methods well known and in use.

Effect of Additions. Treatment of Sulphides.

"It may be mentioned that the addition of a salt, such, for example, as sodium sulphate or ammonium sulphate, may be added to the solutions to render the same more conductive In practice I find the addition of from 10 lb. to 20 lb. of either of the above salts to a ton of solution gives satisfactory results

"In treating ores containing amongst other compounds sil-

ver sulphides or other sulphides it may not be necessary to add a halogen compound, such, for example, as potassium iodide, to the thiocyanate or cyanogen-bearing solutions, as the nascent oxygen produced by the electric current regenerates the alkaline cyanide from the alkaline thiocyanate.

"It is, therefore, obvious that after treating the silver ore or other sulphide-bearing ores with the ordinary cyanide solutions, the cyanide undergoes destruction and is said to be consumed (technically called cyanide consumption). Means at present employed to prevent this cyanide consumption and increase the extraction are the use of lead salts, such, for example, as lead acetate, litharge, etc., added to the cyanide solution, but these chemicals are expensive and irrecoverable and often inefficient when dealing with ores high in sulphides. The consumption of cyanide is largely due to the sulphide constituents of the compounds contained in the ores acting upon the cyanide solution in the presence of atmospheric oxygen; this gives rise to the formation of sulpho-cyanides—the formation of sulpho-cyanide represents a direct loss in cyanide because the cyanide is then in a non-available form and cannot be made to dissolve the precious metals by the ordinary means employed; but by my process a non-available cyanide solution is converted into an available cyanide solution, ready and active, together with any unconsumed cyanide, to dissolve the precious metals, which may be seen from reading the following descriptive example:

"Two thousand pounds of the pulverized sulphide ore are suspended in a solution of 6 lb. potassium cyanide (carrying 3 lb. to 5 lb. of protective alkalinity, the alkali used being preferably lime) in 2000 lb. of water; the whole mixture, being in the proportion of two parts of the said solution to one part of ore, is contained in an agitation tank. This mixture of ore and solution is continuously subjected to agitation and aeration by a current of air for a period of about 16 hours to 72 hours (the time depending upon the cyanide consumption). About when the cyanide consumption has ceased, the whole mixture of ore and solution is then subjected to electrolysis, in the said agitation tank, by introducing electrodes (preferably inert) into the said mixture carrying a current density of about 50 amp per square foot of electrode surface. About from 50 amp to 100 amp are used per ton of ore. The electrolysis is continued for about from 10 hours to 12 hours, the length of time depending upon the regeneration of the unavailable cyanide into alkaline cyanide. The latter may be determined hourly by titration or otherwise.

"From the above descriptive example it will be obvious that to start electrolysis of the ore pulp, together with the ordinary cyanide solutions, would result in immediate destruction of the cyanide, as there would be insufficient or no sulpho-cyanide present in solution to protect the cyanide against the nascent oxygen produced at the anode, which would at once oxidize the cyanide to cyanate.

Examples.

"I give the following examples of my process with and without the use of halogen salts without, however, limiting myself to the details of each.

"In the examples given the term electrolyzer has the same meaning as an agitation tank provided with electrodes, the circulation generally being caused by air agitation.

"*Example I.*—Two thousand pounds of pulverized ore are suspended in a solution of 2 lb. potassium thiocyanate, 2 lb. caustic soda or lime, 1 lb. potassium iodide in 2000 lb. of water: the whole mixture, being in the proportion of two parts of said solution to one part of ore, is contained in an agitation tank. This mixture of ore and solution is continuously circulated for a perid of 12 hours through the electrolyzer having a current density of about 50 amp per square foot of electrode surface. About from 90 amp to 120 amp are used per ton of ore.

"*Example II.*—Two thousand pounds of pulverized ore are suspended in a solution of 2 lb. potassium thiocyanate, 1 lb. potassium cyanide, 2 lb. caustic soda or lime, 1 lb. potassium iodide in 2000 lb. of water; the whole mixture, being in the proportion of two parts of said solution to one part of ore, is contained in an agitation tank. This mixture of ore and solution is continuously circulated for a period of 12 hours through the electrolyzer having a current density of about 50 amp per square foot of electrode surface. About from 90 amp to 120 amp are used per ton of ore.

"*Example III.*—Two thousand pounds of pulverized ore are suspended in a solution of 2 lb. potassium thiocyanate, 2 lb. caustic soda or lime, 2 lb. potassium bromide, ½ lb. potassium iodide in 2000 lb. of water; the whole mixture, being in the proportion of two parts of said solution to one part of ore, is contained in an agitation tank. This mixture of ore and solution is continuously circulated for a period of 12 hours through the electrolyzer having a current density of about 50 amp per square foot of electrode surface. About from 90 amp to 120 amp are used per ton of ore.

"*Example IV.*—Two thousand pounds of pulverized ore are suspended in a solution of 2 lb. potassium thiocyanate, 2 lb. of lime equal to 2 lb. of protective alkalinity in 2000 lb. of water; the whole mixture, being in the proportion of two parts of said solution to one part of ore, is contained in an agitation tank. This mixture of ore and solution is continuously circulated for a period of 12 hours through the electrolyzer having a current density of about 50 amp per square foot of electrode surface. About from 90 amp to 120 amp are used per ton of ore.

Functions of Electric Current. Chemical Reactions.

"It will be seen that the function of the electric current is simply one to produce nascent oxygen in contact with the ore in said solution. The nascent oxygen in the above examples liberates iodine from potassium iodide which combines with the cyanide or the thiocyanate to form cyanogen iodide. This cyanogen iodide as produced is an excellent solvent for gold contained in refractory ores containing gold tellurides, sulphides, arsenides, etc.

"It may be seen from the following equations the reason for using thiocyanate alone or in conjunction with the simple cyanide solution in preference to the ordinary alkaline cyanide solution is that if cyanide solution be used without containing sulphocyanide or the addition thereof, the nascent oxygen produced by the electric current destroys the cyanide by converting the same into cyanate, which is not a gold solvent, and, therefore, simply results in the destruction of the cyanide—after the equation:

$$KCH + O = KCNO; \qquad (1)$$

whereas by using thiocyanate the equation is

$$KCNS + H_2O + 3O = KCN + H_2SO_4. \qquad (2)$$

It will be seen from the equation No. 2 that cyanide is produced when the solution is kept alkaline, and the cyanide so formed is not decomposed as long as there is thiocyanate present in the solution. By adding a halogen compound to the thiocyanate solution, which may or may not contain free alkaline cyanide and electrolyzing same, the following equation takes place:

$$KCNS + KI + 4O = ICN + K_2SO_4. \qquad (3)$$

"It may be stated at this point that any alkaline sulphides present in the solution would act to regenerate potassium thiocyanate as:

$$ICN + K_2S = KCNS + KI.$$

This reaction would also open up a way for substituting alkaline sulphides for zinc as a silver precipitant.

"The same solution can be used again and again by keeping the requisite strength in thiocyanate, but no further addition of potassium iodide is necessary, except to compensate for mechanical losses. The nascent oxygen produced by the electric current regenerates the iodine to begin work over again. Between each operation it is desirable to pass the liquor through

zinc shavings in the ordinary way to extract the values which it contains.

"In the above examples the desired result is prevented if the solution be substantially acid, but the solution may be alkaline, neutral or slightly acid. By slightly acid I mean this: That while a solution of thiocyanate may be technically termed 'slightly acid' because showing acidity to certain tests, yet for practical purposes it is substantially a non-acid solution, as well as those which are alkaline or neutral.

Use of Ozone.

"Another way of carrying my invention into practice is to treat pulverized ore with a solution of a thiocyanate containing a halogen compound with ozone or with a solution of thiocyanate containing a halogen compound in the presence of an electric current, as the following examples will make clear:

"*Example A.*—Two thousand pounds of pulverized ore are suspended in a solution of 4 lb. of potassium thiocyanate, 2 lb. of potassium iodide, 2 lb. caustic soda or lime in 2000 lb. of water; the whole mixture, being in the proportion of two parts of said solution to one part of ore, is contained in an agitation tank. This mixture of ore and solution is continuously subjected to a current of ozonized air for a period of 12 hours, the amount of ozone passed into solution being about 100 grams per ton of ore. The amount of ozone present in the ozonized air is determined by titration or otherwise. The ozone liberates the iodine from the potassium iodide, which combines with the thiocyanate to form cyanogen iodide. This cyanogen iodide is an excellent solvent for gold contained in refractory ores containing gold tellurides, sulphides, arsenides, etc.

"*Example B.*—Two thousand pounds of sulphide ore are suspended in a solution of 4 lb. of potassium thiocyanate, 1 lb. of potassium iodide, 2 lb. of caustic soda or lime in 2000 lb. of water; the whole mixture, being in the proportion of two parts of said solution to one part of ore, is contained in an agitation tank. This mixture of ore and solution is continuously agitated for a period of 12 hours in the presence of inert electrodes having a current density of about 50 amp per square foot of electrode surface. About from 90 amp to 120 amp are used per ton of ore.

"The function of the electric current in example B is analogous to the function of ozone in example A—that is to say, both the electric current and ozone or ozonized air liberated the halogen from its compound."

New York Meeting of American Iron and Steel Institute.

The first formal meeting of the American Iron and Steel Institute was held in New York City on Oct. 14. It will be remembered that the Institute was founded to represent the commercial and economical rather than the technical side of the industry. Some 200 members were present when the president, Judge Gary, called the meeting to order at the Waldorf-Astoria. An international flavor was given to the meeting by the presence of some 30 prominent representatives of the steel industries of Great Britain, Germany, Austria, France and Belgium. Practically every American steel company and many blast furnace and rolling mill companies of this country were represented.

The proceedings started with the presentation by Judge Gary of his presidential address on the maintenance of steady conditions and co-operation in the steel industry. The address, dealing with fundamental problems of social economy, was exceedingly interesting. It will be found on page 613 of this issue.

The following papers were then presented:

Mr. James A. Farrell, president of the U. S. Steel Products Export Company, spoke on "Foreign Relations."

Mr. Wm. B. Dickson, first vice-president of the U. S. Steel Corporation spoke on "The Betterment of Labor Conditions in the Steel Industry," dealing with prevention of accidents and employers' liability and hours of labor.

Mr. Charles Kirchhoff gave a concise summary of the proceedings of the International Metallurgical Congress, held this summer in Düsseldorf.

Mr. Willis L. King, vice-president of the Jones & Laughlin Steel Company, spoke on "Contract Obligations," dealing with the dividing line between contract and option.

In the evening a dinner attended by over 200 members and guests was held in the ballroom of the Waldorf-Astoria. A very interesting series of lantern slides, showing the town of Gary in the making, was exhibited and commented upon by Mr. George G. Thorp, vice-president of the Indiana Steel Company.

Judge Gary in opening the after-dinner speeches emphasized the idea that the assembled iron and steel manufacturers advocated the gospel of peace and good will and co-operation instead of destructive rivalry. Mr. Charles M. Schwab welcomed the guests from abroad. Sir John Randles replied for England, Baron von Bodenhausen for Germany, Mr. L. Spark for Belgium and Mr. R. de Labriolle for France. Mr. G. S. Smith, Mr. E. Schaltenbrand and Mr. Wm. B. Peat also spoke.

Baron von Bodenhausen, of the Krupp Works in Essen, Germany, put in a good word for competition. But one should know his competitor and be friendly with him. While to know one's customers personally means half of business success, the other half of success is to know his competitors personally.

Mr. Wm. B. Peat, who was the last speaker, thought the day to be past when a policy of cut-throat competition was considered the best either for a country or for individuals. Good feeling between those engaged in competitive commerce is able to smooth over difficulties which would otherwise prove insurmountable.

On Saturday, Oct. 15, an excursion was made by the members and guests of the Institute on a steamer around New York City.

On Sunday, Oct. 16, a tour by special train started covering Buffalo (Lackawanna Steel Company), Niagara Falls, Chicago (Indiana Steel Company at Gary, Illinois Steel Company at South Chicago, International Harvester Company, Union Stock Yards), Pittsburgh (Jones & Laughlin Steel Company, Homestead works of Carnegie Steel Company, works of Mesta Machine Company, Carnegie Technical Schools, Westinghouse Works), and Washington (Bureau of Standards). Social features of the trip were dinners at Chicago and Pittsburgh, and the reception by President Taft at Washington. Judge Gary addressed the President briefly, who in his reply referred to the basic character of the iron and steel industry. The members of the party were then presented to President Taft.

The trip ended on Saturday night, Oct. 22, by the return of the special train to Jersey City.

The Camp Bird, Ltd., Ouray, Colo., treated 6813 tons of ore during August, 1910, producing 6280 oz. of bullion and about 800 tons of concentrates. The financial statement shows about $150,315 applicable to profit. In the annual statement for the year ending April 30, 1910, it is stated that during the year a saving of 95 per cent was made on the gold value of the ore, which is the record saving in the history of the company's operations. Of the total value recovered there was obtained through amalgamation 70.85 per cent; through concentration, 21.71 per cent, and through cyaniding, 7.44 per cent. Of the metals recovered, gold constitutes 95.50 per cent; silver, 2.84 per cent; lead, 1.52 per cent, and copper, 0.14 per cent. At the company's Santa Gertrudis property, Mexico, a mill of 600 tons daily capacity is under construction and should be in operation by March 1, 1911. It will have modern coarse crushing equipment, followed by 60 1550-lb. stamps and 10 tube mills. The plant will include Pachuca tanks and the latest developments in filter and precipitation practice.

Chicago Meeting of the American Electrochemical Society

The great success of the Pittsburgh meeting in the spring of this year encouraged the officers of the American Electrochemical Society to go still further westward and to accept the invitation of the newly formed Chicago local section to hold its fall meeting in Chicago.

The success of the Chicago meeting has proved that this was a wise policy. It will strengthen the influence of the society in the West, where there is open for it a wide and important field of usefulness.

The local Chicago committee had arranged all details with greatest carefulness and foresight and all Chicago members did their best to make the visitors feel at home and succeeded admirably. Special mention should be made of the indefatigable work of the chairman of the local committee, Dr. H. N. McCoy, and the secretary, Mr. Arva B. Marvin, as well as the chairmen of the various sub-committees, Mr. Harry McCormack, Entertainment Committee; Mr. F. T. Snyder, Excursion and Transportation Committee; Mr. H. T. Matthew, Press Committee, and Mrs. W. D. Richardson, Ladies' Committee.

W. H. WALKER, PRESIDENT, AMERICAN ELECTROCHEMICAL SOCIETY.

The attendance at all sessions was very gratifying. The last printed registry list contained 213 names, of which 100 were from other cities than Chicago.

Of particular interest were the visits and excursions which had been arranged for the three afternoons. There were, indeed, so many of them that it was often hard to select the one most desirable out of three or four alternate excursions for one afternoon.

Of greatest metallurgical interest were the visits to the magnificent Gary steel works of the Indiana Steel Company and to the equally famous South Chicago works of the Illinois Steel Company. At the latter place the 15-ton Héroult electric steel furnace (see description in our April issue, p. 179) was seen in operation.

Of great general interest was the visit to the Union Stock Yards and packing plants, where the most interesting sections of the plants of Swift & Co., Armour & Co. and Morris & Co. and the Libby canning establishment were visited. At the Morris plant the chemical laboratory and the butterine factory were shown. Chemistry plays a greater part in the packing plants than is generally appreciated. Thus in the laboratory of Swift & Co., which is in charge of Dr. W. D. Richardson, some thirty chemists are employed and it is interesting to learn that about half of the work is analytical routine work in connection with the soap, etc., manufacture, while the other half is research work to find new uses for the by-products.

Of equally great general interest was the visit to the mail-order establishment of Sears, Roebuck & Co., with a unique organization which in its way is not equaled elsewhere.

There was also quite a number of excursions to plants of electrical interest. Among them were the Fisk Street power house (a 120,000-kw steam plant with 12,000-kw generating units driven by Curtis steam turbines) and the Market Street converter and storage battery station of the Commonwealth Edison Company; further the telephone and instrument factory of the Western Electric Company, the factory of the Automatic Electric Company, where 1300 persons are employed in the manufacture of automatic telephones.

The Hoskins Manufacturing Company (represented by Mr. W. V. Young and Mr. A. L. Marsh) had at its salesroom an interesting exhibit of various electric furnaces for laboratory work, and especially for steel works laboratories, in connection with pyrometers. An instructive pyrometer exhibit, comprising quite a number of different types, had been arranged at the Congress Hotel by the Taylor Instrument Companies of Rochester, N. Y. (represented by Mr. R. S. Schwarz and Mr. J. A Sutherland), while the firm of E. Leitz, of New York and Chicago (represented by Mr. H. Vollrath), exhibited its new metallographic microscope.

For Saturday afternoon visits to the laboratories and museums of the University of Chicago and to the Field Columbian Museum had been arranged, while another party visited the Lockport power station and drainage canal. The canal is one of the world's great artificial waterways. It receives the drainage of the city diluted by a large volume of pure water from the lake. At Lockport, 35 miles from the lake, the canal empties into the Des Plaines River with a 36-ft. fall. The Sanitary District has at this place a modern generating plant of 36,000 hp.

On the evening of Friday a delightful informal subscription dinner was held at the Congress Hotel, Mr. H. B. Coho acting as toastmaster. Among the speakers were Mr. Harry McCormack, chairman of the Sanitary District, who gave some interesting details of the building of the drainage canal and its connection with politics, and Mr. E. A. Sperry, chairman of the New York section, who invited the society to hold its next meeting in Spring, 1911, in New York City.

The two professional sessions of Thursday and Friday morning were held in the Congress Hotel, that of Saturday morning in the lecture room of the Kent Chemical Laboratory of the University of Chicago.

President Dr. Wm. H. Walker called the first meeting to order on Thursday at 10 o'clock. The first two papers dealt with the general subject of heat insulation of electric furnaces.

Determining Thermal Conductivity.

A paper by Mr. CARL HERING, of Philadelphia, Pa., suggests a new method of determining the thermal conductivity of refractory materials, such as used for furnace walls. In determining the heat flow through a specimen the lateral leakage of heat is troublesome. Mr. Hering shapes his test piece so that it has no lateral sides at all.

The ideal case is to shape the sample piece in form of a hollow sphere. The inside is filled with a good heat-conducting material, like a molten metal, which is heated electrically, say, by means of an insulated coil of wire. The outside is surrounded by any suitable liquid, preferably kept in good circulation as it receives the flow of heat; this liquid is provided with a cooling worm which carries off the heat exactly as fast as it arrives through the sphere. The temperatures of the inside and outside surfaces of the sphere are determined by means of pyrometers or thermometers and the electric energy set free in the inside is measured by means of an ammeter and a voltmeter.

These measurements are made when the stable state has been reached. The thermal conductivity k (in watts for 1 in. cube for one degree C.).

$$k = \frac{l}{VS} \frac{W}{T}$$

Where W is the energy in watts set free in the inside, T the difference between the inside and outside temperatures in degrees C., t the thickness of the shell in inches and S and s the outside and inside surfaces in square inches.

The conductivity may vary with the temperature so that it would be different in different parts of the sample. But the above method gives directly the "true and correct average or effective mean" of the conductivity for that range of temperature, that is, it is that conductivity which an equivalent piece of material would have whose conductivity is constant over that range of temperature, and which as a whole acts just like the sample.

While the hollow sphere is the ideally perfect form to use in this test, because it is absolutely symmetrical all over, it is probably also the most impractical. Mr. Hering suggests as a convenient form of the test samples a cylindrical cup, rounded at one end and open at the other, as shown by C in the adjoining Fig. 1, which is a diagrammatic vertical cross-section of the whole apparatus. The inside is nearly filled with the heating liquid containing the heating coil; the outside vessel is nearly filled with the cooling liquid and its cooling worm. T and t are the two temperature measuring instruments. When both temperatures are very high the outside liquid may also be a molten metal, in which case it had better also be provided with a heating coil to save the time it would take to heat it with the heat which has to pass through the test piece; but the heat generated in this coil, of course, need not be measured.

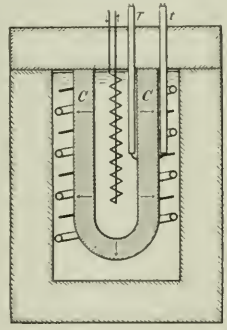

FIG. 1.—APPARATUS FOR MEASURING CONDUCTIVITY

The objection to this form is that there will necessarily be some loss of heat through the cover over the top. This can be made relatively small, however, and can also be allowed for. Care should, of course, be taken to mold or grind the test cup so that its wall thickness is the same throughout. For testing loose, granular or fibrous materials, like infusorial earth, sand, asbestos, magnesia, etc., the cup may be made of two thin shells of iron, the material to be tested being filled in the space between them.

For such a cylindrical cup, with its one end rounded off to a half sphere, the value of the thermal conductivity k is as follows

$$k = a \frac{W}{T}$$

in which a is a constant, calculated once for all for each test piece, and is equal to

$$a = \frac{1 \log D/d}{2.7288\,l + 1.5708\,D\,d \log D/d}$$

k is in watts for 1 cu. in. or 1 cu. cm., depending upon the linear units used for the other quantities; W represents the watts generated in the inside of the cup when the stable state has been reached (after deducting the loss through the lid). l is the length of the straight part of the cup, down to the beginning of the hemispherical bottom. t is the wall thickness of the cup. D and d are the outside and inside diameters, respectively. T is the drop of temperature in centigrade degrees between the inside and outside surfaces of the cup, all linear dimensions must be either in inches or in centimeters and the value of k will then refer to inches or centimeters respectively. $\log$ in the above formula means common logarithm. The constants in this formula are purely mathematical and involve nothing empirical.

In the discussion Dr. Richards referred to experiments made by Prof. Landis with a hollow sphere, electrical energy being generated at the center and thermocouples being inserted at different points of a radius to determine the gradual temperature drop from the inside to the outside. When a graphite sphere was used it was impossible to keep the outside temperature of the sphere cool as desired on account of the good conductivity.

Mr. F. T. Snyder referred to the method of Clement and Egy, which was described and illustrated in our July issue, page 414. They use a test piece in form of a cylinder and this has the advantage that manufacturers of refractory material have no difficulty in getting the test piece into this form.

Flow of Heat Through Furnace Walls.

Mr. F. T. Snyder, of the Metallurgical Engineering Company of Chicago, presented a paper on the flow of heat through furnace walls. He analyzed the part which heat convection and heat radiation play in compound furnace walls, consisting of several layers of different materials.

Mr. Snyder uses as the unit of heat flow the temperature drop in degrees Centigrade per inch of thickness per kilowatt per square foot. The constituent units are the actual ones in which the data comes to the designer. The furnace drawings give dimensions in feet and inches; American refractories are made in inch dimensions and the pyrometers usually available are calibrated in Centigrade degrees.

The radiation of heat from the outside surface varies with the difference between the fourth powers of the absolute temperatures of the surface and the air, multiplied by a constant to deduce the results to kilowatts of heat flow per square foot. The convection of heat from the surface is affected by two factors, the transfer resistance of the flow of heat from the surface to the air, and the movement of the heated air away from the surface.

The radiation according to Stephan's law is

$$R = a\,(T^4 - t^4)$$

where

R = kilowatts per square foot.
T = hot temperature.
t = cold temperature.
a = 0.0033×10^{-9} for ordinary sheet iron.
a = 0.0042×10^{-9} for brick and slag.

The convection of heat from the surface to the air as function of the velocity v of the convection currents is given by the formula

$$C = b\,(T - t)\,(2 + 5.5\,\sqrt{v})$$

where

C = kilowatts per square foot.
T = temperature of furnace surface
t = temperature of air.
b = 0.00011 for sheet iron furnace shell.
b = 0.00015 for black furnace surface.
v = velocity of convection currents in feet per second.

The velocity of the convection currents is in turn determined by the temperature of the surface. Strictly this velocity is also affected by the shape and position of the surface, but practically it has been found from measuring the velocity of the air for different points on the bottom, sides, and top of a number of commercial electric furnaces and plotting it against the temperatures of the surface for the same points that this velocity can be expressed as a simple function of the difference between the temperatures of the surface and the air. This empirical formula is

$$v = 0.0022\,(T - t)^{3/4}$$

For the flow of heat through the wall material an experimental constant is determined giving the temperature drop in degrees C. through one inch of thickness of the material for a heat flow of 1 kw per square foot. This constant varies for any material with the temperature range and should be used only for the range for which it is determined. If not available for a desired range, it may be approximated from the

constant of a known range by varying the known constant inversely as the absolute temperatures of the means of the two ranges.

A set of data for different materials is given in the following table:

Thermal Conductivity of Refractories.

D = Degrees C. required to force a heat flow of 1 kw per square foot through 1 in. of thickness of the material.
T = Temperature at hot side of material.
t = Temperature at cold side of material.

	t	T	D
Alumina brick	0	700	320
Ashes, wood	0	100	3840
Brick dust, coarse	0	100	1680
Carbon, retort, solid	0	100	45
" electrode	100	400	7.1
" "	100	800	5.2
" "	100	1200	5.0
" "	100	1600	4.5
Carborundum sand	18	98	1300
Cement	0	700	3800
Charcoal	0	100	2960
Clinker, small grains	0	700	600
Coke, powdered	0	100	1480
Copper metallic	0	30	0.7
Fire brick	0	500	465
" "	0	1000	155
" "	0	1300	105
" " dust	20	98	2300
Graphite, dust	20	100	1630
" electrode	100	400	1.9
" "	100	800	2.1
" "	100	1200	2.3
" "	100	1600	2.5
Infusorial earth	17	98	5000
" "	0	650	1720
Iron, metallic	100	200	4.4
Lime	20	98	2250
Magnesia, brick dust	20	100	1300
" calcined, Grecian, granular	20	100	1450
" " Syrian "	20	100	1920
" " light, porous	20	100	4100
Quartz sand	18	98	1090
Silica brick, burned to 1050 C	0	1000	330
" " " 1310 C	0	1000	210

Mr. Snyder shows in detail how to calculate the heat flow and temperature distribution for a proposed furnace wall and gives various curves for practical use. Finally a numerical example is added on the calculation of the temperature distributions through the bottom of a small steel furnace.

In conclusion, Mr. Snyder remarks that with regard to the published data of the actual heat flow through composite walls, allowance should be made for the fact that the water jacket usually used to absorb the heat for measurement on the outside changes the regular operating conditions, firstly by cutting off the convection and secondly by cooling the outside of the wall below its usual working temperature.

In the discussion which followed Mr. Hering referred to the fact that there is no temperature equilibrium between a moving gas or liquid and a surface in contact with it. There is at the contact a sudden temperature drop. This is, for instance, so in boilers.

Dr. Richards referred to a simple method of testing a given furnace of given wall thickness and given dimensions. The temperature of the outside of the furnace may be explored by means of an iron-nickel wire thermocouple. The outside temperature determines the loss. The radiation losses can be reduced by experimenting with the form of the outside shell of the furnace, since the radiation depends on the roughness, porosity, color, etc., of the surface. The harder the outside shell the more economical is the furnace."

Reduction of Tin Dross.

Mr. R. S. WILE, of the Riverside Metal Refining Co., of Connellsville, Pa., discussed the reduction of tin dross in an electric furnace.

Electrical heat was resorted to for the smelting of tin dross because of the fact that the heat could be internally applied to the slag, which is on the bottom of a shaft-type of furnace, thus enabling the dross to be thrown on top of the slag instead of being mixed with it as is done in the old style of furnace. The dross, being on top, comes in contact with the slag only at the point of reduction. The liberated gases filter through the dross, while any tin oxide which is volatilized is condensed in the colder portion of the dross, which is on top of the slag. The globules of tin produced in smelting pass downward through the slag and lose most of the impurities, so very little refining of the resultant product is necessary.

In operation, the top carbon, which is movable, is brought into contact with the lower carbon, which is stationary, and an arc formed. The slag is fed in and melted, and the carbon is raised until the desired amount of slag is added. The dross, mixed with the right percentage of carbon, is added, and the tin tapped from the bottom from time to time. The loss of tin has been kept as low as 0.25 per cent, and the average below 1 per cent. The amount of tin recovered varies largely on account of the varying percentages in the drosses treated. The average is about 2500 lb. (1100 kg) per day.

The plant consists of two furnaces, connected in series, both being 20 in. (50 cm) in diameter and 80 in. (200 cm) high inside, two 50-kw transformers and necessary electrical apparatus. In operation they consume about 44 kw. During the run it is desired to keep the amperage as near constant as possible, the voltage varying. At the start the voltage of each furnace is about 80, but toward the end, as the slag becomes less refractory, due to the combination of the iron and zinc of the dross and the slag, the voltage of each furnace will drop as low as 45 or 50. This and an analysis of the slag denotes the end of the operation, and the slag must be drawn out and new put in. This is done alternately with each furnace. The furnaces run continuously until time to be relined, which is about every three or four months.

In the discussion Dr. J. W. Richards remarked that this electric furnace process represented a rather general method which could prove useful for other purposes than the reduction of tin dross.

Cast Copper of High Electric Conductivity.

A paper by Dr. E. WEINTRAUB, of the West Lynn, Mass., research laboratory of the General Electric Company, was then presented on a new method of producing copper castings of high electric conductivity.

The production in the foundry of pure copper castings mechanically sound and possessing an electrical conductivity comparable to that of pure forged copper has been an important problem ever since the development of electrical machinery and apparatus. The literature as well as the patent files are replete with descriptions of methods for accomplishment of this result. Secret compositions, warranted to give high conductivity copper, were and are being sold to foundrymen, and there are but few foundry foremen to be met with who do not claim to have some time or other obtained the desired result by adding "phosphorus" or similar material. Obstinately, however, the casting of high-conductivity copper refused to make its entrance in the foundries and it may be assumed that there were good reasons for it.

The cause of the difficulty of producing sound pure-copper castings has been sufficiently well understood for a long time. Molten copper has the property of dissolving gases such as oxygen and of setting a part of these gases free on cooling. This produces pin holes and even big cavities. The casting obtained is, therefore, mechanically unsound and has naturally a low electrical conductivity.

The elimination of these dissolved gases presents but little difficulty. It is sufficient to add one of the well-known deoxidizers such as zinc, magnesium, phosphorus, etc., in small quantities to bind the oxygen chemically. Copper castings free from blow holes are thus easily produced. The electrical conductivity of the copper thus produced is, however, as a rule, low; and this because all the deoxidizers used had also the property of combining with copper, and the remarkable rate in which the conductivity of copper drops when small amounts of impurities are alloyed with it is well known. The amount of oxygen dissolved in copper during the process of melting is a variable quantity and is distributed throughout the whole mass. It is, therefore, practically impossible to add the deoxidizer in such a way as to eliminate oxygen completely without, at the

same time, alloying it to a certain degree with the copper. Accordingly, the conductivity of the resultant product was a matter of accident. In the laboratory sometimes good results were obtained, but they were hard to duplicate and the transfer to the foundry was always a disastrous experiment.

Dr. Weintraub argued that what was needed for the solution of the problem was the finding of a deoxidizer which would have no affinity for copper (at least at the melting point of the latter) and which, therefore, could be added in excess with impunity.

While carrying on the work of the isolation of the pure element boron (this journal, vol. VII, page 509) and its fusion, the lack of affinity of this element for copper manifested itself. Boron has, at the melting point of copper, a great affinity for oxygen, nitrogen, etc., and it occurred to Dr. Weintraub, therefore, that boron ought to be the ideal substance for deoxidizing copper.

Pure boron, as prepared by methods described in his former paper, can by this time be produced in large quantities and is rather inexpensive for uses based on its electrical properties. For casting copper, however, its price would probably be too high.

In the course of the investigation mentioned he had prepared, however, an amorphous powder which contained as essential impurity only a certain percentage of oxygen and which he termed "boron suboxide." The substance is prepared by a modification of the reduction-process of boric anhydride by magnesium. Obviously for the purpose of eliminating oxygen from copper without introducing any impurity capable of alloying with copper this boron suboxide would be as good as pure boron or perhaps better on account of its being obtained in form of a very fine powder.

This substance was used in the first experiment. The very first attempt was successful. A perfectly sound bar was obtained with a conductivity of 94 per cent; 0.1 per cent of the weight of the copper was added. Further experiments showed that the amount of boron suboxide could be reduced to 1/30 per cent and also that the addition of a large excess, say of 1 per cent, did not diminish the conductivity or change in any way the mechanical properties of the cast copper.

Here was, therefore, a method which could be put into the hands of any foundryman and which would always give good results provided the copper (ingots or scrap) melted down was pure metal.

A further simplification could, however, be introduced into the process. The reduction of boric anhydride of magnesium, the first being in excess, gives a product containing magnesium borate, boric anhydride and boron suboxide. While the last one is the active agent, the first two are harmless. In fact the presence of boric anhydride and magnesium borate is even an advantage, as the addition of a larger mass of material makes the operation easier; also the boric anhydride would probably dissolve the copper oxide and bring it to the surface as copper borate. It is therefore unnecessary to submit the direct product of the reaction to chemical treatment for the purpose of isolating the boron suboxide. It is sufficient to grind up the mixture into coarse grains and use an amount of it which contains the necessary amount of boron suboxide.

The foundry practice at present is to add 1 per cent to 1½ per cent of this material, which is equivalent to 0.08 per cent to 0.1 per cent of boron suboxide. The properties of the material obtained are as follows.

The electrical conductivity obtained can be as high as 97½ per cent if the copper melted down is perfectly pure. In the foundry, where the scrap copper which is melted down cannot always be very carefully selected, a conductivity of 91 to 95 per cent is obtained and a guarantee of 90 per cent is given.

The resultant casting is readily machined; fears which were entertained on this subject did not materialize and the engineers using the material have not complained on this account. The mechanical properties are as follows:

Tensile Strength.	Elastic Limit.	Elongation.	Reduction in area
24,350	11,450	48.5 per cent.	74.49 per cent.

Castings are being made of all shapes and sizes with no more difficulty than in the case of brass. Sand and iron molds can be used and for standard articles the latter are preferred.

The use of cast copper lies in two different directions: (1.) Replacing forged copper. In this case a great saving in cost is often obtained. (2.) Replacing alloys of copper. The gain in conductivity allows a reduction in bulk and thus a saving in material.

The use of this material is spreading as the engineers get better acquainted with it and is already of considerable magnitude. It is interesting to note that in spite of the fact that cast copper has lower conductivity than forged copper in many cases, as for instance, in case of current transformers, cast copper is being used now instead of forged copper. This is due to the fact that in casting a chance is given to eliminate riveted, screwed or soldered joints, whereby a number of poor contacts are avoided. Not only this, but the elimination of joints often insures a better operation.

The cost of the boron added is small; at the present prices of magnesium and boric anhydride the cost per pound of copper is less than ¾ cent.

The success with "boronizing" copper, as it is now called in the foundry, suggests new lines of work which have been taken up lately by Dr. Weintraub. Among these are mentioned:

First, the addition of boron to different copper alloys.

Second, the replacing of the poling operation in the process of refining copper by a "boronizing" operation.

In conclusion Dr. Weintraub acknowledged the assistance of Mr. F. A. Kroner.

Induction Furnace for Steel Melting.

Dr. Jos. W. Richards, of Lehigh University, presented a paper on the Hiorth induction furnace, which is in commercial operation for steel melting in Norway. He first made some general remarks on the status of electrometallurgy in Norway and Sweden.

At the Hiorth plant, which is located on a fjord in Norway, there are available 1000 hp electric power, of which 500 hp are used in the Hiorth furnace plant and the balance for a pulp mill. The entire cost of installation was $25 per horse-power and the total cost of the electrical energy, including interest and depreciation, is stated to be between $3.80 and $4.10 per electric horse-power-year. This is, according to American ideas, an exceedingly low price, but Mr. Eyde has added that equally cheap power is available elsewhere in Norway. In addition, these water-powers are located right next to the ocean and there is direct deep-water communication with the markets of Europe. The only drawback of Norway is the labor problem, due to the lack of population.

It is different in Sweden. There the water-powers are somewhat more expensive, the cost of the horse-power-year being between $7 and $10. But the labor supply is better. Further, Sweden has had for many years considerable experience in iron and steel making.

The Hiorth furnace is a double induction furnace, the two rings being in communication with each other at the center, where a wide hearth is provided for carrying out the operations quite similar to the Roechling-Rodenhauser arrangement. The arrangement of the furnace and of the primary coils (which are in two parts, one above and one below the bath) is such that the magnetic leakage is claimed to be small.

The frequency of the alternating current employed is 12.5 cycles and the power factor is stated to vary between 0.75 and 0.85.

Means are provided for supplementary heating in the central compartment, quite similar to the Roechling-Rodenhauser furnace, to keep its temperature up. But in actual practice it has been found that this supplementary heating was not absolutely necessary; even when the furnace was run as a pure induction

furnace, the temperature of the central compartment was quite as high as that of the outside rings. While the current density was low in the central compartment, yet the radiation loss was small.

The metallurgical process carried out is exceedingly simple. What they are doing at the Hiorth works is to take the same high-grade materials which are melted in Sheffield in crucibles and melt them in the Hiorth electric furnace. These are Dannemora pig iron, costing about $25 a ton, and Dannemora soft iron, costing $56 a ton. The ingots are shipped to Sheffield. At the Hiorth plant nothing is being done but melting down the starting materials to ingots. The plant is commercially successful.

The lining of the hearth of the Hiorth furnace is magnesite. They use Syrian magnesite, since their experiments with Norwegian magnesite were not successful. Dr. Richards thinks that the reason was probably that it had not been calcined at a sufficiently high temperature. For full success it is important that the magnesite be very hard burned.

The slag from Dannemora blast furnaces has been used successfully in the electric steel furnace.

As to the specific energy consumption, Dr. Richards stated that at the time of his visit to the Hiorth plant 700 kw-hours were required per ton of cold material charged into the furnace. But since that time he has been informed that this figure has been reduced to 685 to 690 kw-hours.

The steel produced in this way has proved in every respect equavalent to the best Sheffield crucible steel.

While the Hiorth furnace is thus used simply for melting down high-grade starting materials, Dr. Richards pointed out that another application would be the dead-melting of ordinary open-hearth or Bessemer steel; the power consumption would be low and the steel would be improved over ordinary open-hearth and Bessemer steel, as is known to be the case.

Since the large central space is available, a certain amount of refining could also be done, but the two special fields of the Hiorth induction furnace are the melting down of high-grade material and the dead-melting of ordinary steel.

The paper was discussed by Messrs. Seede, Hering, Bennie and Frary.

Triboluminescence.

The first paper presented at the Friday session dealt with triboluminescence, the author being Mr. W. S. ANDREWS, of the General Electric Company. Triboluminescence is the property possessed by certain bodies of emitting light when rubbed or scratched. An artificial product showing triboluminescence can be easily prepared from 70 parts by weight of chemically pure zinc carbonate, 30 parts of flour sulphur and a trace of manganese sulphate.

The zinc carbonate in fine powder should first be mixed with the flour sulphur. Then a small particle of manganese sulphate should be dissolved in distilled water and enough of the solution added to the powdered materials to make a thick cream. After being thoroughly triturated in a mortar, the mixture can be poured into a shallow glass or earthenware dish and allowed to dry at a gentle heat. When quite dry it should be again reduced to a fine powder and packed hard into a porcelain or Battersea crucible with a tight cover and subjected to a bright red heat for twenty minutes. The mixture shrinks considerably and sinters together into a stone-like mass, which possesses the peculiar property of triboluminescence to a marked degree, so that when scratched with a knife blade or sharp point it emits a train of minute sparks showing a yellowish light. This light appears to have little or no heating effect as it will not ignite any inflammable vapor or gas, so that in this regard it differs essentially from the light and sparks produced by striking together flint and steel or the scratching of a pyrophoric alloy of iron and cerium.

A sample of this material exhibiting triboluminescence was shown and it was suggested in the discussion that the phenomenon might be due to the formation of zinc sulphide.

Electrolytic Preparation of Calcium.

Prof. ARDEN R. JOHNSON presented a paper on the electrolytic preparation of calcium, giving the results of experimental work carried out in 1905-6, but not published before.

Two different designs of apparatus are distinguished; first, that of Borchers, using a very high current density and consequently obtaining the calcium in molten condition, in which it rises to the surface and is skimmed off; and, second, that of Rathenau, in which the calcium is plated out at a moderately low current density and is obtained in form of a solid stick which is gradually raised out of the electrolyte.

The author first tried an apparatus of the Rathenau type and described the difficulties which he encountered with it and with a modified form of apparatus. These were especially the formation of nodules and trees and the contamination of the metallic calcium deposit with undecomposed calcium chloride. In the last experiments an iron ribbon was used as the cathode on which the calcium was deposited and which was gradually raised, presenting new cathode surface.

A considerable amount of data are given in the paper on current efficiency, chemical changes in the electrolyte, and decomposition voltages.

In the discussion Prof. Frary said that no troubles are experienced in the production of metallic calcium on a commercial scale at Bitterfeld by the Rathenau process and that Prof. Johnson's troubles were chiefly due to not using an artificially cooled cathode. Prof. Johnson thought that his ribbon cathode was a kind of cooled cathode. According to Prof. Frary, current density is important only as it determines cathode temperature. The cathode temperature is the main thing. When the metallic calcium is set free in molten state, it should be frozen as soon as possible. At Bitterfeld rods of calcium are produced 1 and 1½ in. in diameter, perfectly solid and containing no chloride.

Efficiency in Production of Metallic Calcium.

Prof. FRANCIS C. FRARY, of the University of Minnesota, presented a paper on the ampere-hour efficiency in the electrolytic production of metallic calcium. Messrs. H. R. BICKNELL and C. A. TRONSON being joint authors with him.

The main thing, in order to obtain a good efficiency, is to withdraw the cooled calcium stick cathode at the proper rate. The addition of electrolyte is also of importance, and both the raising of the cathode and the addition of electrolytic should be continuous. The aim of the authors was to withdraw the cathode as rapidly as was possible without striking an arc. Practice in doing this increased the yield obtained. In none of the ten runs made the ampere-hour efficiency was less than 45 per cent and in three runs it was raised to practically 100 per cent.

When the electrolysis is going on properly, and the stick of metal is at the right depth, the electrolyte seems to flow rather rapidly across the surface in a fixed direction and is hottest at the cathode. The operator must be guided by the appearance of the electrolyte at the cathode. There should be a rosette-like spot with radial markings here; if the stick of metal is pulled out too fast this spot becomes almost white hot, and if the metal is not at once lowered an arc forms, and part of the metal is melted off and lost. If the stick is not raised fast enough the spot becomes less noticeable and spongy metal deposits below the surface, causing losses. When the electrolysis was well under way new electrolyte was added in small portions, taking care to make these additions as continuous as possible, and in such a spot that the convection currents carried the cold chloride away from the metal.

The authors tried both plain chloride and a mixture of chloride and fluoride as electrolyte, but from their experiments they conclude that the plain chloride is a little more satisfactory than the mixture of the chloride and fluoride in point of efficiency and decidedly easier to manipulate. Satisfactorily high efficiencies can be obtained with it, and efficiency is more

largely a matter of careful regulation of conditions than anything else.

In reply to a question of Mr. Hering as to the watt-hour efficiency Professor Frary explained that the voltage applied varied between 18 and 42, which is many times the theoretical voltage (3.2 volts), so that the watt-hour efficiency may be less than 10 per cent. But the excess energy is not wasted, as it is employed for heating.

Polonium.

Prof. A. R. Johnson presented a preliminary paper on the electrochemistry of polonium and other radioactive elements. He prepared the material from pitchblende by the methods used by Boltwood and others. He tried to separate the polonium by plating it out electrolytically. Polonium is a metal less electropositive than copper and bismuth. It is precipitated by copper, but not by mercury. In electrolysis polonium moves in the electrolyte with the current like metals.

The quantity of polonium plated out of the solution was so small that it could not be weighed, but only detected by its radioactivity. The author made some speculations as to the applicability of Faraday's law. If polonium behaves in electrolysis like other metals, why does it require such an enormous amount of coulombs to deposit an infinitesimal amount of polonium? He suggested the possibility of an abnormally high valency or an abnormal electrochemical equivalent.

In the discussion Dr. Schlundt referred to similar experiments of his own. To make the deposition complete it takes several days. Dr. Kahlenberg pointed out that in order to apply Faraday's law one must make sure that nothing but polonium is deposited, or it must be known how much of other substances is simultaneously deposited. Dr. Richards and Mr. Hering pointed out that as polonium was present in an almost infinitely dilute solution, it was not surprising that it took so many coulombs to get it out.

Ozone.

Mr Ralph D. Small, of the Standard Electro-Utilities Company, of Chicago, presented a paper on the present status of ozone, with special reference to air purification.

There are various ways of producing ozone, such as the oxidation of phosphorus or the action of ultra-violet rays upon air, these rays being produced by a mercury-vapor lamp in a quartz receptacle, or by heat, according to Nernst and Clement, by which they produce 3 grams per kw-hour But commercially ozone is now chiefly produced by electric discharges through air.

Alternating current is required. Direct current must first be converted into alternating current, peferably with a frequency of not less than 60 cycles. The voltage is then raised in transformers up to any voltage between 5000 and 30,000, although it is now thought that to obtain the highest economy a voltage below 8000 is poor practice. The discharge is produced between electrodes on either side of a dielectric, such as glass, micanite, mica or some high-insulating material, there being an air-gap between the dielectric and the electrodes. The electric discharge, which is not accompanied by sparks and which is comparatively heatless, is a brush discharge, purple in color and fairly quiet in its action.

The consensus of modern opinion is that point electrodes, such as employed by Andreoli and others, have various disadvantages, firstly, because they quickly lose their pointed character and round off, secondly, because a minute particle of dust can do a great amount of destructive work on a small point in generating heat and interrupting the flow of electricity thirdly, because the necessary cleaning of an indefinitely large number of points requires too much attention.

Other types of electrodes are either plates or cylinders on either side of a dielectric material, and these may be made in various forms, flat, serrated, graded or perforated. Another type is the covered electrode with no metal exposed, which is very recommendable.

The fourth type is the rotating fan electrode, such as is used in the Vohr ozonizer, which has the advantage of quickly forcing the air through the generating field and ozonizing it at the same time, thus yielding a product of high purity. It is generally regarded as axiomatic that the briefer the period of ozonization the purer the resultant gas, purity meaning practically an absence of oxides of nitrogen, the formation of which is favored by heat, sparking or the presence of dirt.

The amount of nitrous oxide generated by any properly made type of air ozonizer, however, is so infinitely small that it can be totally disregarded. This statement, however, does not apply to all ozonizers.

The composition of normal air has recently been given in parts by volume per 1000 as follows: 769 nitrogen, 206 oxygen, 14 aqueous vapor, 9 argon, 0.33 carbon dioxide, 0.10 hydrogen, 0.008 ammonia, 0.0015 ozone and 0.0005 nitric acid. Authorities differ as to the amount of ozone in the air. It has been given all the way from 1 part in 1,000,000 by volume up to 1 part in 20,000,000. Perhaps a figure from 1 in 1,000,000 to 1 in 5,000,000 would represent what most have agreed upon as correct for those places where the presence of ozone is easily detected, so that the percentage of ozone in air given out by the best constructed ozonizers of later years has been in alignment with, or very little in excess of, the concentration which nature provides in localities remarkable for their invigorating air.

If a much higher density is used another factor must be relied upon to dilute the ozone to a point where it is agreeable to breathe—that is, the rest of the air in the room. This is unscientific because one is dealing with a factor outside of control, and the air currents of any room differ from those of every other. The machine that delivers the ozone with a continuous uniform concentration and with a controllable yield is built along sane, scientific lines.

Ozone can be condensed to a liquid state of a bluish color, with a specific gravity of 1.46. In this state it boils at about 120° C. Its density is 1.5 10 as compared with oxygen and 1.66 100 as compared with ordinary atmospheric air. It is slightly soluble in water and more soluble in certain oils. It reverts to oxygen at 270° C. It is unstable and rapidly disintegrates on standing. The loosely attached third atom is constantly tending to fly off and combine with other atoms as it has a remarkable affinity for all carbon or for compounds of carbon, which so far as ventilation is concerned may be referred to as organic matter.

Its value as an oxidizing agent is due to the ease of application and because nothing but oxygen is used in reaction. In the lighter concentrations its odor faintly resembles that of freshly bleached linen and is crisp, refreshing and exhilarating. It can be detected by the nose when present in one part to 10,000,000 of air. In the higher concentrations its odor is pungent, heavy and disagreeable.

Heat and humidity are unfavorable conditions for the generation of ozone. For this reason different methods have been employed to keep the electrodes cool. Steynis uses ammonia gas in pipes. Gerard uses oil. Others employ water jackets. The preferable method for air purification is the rapid passage of air by suction pumps or fans, such as are found in the Siemens & Halske later models in Germany and the Vohr model in the United States.

Any type of ozonizer where no mechanical means are provided for forcing the air rapidly past the electrode surface is considered to be utterly unreliable. In the first place, the air will remain for too long a period in the heat zone and thus produce an unsafe amount of nitrous oxide. Furthermore, such a type of generator is unscientific, as there are no means of obtaining either concentration or yield, since there is no regular air flow to measure by.

The amount of ozone generated is in direct proportion to the wattage of discharge per unit of air ozonized. Some element of heat is present in the so-called "cold" discharge, and this tends to destroy a portion of the ozone generated and

also destroys the theoretical accuracy of the above statement. Heat increases as the square of the current of discharge. It is, therefore, desirable to use a high voltage and low amperage and it has been found that from 8000 to 12,000 volts produce ozone with commendable economy.

Higher concentrations are needed for such purposes as water purification, the bleaching of fabrics, the destroying of disease germs, treating of petroleum oils, cyaniding (where with ozone it is now possible to get 94 per cent efficiency off the plates where in many instances only 60 per cent was obtained before), sterilizing infected rooms, ships, etc. For all of the above from 1 up to 12 grams per cubic meter is required. As high as 30 grams concentration has been obtained. High concentrations necessitate high wattage of electrical discharges per unit of air. Here the air passage must be slow, which, of course, results in low volume of ozonized air. The converse, of course, holds true that the more rapid the air flow the higher will be the volume of ozonized air, but the lower the density or concentration.

For all these large concentrations it is better to dry the air by the use of quicklime or other means, but for ventilating purposes it is not necessary to resort to this in the more recent types of ozonizers. One of the objections to extraordinary humidity is that an infinitely small amount of peroxide of hydrogen is liable to be formed.

The determination of the quantity of ozone generated is frequently obtained by using iodide of potassium solution. The iodine is freed by ozone, is acidified, then titrated by sodium thio-sulphite and starch.

The author then discussed at great length the chemical and physiological problems of air purification. What is the harmful part of bad air? Carbon dioxide in itself is not harmful. It is harmful only in that it reduces the oxygen supply by replacing it.

As long as one can get the proper percentage of oxygen in the air to breathe the amount of carbon dioxide produced in any chemical laboratory can be twenty times that which some engineers have put as a figure destructive to life. Carbon dioxide is not poisonous, but when present in considerable quantity in any air it is a danger sign that impurities exist in the air. However, it is a sign-board of danger only, not the danger itself. It is the organic impurities exhaled at the same time as the carbon dioxide that are poisonous.

Organic impurities come from a variety of sources. In all confined spaces the chief source of air pollution is the organic matter cast off through the pores of the skin and the nose, throat and mouths of the individuals occupying that space. The longer that matter which the body has cast off remains suspended in the air the quicker it deteriorates. When it reaches a putrescent state it is unfit to be rebreathed. This is air sewage or "crowd poison."

While ventilation is necessary to bring in out-of-door air to replace the oxygen which our breathing uses up outdoor air alone is absolutely insufficient to purify the indoor air. It never did and never will destroy the impurities that constitute the danger element. Nature purifies her atmosphere with ozone. Man can purify it only with ozone.

The author then sums up the use of ozone as a deodorizer. Its properties are remarkable in this line and with higher concentrations almost any odor known, excepting the extraordinary perfumery from the pole-cat, can be successfully combatted. In an office, factory or home the chief odors to be eliminated come from tobacco smoke, cooking, laundry, sick rooms or any overcrowded spaces.

The author records it is as the accepted opinion of many physicians that colds and kindred diseases come from bad air, chiefly because bad air disturbs that part of the nervous system which regulates the blood circulation and local blood congestion ensues, bringing with it an interference with certain bodily functions which destroy the bacteria constantly taken into the system. Thus infection occurs, lowering resistance to disease, and we start with colds which are frequently the forerunner of tonsilitis, bronchitis, pneumonia and even tuberculosis.

Ozonized air acts as a preventative of conditions which bring about many of our colds, and it is on the preventative rather than on the curative side that it has a serious claim to consideration. There is just as much and no more magic in ozone for continuous breathing purposes as there is in pure air. Pure air will help the body to cure itself and that is all that ozonized air in a concentration proper for breathing will do.

Abroad ozone as an air purifying agent and as a supplement to ventilation is being extensively used in factories, theaters, mines, ships, schools, hospitals, hotels, restaurants, gymnasia, offices, cold-storage houses, libraries and homes.

The use of ozone in the purification of air is undoubtedly one of the most valuable of the newer achievements of electrochemistry. First, as a preventative of conditions which undermine health; second, as a promoter of working efficiency, which involves the third, an increaser of business output, so that commercial and physical welfare are materially aided by a little boxed-up lightning placed in one's office, factory or home.

Evolution of the Lead Storage Battery.

A paper by Mr. PEDRO G. SALOM, of Philadelphia, dealt with the evolution of the lead storage battery. The first step was the Plante cell, with its great number of alternate charges and discharges, requiring weeks or months. Then came the Faure-Brush type of cell as the second step; that is, the cell with pasted active material and subsequent formation. Since 1881 no improvement has been made over this type except in mechanical details.

The logical evolution was to go a step further, to produce the active material in bulk outside of the cell and apply it mechanically to the grid, whereby the plate is finished and no subsequent formation is required. This is the "initially formed cell" of Mr. Salom, which is now being made by the Niagara Lead and Battery Company, of Niagara Falls, N. Y.

In our September issue, page 549, Mr. Salom's patent for this new cell has already been mentioned. The negative plate is made as follows: A mass of spongy lead is produced by electrolytic deposition or electrolytic reduction, thoroughly washed with cold water, passed under stirring or rubbing through a sieve of 40 mesh and is then exposed to the atmosphere and stirred to permit evaporation of the water. While the evaporation proceeds, a thin superficial coating of suboxide of lead is formed throughout the interstices of the spongy mass. For the success of the process it is of importance that this is stopped at the proper moment: this is indicated by the "feel" of the damp mass, which when squeezed between the fingers should have the "temper" of molding sand: it then contains still moisture, about 5 per cent in bulk of the total mass. In this state the superficially suboxidized spongy lead is applied to the grid and compressed within the compartments by means of heavy pressure (1,500 lb. per square inch). The finished plate presents smooth faces, flush with the thickness of the frame and main bars.

The chief advantages claimed for this new process of Mr. Salom over present practice are that the new plate, while remarkably homogeneous, carries a greater quantity of spongy lead and that the cost of producing the spongy lead electrolytically in bulk and applying it mechanically to the grid is less than the cost of an equivalent amount of litharge as employed in present practice.

Mr. Salom compared at some length in his paper the relative advantages of the Plante cell, the Faure-Brush cell, and the Salom initially formed cell. Quite a lively discussion followed.

Mr. Sperry said that in the popular mind the lead battery was something exceedingly heavy. But when a comparison was made concerning the amount of weight required to store

a certain amount of power it was found that the electric storage battery (an old type of cell being assumed in this calculation) required 80 lb. of total weight per hp-hour and a compressed-air system 212 lb. of total weight per hp-hour.

Mr. Hepburn gave 62 lb. per hp-hour as the corresponding figure for the Salom cell.

Mr. Salom stated that he does not know yet what the life of his negative plate would be. But experts have judged that it should last as long as the negative of the Plante cell, whose life is ten years. The capacity of the negative plate per ounce of active material is 2 amp-hours; that is, about four times the theoretically necessary amount of active material is used. This is about the same proportion as in other batteries. But Mr. Salom emphasized that he puts about 25 per cent more active material into his plate.

Mr. Hering remarked that a battery should not be judged by the figure of weight per hp-hour alone. Life is very important. And the most economical life should be found in the same way as for incandescent lamps, from these two factors: First, the depreciation of the battery, and, second, the cost of a new battery. A battery of shorter life and lower price may be more economical than one of longer life and higher price.

Mr. Hepburn emphasized the excellent and perpetually good contact between active material and support, which should insure a practically infinite life to the negative plate of the Salom cell. Under abnormal conditions the battery stands very well, and this is especially true for the negative plate. One important point in storage battery maintenance is never to allow the temperature to rise to too high a degree.

Mr. Sperry asked how the cell compared with the Edison iron-nickel cell and Mr. Salom gave for the Edison cell a specific capacity of 15 watts per pound of total weight, including everything. In this respect the Edison battery has an advantage over the lead cell, for which Mr. Salom does not claim more than 10 watts per pound. But Mr. Salom said that the claims, sometimes made for the Edison battery, of 100 per cent greater watt-hour efficiency were not justified, as it was only 10 or 15 per cent greater.

Dr. Edward Gudeman emphasized that for certain purposes the available space was a consideration and that the figure of power per volume should be given besides the figure of the power per weight.

Self-Discharge of Batteries.

A paper on the self-discharge of Plante and Faure storage batteries by Prof. O. W. Brown and Mr. W. G. Bowers was then presented. Comparative tests were made with four different types of batteries: (1) Both plates pasted, (2) both plates Plante, (3) both plates Plante, but more deeply formed, (4) positive Plante and negative pasted plate. The results of the comparative tests are summed up as follows:

The Plante type of cell looses its charge when allowed to remain on open circuit for periods of ten to twenty days. The Brush-Faure cell, when made from pure materials, suffers only very slight loss in capacity when left on open circuit; and in every test the self discharge of the Brush-Faure cell was much less than that of cells of the Plante type.

The data also shows that the Plante cell suffers permanent injury when considerable periods of time are allowed to elapse between charges, while one or two recharges of a Brush-Faure cell restore it to its original capacity.

In the light of these results the Brush-Faure cell is best suited for all service where intervals of a week to a month or more must elapse between charges. This condition is found in some telephone plants, in isolated lighting plants and where cells are used for ignition purposes and for signal systems.

Overvoltage Phenomena.

Mr. Carl Hambuechen, of the Northern Chemical Engineering Laboratories, of Madison, Wis., presented a paper on the practical significance of overvoltage phenomena.

The overvoltage of a certain metal is that excess voltage which is required for the setting free of hydrogen gas at a cathode of the given metal, as compared in general with a hydrogen electrode (platinized platinum electrode covered and saturated with hydrogen gas). Under otherwise the same conditions a higher potential is required to set hydrogen free at some metallic plates than at others. Hydrogen shows a decided aversion to tin and lead and zinc, as shown by the higher voltage required to set hydrogen free on these metals.

Mott gives the following list of overvoltages of hydrogen on the various metals:

Tin	.55 volt
Lead	.50 "
Antimony	.45 "
Bismuth	.45 "
Gold	.20 "
Silver	.15 "
Platinum	.15 "
Iron	.10 "
Copper	Reference standard
Brass	.14 volt.

Neumann in his table of electrochemical potentitls of various elements making contact with electrolytes gives the following values of elements against normal chloride solutions:

Aluminium	+ 1.015 volt.
Zinc	+ .503 "
Cobalt	— .015 "
Nickel	— .020 "
Tin	— .085 "
Iron	— .087 "
Lead	— .095 "
Hydrogen	— .249 "
Bismuth	— .315 "
Copper	— .5+ "
Platinum	— 1.140 "
Gold	— 1.356 "

According to the generally accepted theory, the corrosion of metals is caused by galvanic couples due to impurities or electronegative elements in the metal under consideration, and the rapidity of corrosion will then be dependent principally upon two factors, the resistance of the local circuits and the electrochemical voltage. If the latter were the only factor governing the motive force which sets up the current we could use the Neumann table of single potentials as a guide to tell just how bad certain combinations of metal are from the corrosion standpoint. Thus we would know that zinc containing particles of copper would be subject to corrosion couples of 1.003 volts, zinc with iron .59 volt, zinc with tin .588 volt, etc.

However, there is another important factor, namely, the overvoltage values; the active galvanic couples always give rise to the liberation of hydrogen on the electronegative or cathodic particles and therefore the overvoltage values must be considered.

Thus, a couple of zinc and copper would have the value approximately as above given, 1.003 volts, because hydrogen on copper has very slight overvoltage; zinc with iron would have, not the value .59 volt, but .1 volt less, or .49 volt effective pressure; and zinc and tin would have .558 less .55 volt (the overvoltage of tin) or .038 volt.

While the electrochemical potential table would indicate that tin is as bad an element as is iron for producing corrosion on zinc, the correction for overvoltage shows that tin is far less harmful. And this is borne out by the facts in various practical instances. It is a matter of practical knowledge that iron or copper introduced as an impurity in dry cells is very harmful, producing rapid deterioration, while lead and tin (having high overvoltages) are almost harmless. This is fortunate, as otherwise the solder for making the seams

would have to be prevented absolutely from entering the container and making contact with the electrolyte. The dry cell construction would be therefore more difficult and expensive.

A well-known household method of cleaning silverware of tarnish consists in placing the silverware in a zinc vessel containing an electrolyte such as baking soda and salt. The tarnish is removed by electrochemical reduction—the zinc acting as anode and the silver surface as cathode. In studying this process the writer noticed that a zinc vessel having been used thus a few times lost its effectiveness due to the formation of a high-resistance film on the zinc surface which prevents the silverware from making electrical contact therewith when resting upon it.

It was found that a grating of tinned iron wire soldered to the zinc entirely prevented this difficulty. The tin acting as the cathode surface is always maintained bright and chemically clean, and due to the overvoltage of tin this is kept clean at the expense of a minimum amount of current. Then, when the silverware is placed upon such tinned grating, it cannot avoid being placed in electrical contact, and the current generated by the dissolving of the zinc is concentrated upon the silver to the practical exclusion of the tin.

In the making of electrical contacts the usefulness of tin is well known, and this is dependent largely upon its high overvoltage.

Zinc is employed extensively as a means of preventing boiler corrosion. It is an old remedy and is effective when properly employed. The principal difficulty in the use of zinc is that it must be attached firmly to the iron or other metal structure which it is designed to protect so as to insure electrical contact. The protection is effected at the expense of the zinc, which is corroded, and which must be replaced from time to time. It is evident that in the interior of a boiler it is difficult to introduce zinc, and it appears that the overvoltage of tin may be called into play in this connection.

As an example of the various methods in which this might be employed, the following is mentioned. Through the boiler which is to be protected is passed an iron rod or tube coated with a layer of tin. This rod is bolted to the boiler shell in such a way as to make electrical contact. This tinned tube forms with the iron of the boiler a galvanic couple which would be detrimental to the boiler if it were not for the overvoltage which is set up by hydrogen on the tin. From this tinned rod may be suspended plates or chunks of zinc by means of a tinned stirrup. The exposed tin always acting as a cathode will be kept clean and therefore be capable of making electrical contact. A convenient form in which to employ the zinc is in rings of zinc coated on the inner surface with tin or solder and slipped over the tinned rod. This can be readily provided for by stuffing boxes or hand holes through which the tinned rod is passed. The worn-out zinc pieces can be thus removed and new zinc introduced.

Mr. Mott has already pointed out the desirability of studying overvoltages of the various metals in connection with the investigation of corrosion phenomena; that an alloy consisting of two elements widely separated in the electrochemical series may not necessarily be a corrodible alloy if the negative element possesses a high overvoltage.

Mr. Hambuechen concludes that in studying corrosion we are concerned not only with the contact potential and the overvoltage of the individual elements constituting an alloy, but the study must be extended to include the single potential and overvoltage of the alloying compounds. Thus, for example, it has been found that brass has an extremely low overvoltage, lower than either copper or the zinc of which it is composed. This is undoubtedly due to the fact that the brass is composed of one or more alloying compounds.

Metallic Food Containers.

The first paper of the Saturday morning session was presented by Dr. EDWARD GUDEMAN, of Chicago, on the corrosion of metallic food containers.

The powers in food chemistry have come to the viewpoint of considering a quantity of 0.001 per cent as a large quantity. Food products are sometimes condemned for very small quantities of impurities.

This is of importance in connection with the use of tinplate for food containers. Until recently there has been no objection to tin. But now the Department of Agriculture requires that food must not contain more than 300 milligrams of tin per kilogram. This is the permissible maximum.

When food ages it undergoes decomposition. Sulphuretted hydrogen and carbonic acid are set free. Carbonic acid attacks, for instance, lead rapidly. There may be, in general, oxidation, reduction, action of acids, and action of alkalies.

The solubility of tin is small. But a complete uniform covering of an iron plate with tin without pores is impossible. Dr. Gudeman has made a series of experiments to produce some continuous coverings which will not be corroded and will not be attacked by the food product.

He first experimented with coatings of gold on iron. These can be made continuous, and the gold film may be made so thin that the cost of the gold itself is not prohibitive. The iron plate is treated with an amalgam of gold and mercury and the mercury is distilled off. The process is workable, but 98 lb. of mercury are to be distilled off for 2 lb. of gold. This indicates the inherent difficulties of the process.

He then tried to enamel iron plates. But the enamel should not be applied simply to the outside of the iron plate, as it might crack off; the enamel must be made an integral part of the iron plate. Dr. Gudeman forced enamel into iron by high pressure, some 5000 to 8000 lb. per square inch. The excess of enamel was taken off and the plate heated to fuse the enamel to the iron. Enameled iron plates, prepared in this way, have in fact withstood the action of boiling sulphuric acid and of boiling alkali better than Jena glass.

The question then presented itself to Dr. Gudeman: if real enamel can be fused onto iron plate, why not impregnate iron plate with glass? This can be done. Such a glassed iron plate withstands the dissolving action of food as well as glass. But a fusible glass is needed. This means lead glass. But lead is not wanted in the food. Thus all difficulties have not yet been solved. Dr. Gudeman is continuing his researches.

The Electric Furnace in Metallurgy.

A paper by Mr. MARCUS RUTHENBURG, of London, England, was then presented on "The Electric Furnace and Its Applications." He briefly traced its history from the first electric furnace built by Siemens thirty-five years ago.

Direct current is required only if an electrolytic action is to be performed in the furnace, as in the manufacture of aluminium. Alternating current is preferable for all electric furnaces where only a heating effect is to be produced.

While most large electric furnaces are operated at constant voltage, it is advisable to consider constant-current operation.

There is a considerable waste of electrodes and this represents a great expense. This waste of electrodes is not all at the ends but also at the sides. Water-jacketing the electrodes is very useful.

To use slag as a conductor of heat is also recommended to save the electrodes.

Mr. Ruthenburg then reviewed various applications of the electric furnaces in metallurgy, especially for the treatment of:

Nickeliferous iron ores;

Complex lead-zinc-iron ores, in the smelting of which lead and zinc are volatilized and condensed outside;

Tin ores;

Ores of tungsten, manganese, etc.

In conclusion, Mr. Ruthenburg referred to the importance of cheap power supply. He uses himself current from a steam plant which is charged at 0.22 cent per kw-hour.

In the brief discussion which followed Mr. F. T. Snyder spoke on the possibility of central stations supplying electrical

energy to electric furnace plants at a reduced rate at times of light load. He questioned the practicability of such schemes, as the usual electric furnace was not generally a useful load for the central station.

Electrostatic Ore Concentration.

Mr F. S. MacGregor presented a paper on "Applications of the Electrostatic Separation to Ore Dressing," with special reference to the new system of the Huff Electrostatic Separator Company, of Boston, Mass. (see also page 295 of our May issue). The commercial application of the art of electrostatic separation has been tried a number of times within the last decade, but until only within about three years has success attended the efforts.

Two factors contributed to this want of success—a lack of the proper means of continuous electrification, and a practical machine and system of treatment designed to meet the rigorous requirements of mill operation. Recognizing these conditions, the Huff Electrostatic Separator Company, of Boston, has perfected several types of separators and a means of electrification which have been a commercial success from their first installation, and at present the entire concentrating and separating machinery of two plants consists of Huff separators—one located at Platteville, Wis., owned by the American Zinc, Lead & Smelting Company, and the other at Midvale, Utah, owned by the United States Smelting, Refining & Mining Company. Also a plant is being installed in Nevada.

The type of machine is shown in Fig. 3 and consists of a series of separating electrodes placed one above the other. The frame is cast iron, built in sections, with two electrodes to each section, which facilitates easy shipment and also flexibility in the number of treatments which can be given the ore, and which is dependent on its requirements. A three-section machine (the usual type) is 6 ft. high, 6 ft. 6 in. long and 18 in. wide. A feed hopper on the top distributes the ore across the first electrode, and it passes by gravity to each succeeding one. The mineral particles receive their charge while passing through a concentrated static field formed by two electrodes—one a grounded rotated shaft and the other an insulated metal rod.

The electrostatic charging of the electrodes is accomplished by a special electrical apparatus (using commercial dynamos and transformers), which, with no more than the usual oiling, gives either a continuous and steady potential of several thousand volts, or an intermittent or pulsating charge of great regularity. The potential, or strength of the field, can be altered at will, according to the conductivities of various ores

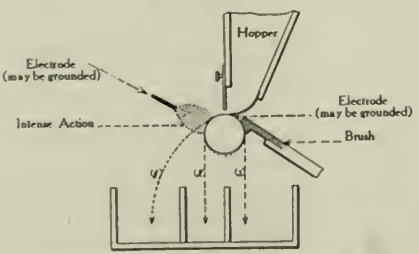

FIG. 2.—PRINCIPLE OF ELECTROSTATIC SEPARATION.

treated. Fig. 2 shows the scheme of separation, the dotted lines indicating *in general* the path of the conductors and non-conductors.

The separating plant at Platteville was started in March, 1908, and was the first plant to be equipped with Huff separators. Although originally built as an experimental plant, it has been in practically continuous operation since then, and used to separate, without roasting, the iron pyrite from the zinc blende, which occur together in varying proportions in southwestern Wisconsin. The ores are milled in ordinary jigs and the lead and lime rock removed. Owing to the nearness in specific gravity of the blende and pyrite, they are recovered in the jigs as a middlings, containing from 10 to 45 per cent zinc, according to the grade of the crude ore, the remainmg percentage being iron and the sulphur in combination. Iron above a few per cent interferes with the smelting of the blende, and the function of this plant is to remove this pyrite, making the low-grade ores into high-grade smelting ores.

The plant consists of a wooden, frame building, 9 x 12 m (30 x 40 ft.), and three stories high. On the first floor are

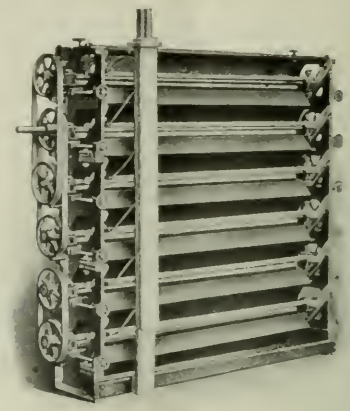

FIG. 3.—ELECTROSTATIC SEPARATOR.

the bins for collecting the finished products as they are delivered from the separators. The second and third floors are separator floors, and in the tower is located the screening system. A spur railroad track runs on either side of the mill, one for cars of raw ore, allowing them to be unloaded either directly into the mill or into storage bins across the track. The other track, which is below the mill level, permits at the same time the loading of empty cars with finished products.

The course of the ore through the plant is as follows. It is shoveled into a wheelbarrow in the freight car, wheeled to scales and weighed, then dumped into the feed hopper of the dryer. The top of this hopper is on a level with the car floors to facilitate easy handling of the ore, and at the bottom of this hopper is an adjustable plunger feeder. The dryer is of the cylindrical type, and the shell is 5 m (16 ft.) long and 0.75 m (30 in.) in diameter, rotating about 4½ r.p.m. Being set at a slight pitch, the ore travels gradually to the discharge end, where it emerges bone-dry. As the separation depends on the difference in conductivity of the minerals it is necessary that all moisture be removed.

A short drag conveyor carries the dried ore to the foot of a bucket elevator, which raises it to the top of the tower and discharges it into a set of screens. There are four, giving an oversize and four other sizes, called A, B, C and D. In general the sizes are as follows:

A size	6 x 3 mesh.	(7 x 8 mm)
B "	10 x 7 "	(4 x 6 ")
C "	24 x 20 "	(0.16 x 0.20 ")
D "	50 x 50 "	(0.08 x 0.08 ")

The oversize passes through a short trommel to remove any foreign materials, and is spouted to the first floor to a set of rolls, which crush it and deliver it to the boot of the elevator carrying the original feed.

The location of the screens in a tower permits the different sizes to flow by gravity through iron pipe 5 cm (2 in.) diameter to the top machine floor. On this floor are located

the rougher machines, where the ore is given a rough split, that is, two products are made, one a product higher in zinc than the original feed, and the other higher in iron. Thus, no matter what the grade of the feed to the mill may be (and it varies within wide limits), a fairly uniform mixture is sent to the finishing machines on the floor below.

These finishing machines give the final cleaning to the ore, one set giving a finished iron product, and the other a finished zinc product. The middlings from these cleaners return by gravity to the boot of the elevator and mix with the fresh ore, so that no ore leaves the system that is not separated into its proper grades of product.

The finished zinc product and finished iron product fall into separate bins on the first floor. These bins are V-bottomed, and constructed of two thicknesses of wood flooring with heavy building paper between. They are elevated from the floor so that they may conveniently be drawn off into a 730-kg (1600-lb.) capacity push car for loading.

The car is then wheeled to a small elevator and dumped into a hopper on the floor level. This feeds the ore into the boot of a small elevator, which discharges either into a freight car by means of a goose-neck spout, or may be allowed to discharge to a second elevator in the storage bins. From these bins a car may be loaded whenever convenient.

The plant generates its own power, and the engine and boiler rooms are located in an adjoining building. The fire-boxes of the boiler and dryer are side by side, enabling the engineer to look after both fires as well as the engine room. Power is supplied by a Corliss engine connected to the mill by a clutch.

The plant operates continuously three shifts per day. The labor required per 24 hours is as follows:

3 shift bosses	(8 hours each)
3 machine helpers	(8 " ")
3 dryer feeders	(8 " ")
2 loaders	(10 " ")
2 engineers	(13 and 11 hours each)

There are also a superintendent, master mechanic, assayer, one buyer and bookkeeper, although the buyer and, to a large extent, the assayer devote their time to obtaining custom work.

Two products of marketable value are obtained: The iron pyrite, hitherto valueless in these Wisconsin ores, is sold for its sulphur content, and by reason of its free-burning qualities, makes a very desirable product for the acid manufacturer; and a zinc product which is, of course, sold for spelter. The pyrite is sold by contract, but the blende is sold to the smelters on the open market. With a penalty for excess of iron or lead, one may see it is necessary for such a plant to turn out consistently high-grade products in order to command a satisfactory market price.

The results taken from the "Record of Shipments" for a month show an average zinc product of 55.2 per cent Zn and 3.4 per cent Fe, and a pyrite product of 3.7 per cent Zn and 44.7 per cent S. The average zinc content of the ore, bought and separated into the above products, was 24.1 per cent Zn, and the average iron content was 25.4 per cent Fe. The minimum zinc content was 6.7 per cent Zn, and the maximum 41.2 per cent Zn. The high grade of the separated products shows the flexibility of the process and the ability of the plant to handle any grade of ore from the district.

As a result of the excellent showing of this electrostatic plant the United States Smelting, Mining & Refining Company decided to put in a plant at their works at Midvale, Utah. The function of an electrostatic plant there is different from that in Platteville, for the ore from the mine of the United States company at Bingham, Utah, contains about 9 per cent of zinc in addition to the gold, silver and lead values.

This ore is shipped to the concentrator at Midvale, where it is crushed and milled by the usual methods of jigs and tables. In this concentration there were formerly two products, one a lead-pyrite product carrying the iron and considerable zinc, and a tailings carrying zinc and some gold and silver values, the object being to keep the zinc as low as possible in the lead concentrate, and yet at the same time make the maximum recovery of the gold, silver, lead and copper values. The zinc added materially to the difficulty and cost of smelting the lead concentrates, and, at the same time, was a total loss. In order to recover this zinc three products were made of the concentrator, a lead-pyrite carrying practically no zinc, tailings free from zinc and middlings containing about 22 per cent zinc, the balance pyrite and some copper, lead, gold and silver.

To treat these middlings an electrostatic plant was built near the concentrator, as shown. It is a wooden frame building 12 m x 12 m (40 ft. x 40 ft.), and covered with corrugated sheet iron. In general plan it is the same as the Platteville plant—finished ore bins on the ground floor and two floors of separators.

The method used for handling the ore is somewhat different. The middlings are shoveled from the table hutches in the wet concentrator into tramcars, which are pushed by hand to the platform elevator at the electrostatic plant. The cars are raised to the third floor level, and, after weighing, are dumped into a 40-ton hopper. This is built over the dryer and feeds automatically. After drying the ore is screened as follows: Making A size 12 on 24, B size 24 on 50, C size through 50 mesh.

The oversize is small in amount and very low in zinc, so, instead of re-crushing, it is put directly into the finished pyrite. The three sizes, A, B and C, are separated, and the products drawn from the bins on the first floor into tramcars. These are run out of the mill to the loading trestle and dumped into freight cars. The pyrite product is smelted in the United States smelter nearby, and the zinc product is shipped to the Kansas zinc smelters.

The plant is electric driven by a 20-hp motor, the 15 Huff separators requiring about ½ hp each. The generating apparatus for the electrostatic field is located in a small room on the finishing floor. A 3-hp motor is direct coupled to a generator, and the strength of field is controlled from a small switchboard.

The plant is being operated in conjunction with the concentrator, and the superintendence, assaying and bookkeeping are taken care of by the smelter staff.

The average assays of the feed and products are as follows:

	Au.	Ag.	Cu.	Pb.	SiO_2	Fe.	Zn.	CaC
	Oz. per Ton.							
Crude ore to concentrator	0.08	3.8	0.41	8.4	28.8	14.3	9.0	6.0
Middlings to electrostatic plant	0.05	2.8	1.11	3.3	4.6	24.3	21.6	1.9
Finished pyrite	0.14	4.8	2.24	6.6	..	26.0	9.0	..
Finished blende	0.02	1.5	0.61	2.0	..	3.6	52.9	..

This electrostatic plant not only removes the objectionable zinc from the lead-pyrite concentrate, but makes of it a marketable product, and, instead of being a total loss, it is made to produce a profit. The electrostatic process in these two plants demonstrates two applications in ore dressing, one the custom plant making low-grade ores profitable and the other as an adjunct to a wet concentrator for the recovery of zinc.

Besides these plants, there is being installed in Nevada a plant working on crude silver ores, wherein the ore goes through no concentrating machinery prior to its electrostatic treatment. The process has been proved applicable to many special problems, such as the concentration of copper and lead-silver ores in heavy gangues, such as garnet, braite, epidote, etc.

The table on page 638 will give some idea of what can be expected with several of these problems.

The perfection of the art of electrostatic separation marks an important step in the dressing of complex or difficultly milled ores.

In the brief discussion which followed Mr. MacGregor explained that the capacity of the electrostatic separator depends on the size of the ore and the character of the ore. At Platteville 15 machines treat 95 tons a day.

TABLE OF ELECTROSTATIC SEPARATIONS

Material	Product	% Cu.	% Fe.	% Pb.	% Zn.	% SiO_2	Oz. Au.	Oz. Ag
Low Grade Chalcopyrite in Quartz	Original	2 56	17 8	. .	. .	. .	0.01	2 6
	Concentrates	5 63	37 0	. .	. .	. .	0 02	5 5
	Tails	0 10	2 0	. .	. .	. .	Trace	0 4
Pyrite and Chalcopyrite in Various Gangues	Original	6 37	25 0	. .	. .	. .	. .	. .
	Concentrates	9 33	36 3	. .	. .	. .	. .	. .
	Tails	0 14	1 9	. .	. .	. .	. .	. .
Chalcopyrite and Bornite in Garnet	Original	3 60	18 6	. .	. .	32 3	(Iron partly	
	Concentrates	19 10	25 7	. .	. .	14 4	in Sulphide	
	Tails	0 34	17 6	. .	. .	36.7	and partly in Garnet.)	
Zinc, Iron, and Silver Ore	Original	2 61	23 8	13 9	19 4	. .	. .	52 8
	Concentrates	3 33	30 7	16.4	6 4	. .	. .	69.0
	Tails	0 91	3.4	2 7	51.8	. .	. .	4 6
Zinc, Lead, and Iron Middlings	Original	. .	20 2	12 6	23 1	5 2	. .	. .
	Concentrates	. .	31 2	21.0	4 0	3.2	. .	. .
	Tails	. .	2.4	0.5	53 8	9 5	. .	. .

Material	Product	% Cu.	% Fe.	% Pb.	% Zn.	% S.	% SiO_2
New Mexico Concentrates	Original	2.11	16.2	. .	29.9	. .	12.9
	Concentrates	5.65	37.2	. .	4.9	. .	6.6
	Tails	0.13	4.2	. .	44.9	. .	15.9
Zinc from Above Tailings	Original	. .	. .	. .	. .	. .	. .
	Concentrates	. .	. .	. .	57.1	. .	. .
	Tails	. .	. .	. .	1.3	. .	. .
Native Copper in Sandstone	Original	2.68	. .	. .	. .	. .	. .
	Concentrates	38.70	. .	. .	. .	. .	. .
	Tails	0.27	. .	. .	. .	. .	. .
Pyrite Concentration for Sulphur	Original	. .	. .	. .	. .	30 1	. .
	Concentrates	. .	. .	. .	. .	44.8	. .
	Tails	. .	. .	. .	. .	3.0	. .
Middlings from Joplin, Mo.	Original	. .	10.8	2 5	49.5	. .	. .
	Concentrates	. .	37.4	9 9	2.9	. .	. .
	Tails	. .	1.6	Trace	62.3	. .	. .

Isolation of Ions.

Prof. Robert A. Millikan, of the Ryerson Physical Laboratory of the University of Chicago, presented a paper on the isolation of an ion and a precision measurement of its charge. The principle of the method is to hold a given small droplet of oil (or other non-volatile substance) under observation for a considerable time, which may be four or five hours, and to watch its movement both under the action of gravity alone and under the combined action of gravity and an electrostatic field, and to watch especially the changes of this movement which occur when the droplet catches ions from the air.

Oil is sprayed through an atomizer so that it can be obtained in minute droplets, and one of these droplets is admitted into the air space between the horizontal plates of an air condenser charged from a storage battery to any voltage up to 12,000 volts.

The oil droplet is so small that it almost floats in the air. It descends very slowly under the action of gravity. The motion of the droplet is observed through a telescope and its speed carefully measured. This is first done while the condenser is not charged, so that the electrostatic field is zero and gravity acts alone. The speed is also measured while the condenser is charged so that both the electrostatic field and gravity act on the oil droplet. From both measurements the charge carried by the oil drop can be found.

Now while the oil droplet is under observation, it is liable at any moment to catch one of the ions which normally exist in air or which have been produced in the air between the plates by some ionizing agent. In that moment the speed under the influence of the electrostatic field changes immediately, while the speed under action of gravity alone remains constant. From this change of speed the charge of the new ion which has been captured by the oil droplet is determined.

For the calculation of the charge only very simple mathematics is required and no assumptions whatever are necessary except that Stokes' law holds good. This is found to be correct as long as the oil droplet is not too small. For sizes of the droplet below certain limits it must be modified, but it is remarkable that the modification which was found to be necessary by Professor Millikan from his experimental results agrees exactly with a curve formerly derived and based on a purely theoretical argument.

The chief results which follow from the very extended series of Professor Millikan's determinations are as follows:

The charge of any ion, whether positive or negative, which is captured at any time by the oil droplet is always a multiple of 4.9×10^{-10} c.g.s. units. Further the charge carried originally by the oil droplet is also always a multiple of the same quantity of electricity.

In other words, we have something entirely analogous to the chemical combination of elements according to definite multiple proportions. The conclusion is that electricity is atomic or ionic, or granular, or corpuscular in structure. All charges are built up of elementary unit charges or ionic charges. The elementary or ionic charge is 4.9×10^{-10} c.g.s. units. The measurement of the ionic charge by this method is directly comparable with atomic weight determination.

In the brief discussion which followed all the speakers complimented Professor Millikan highly on the fundamental character of his work and the astonishing accuracy obtained. Dr. Schlundt referred to the doubtful results recently obtained by Ehrenhaft with a somewhat similar method and asked whether his troubles could be attributed to his using too small droplets. This was confirmed by Professor Millikan, who said that Ehrenhaft had worked with particles 10 or more times smaller than his (Millikan's) droplets and that for such small sizes Stokes' law ceases to hold at all. The apparatus used by Professor Millikan was afterward shown and demonstrated by him in the Ryerson Laboratory of the University of Chicago.

Crystallization Through Membranes.

A paper by Dr. J. H. Walton, Jr., of the University of Wisconsin, dealt with the phenomena of crystallization through membranes. It was presented in abstract by Dr. Kahlenberg. Two parts of a vessel were filled with the same supersaturated (under-cooled) solution and separated from each other by a membrane. By dropping a small crystal into the solution on one side, crystallization was started on this side. The question was whether the crystallization would pass through the membrane to the other side. This was the case with some solutions and some membranes, but not with others. The experiments required utmost care. There seems a connection with adsorption phenomena and the whole matter is of great importance with respect to semipermeable diaphragms which have played such a big rôle in the development of Van't Hoff's theory of solutions.

Relative Basicity of Metals.

Dr. Louis Kahlenberg, of the University of Wisconsin, then presented his paper on the relative basicity of metals as shown by their power to replace one another in chemical compounds.

In the past all the work has been done with the aqueous solution and a series of the metals has been determined according to their potentials in aqueous solutions or their ability to replace one another in aqueous solution. The object of Dr. Kahlenberg's research was to determine how the situation is in solutions where there is no water and whether metals can replace one another in insulating liquids.

Dr. Kahlenberg experimented with some 60 different solvents and found that the series of the metals had an entirely different order in non-aqueous solutions from that in aqueous solutions. In many cases a tremendous power of lead to throw out copper was apparent. Thus copper is thrown out by lead from a copper oleate solution. The series of the metals in carbon bisulphide solutions is as follows: Pb, Zn, Bi, Mg, Co, Sn, Fe, Al, Ag, Sb, Ni, Hg. This series is considerably different from the series for aqueous solutions.

It is also a fact that in these solutions which are practically non-conductors reactions are quite as instantaneous as in aqueous solutions. It is, therefore, a fact that the replacement of one metal by another does not depend on the electric conduc-

tivity of the solution. In some cases the solutions had to have a definite concentration if the throwing out of the metal was to be complete.

In the discussion which followed, Dr. Richards pointed out that in aqueous solutions the series followed in general the order of the heats of formation. Dr. Kahlenberg said that he did not yet know whether the same was true for non-aqueous solutions. The thermochemistry of non-aqueous solutions is now being taken up systematically at the University of Wisconsin. The above observations may offer possibilities for practical use to throw out strong metals by weak ones.

Water in Chemical Reactions.

A paper by Dr. DAVID KLEIN, of the University of Wisconsin, dealt with the effect of water in causing chemical reactions. It was read in abstract by Dr. Kahlenberg.

It has been claimed that for many chemical reactions to go on the presence of water is absolutely necessary, and that in this respect water is unique. For instance, hydrogen sulphide and sulphur dioxide will not react when no water is present. Dr. Klein made experiments with 25 other substances than water and found that 15 of them were as effective as water. The Nernst-Thomson rule does not hold. There is chemical activity in electric insulators. "Wasser ist kein besonderer Saft."

These last three papers form part of the extended research work which is being carried on at the University of Wisconsin under direction of Professor Kahlenberg.

After a hearty vote of thanks to all those in Chicago who had contributed to the great success of the first Western convention of the Society, the meeting adjourned. The next meeting will be held in spring, 1911, in New York City, when the new magnificent building of the Chemists' Club will be ready.

In the following we give a complete alphabetical list of the names of members and guests in the last printed registry list. Where no city is stated, Chicago is meant.

T. B. Allen, Niagara Falls, N. Y.; R. L. Ardrey, W. H. Arison, Niagara Falls; Mr. C. B. Askew, Chas. F. Baker, Wm. C. Bauer, Evanston, Ill.; Mr. and Mrs. A. Bement; P. McN. Bennie, Niagara, Falls; M. Bentley; E. F. Berg, Urbana; Jesse Johnson; Bernhard Wm. Bjorkstedt, Gary, Ind.; David J. Block; F. H. Bosworth; Dr. and Mrs. Wm. Brady; Ivan M. Bregowsky; Morgan Brooks, Urbana, Ill.; John W. Brown, Cleveland, Ohio; A. F. Brunck; T. J. Bryan, Oak Park, Ill.; C. F. Burgess, Madison, Wis.; Wm. W. Burson; J. A. Cameron; T. M. Caven; E. S. Clarage; E. L. Clark, Cleveland, Ohio; Jesse Coates, Coatesville, Pa.; W. W. Coghill, Evanston, Ill.; H. B. Coho, New York; F. L. Collins; R. T. Conger; H. R. Connell, Pittsburgh, Pa.; Wm. A. Converse; G. W. Conway; T. M. Craven; J. S. Crider, Cleveland, Ohio; R. A. Crider, Cleveland, Ohio; C. W. Danforth, Gary, Ind.; G. M. Davidson; Mr. F. M. de Beers; J. C. Dickerman; H. E. Diller, Schenectady, N. Y.; W. W. Dingee Ervin Dryer; J. A. Ede, La Salle, Ill.; H. Ehrlich; H. H. Engle, Cleveland, O.; W. F. Faragher; S. H. Fleming, Cleveland, Ohio; Geo. E. Folk; W. S. Ford; F. F. Fowle; K. G. Frank, New York City; F. C. Frary, Minneapolis, Minn.; A. E. Gibbs, Wyandotte, Mich.; Miss M. E. Gibbs, Wyandotte, Mich.; J. D. Goodwin, Fremont, Ohio; E. Grant; W. H. Graves; Albert E. Greene; J. B. Greene; W. B. Greenlee; Edw. Gudeman; Carl Hambuerhen, Madison, Wis.; M. E. Hanna; M. H. Hanks; G. W. Heise; H. A. Heisc; Miss Alice Henderson, Green Bay, Wis.; D. M. Hepburn, Niagara Falls, N. Y.; Carl Hering, Philadelphia, Pa.; Paul Heroult, N. Y. City; W. T. Holmes, Baltimore, Md.; R. Hornbeck, Livingston, Mont.; Mr. and Mrs. Wm. Hoskins; W. C. Hovey; L. E. Howard; Mr. and Mrs. Alois von Isokovics, Monticello, N. Y.; E. Jalbert; Mrs. J. A. Jackson; G. G. Jenista; A. R. Johnson; Grinnell Jones, Urbana, Ill.; G. H. Jones; L. Kahlenberg, Madison, Wis.; Chas. C. Kawin; W. E. Keily; A. Keller, Milwaukee, Wis.; W. M. Kelsey, De Pue, Ill.; Philo Kemery, Pittsburgh, Pa.; E. T. King, Cleveland, Ohio; W. W. Kinsley, Jr.; Geo. O. Knapp; Otto Kney; M. M. Kohn, N. Y. City; Wm. Koula; O. L. Kowalke, Madison, Wis.; G. Lawson; W. Leeds; F. A. Lidbury, Niagara Falls; C. B. Libme, La Salle, Ill.; O. Linder; M. G. Lloyd; J. A. Lounsburg; F. H. Loveridge; A. Lowenstein; Ernest Lunn; Mr. and Mrs. James Lyman; W. P. Marseilles, Niagara Falls, N. Y.; Mr. and Mrs. A. B. Marvin; Mr. and Mrs. H. T. Matthews; G. M. Mayer; Miss Eleanor Mayher, Easthampton, Mass.; F. S. MacGregor, Boston, Mass.; Harry McCormack; Dr. and Mrs. H. N. McCoy; J. E. McCullough, Maywood, Ill.; P. C. Messinger; J. M. Morehead; J. M. Muir, N. Y. City; P. J. Myall; F. J. Newman; F. J. O'Byrne; O. A. Olson; Chas. G. Osborne; Geo. S. Page, Pittsburgh, Pa.; H. Pagenstechre; J. H. Parker, Reading, Pa.; H. F. Pfaff; A. C. Parsons, Buffalo, N. Y.; C. R. Pierce; W. B. Pietz, Cleveland O.; H. F. Plaff; B. G. Poucher; H. C. Quigley; Albert A. Radtke; A. G. Reeve, Niagara Falls, N. Y.; J. B. Replogle; Edward T. Rice, Schenectady, N. Y.; Jos W. Richards, South Bethlehem, Pa.; Dr. and Mrs. W. D. Richardson; E. F. Roeber, New York City; O. E. Ruhoff, Madison, Wis.; F. C. Ryan, Hammond, Ind.; P. G. Salom, Philadelphia, Pa.; L. E. Saunders, Niagara Falls; M. Scharitz; C. J. Schwarz; R. Schwarz, Rochester, N. Y.; W. V. Schenck; Herman Schlundt, Columbia, Mo.; W. B. Schulte, Madison, Wis.; J. A. Seede, Schenectady, N. Y.; R. D. Small; E. S. Smith, Niagara Falls, N. Y.; W. R. Smith; Edwin H. Smythe; Mr. and Mrs. F. T. Snyder; J. L. K. Snyder, Cleveland, O.; M. A. Spencer, Jackson, Mich.; E. A. Sperry, N. Y. City; E. C. Sprague, Niagara Falls; S. E. Stafford, Pittshurgh, Pa.; P. Mc. M. Stanton, N. Y. City; W. M. Stein, Philadelphia, Pa.; Geo. W. Sternbergh; Leland L. Summers; J. A. Sutherland; C. J. Swartz; Mr. and Mrs. E. R. Taylor, Penn Yan, N. Y.; E. C. Taylor, Penn Yan, N. Y.; Miss E. B. Taylor, Penn Yan, N. Y.; B. F. Thomas, Chattanooga, Tenn.; C. B. Thwing, Philadelphia, Pa.; J. F. Towers, Gary, Ind.; D. C. Ulray, De Pue, Ill.; Paul C. Van Zandt; Chas. E. Veat; F. K. Vial; C. H. Viol; A. von Babo; Max von Vopelins, Berlin, Germany; L. D. Vorce, Detroit, Mich.; H. J. Wagner; E. W. H. Waldo, Urbana, Ill.; Wm. H. Walker, Boston, Mass.; J. H. Warder; J. R. Warner; O. P. Watts, Madison, Wis.; F. V. Wedlock; Harold K. Weld; Fred B. Wheeler; C. S. Wickes, Merchantville; J. R. Wilson, Niagara Falls, N. Y.; Wm. L. Wilson; Edson R. Wolcott; Edward Wray; H. W. Young; W. V. Young; J. A. Yunck, South Orange, N. J.; E. Zaremba, Buffalo, N. Y.; J. G. Zimmerman, Milwaukee, Wis.

Notes on Chemistry and Metallurgy in Great Britain.

(From Our London Correspondent.)

The British Association.

At the recent meeting of the British Association at Sheffield Mr. J. E. STEAD, president of the chemical section, delivered a noteworthy address dealing mainly with the effect of sulphur and silicon on the state of the carbon in cast iron. He said there was experimental proof that when iron sulphide is present the carbide of iron crystallizes with a very small proportion of sulphur—not more than about one-thousandth part of the weight of the carbide. The exact nature of the iron-carbon-sulphur compound had not been ascertained, but it appeared almost certain that the sulphur rendered the carbide stable.

The evidence supported the view that during the solidification of iron-carbon hypo-eutectic alloys, after the crystallization of the primary austenite and also in the eutectic and hypo-eutectic alloys, the carbide is first formed and subsequently decomposes into graphite and austenite. It had been proved by chemical research that the freezing of the hypo-eutectic alloys low in silicon results in nearly all the silicon crystallizing out with the primary austenite, and it followed that on gradually increasing the carbon and thus reducing the quantity of primary austenite—silicon being constant—the austenite formed would be gradually enriched in silicon until saturated, when the excess of silicon united with part of the carbide forming carbo-silicide of iron. The same result followed on gradually increasing the silicon if other elements were constant. In alloys of eutectic proportions, and in hypo-eutectic alloys, as no primary austenite could form, the silicon crystallized primarily with the carbide.

In Cleveland pig containing about 1.5 per cent phosphorus a ternary iron-carbon-phosphorus eutectic took the place of the iron-iron-carbide eutectic. In white irons with 3 per cent carbon and less than 2 per cent silicon, after the formation of primary austenite carrying nearly all the silicon, the iron-iron-carbide did not then form, but plates of cementite, or iron carbide; the metal remaining fluid had now reached the composition of the tenary iron-carbon-phosphorus eutectic which solidified at 940° C. In Cleveland irons, which become gray on cooling and which do not contain any primary austenite, the same iron-carbon-phosphorus eutectic is the only one formed during cooling, and instead of an iron-carbon-silicon eutectic two distinct cementites crystallized—one a silico-carbide and the other carbide of iron with possibly a little silicide in solid solution.

Micro-examination of cold alloys to which a small proportion of sulphur had been added during fusion went to show that the carbo-silico-cementite crystallized primarily. The primary carbo-silicides were very unstable and were first to decompose, yielding graphite and silico-austenite. When no appreciable quantity of phosphorus was present, two cementites formed, one a silico-carbide and the other the carbide, and the two crystallized together as a eutectic mixture. The precise composition of these cementites had not at present been ascertained.

The grayness of commercial metal with high silicon and low sulphur resulted from the very unstable nature of the silico-carbides. When silicide of iron was heated at 1000° C. with pure white iron free from silicon, the carbide of the white iron was decomposed by the silicide. From this it would seem that when all the silicon present in hypo-eutectic alloys crystallized

out with the primary austenite then after the carbide had solidified the silicide diffused and caused separation of graphite from the carbide. Many gaps remained to be filled, and he trusted that others would follow up the research.

The Testing of Lathe Tool Steels.

In Section G, Professor Ripper read a paper in which he called attention to the want of uniformity of standard in testing lathe cutting tools. The methods most commonly employed were, firstly, finding how long the tool would run under certain constant conditions, and, secondly, the determination of the cutting speed which would wear out the edge in 20 minutes. He considered that the speed test afforded the more useful information, and suggested a "speed increment" test with which he had obtained trustworthy results with the smallest expenditure of time and material. The testing lathe should be capable of very fine adjustment of speed. The tool was started on a standard cut at a given surface speed—say, 30 ft. per minute—which was gradually and uniformly increased by 1 ft. per minute until the tool gave out. This corresponded to the way the load was increased in tensile tests. If the mean cutting speed throughout the test were multiplied by the area of cut, the product gave the cubic measure of material turned off and represented the efficiency of the tool.

The Testing of Files.

Professor Ripper also read a paper on this subject. He attributed certain irregularities in results obtained with the Herbert testing machine to the fact that the teeth of the file worked precisely in the same grooves on the test-bar stroke after stroke. This occasionally resulted in the teeth ceasing to cut and the face of the bar assuming a glazed appearance. He had devised a modification in which the ends of the file were connected to ball joints instead of being held rigidly. This ensures a variation of the path traversed by the file, and irregular results were to a great extent got rid of.

The Closing and Welding of Blowholes in Steel Ingots.

Professor H. M. Howe, of New York, stated in his paper that his investigations went to show that the gases in blowholes are reabsorbed and that the blowholes can be welded, but that reheating and rolling without delay at a high temperature should be carried out.

Professor Arnold was not able to endorse the author's conclusions as applying generally. There were blowholes free from oxidation which might be welded, and there were oxidized blowholes which could not.

Mr. W. H. Hatfield thought that the production of blowholes to exactly the extent necessary to remedy piping would be found very difficult. He deprecated the practice as tending to the production of unsound material.

Institute of Metals.

The autumn meeting of this Institute was held at Glasgow and was opened on Wednesday, Sept. 21, when Sir Gerard Muntz presided. Abstracts of the papers presented will be found on page 561 of our October issue and in the Synopsis of this issue. In the following an account of the discussions which followed is given.

Magnetic Alloys Formed from Non-Magnetic Materials.

The discussion of Mr. A. D. Ross's paper was opened by Dr. C. H. Carpenter, who remarked that all the alloys showing high magnetic properties contained manganese, which itself contained a certain amount of iron. The point arose as to the influence of the iron. He would assume that it was negligible, but the fact remained that all the magnetic alloys produced from non-magnetic materials contained some manganese, and he thought it would be premature to state that the influence of the iron was really negligible until similar results had been obtained from non-magnetic materials without manganese.

Prof. A. Gray remarked that manganese steel, which was generally regarded as non-magnetic, could be rendered strongly magnetic by adequate treatment. With regard to Dr. Carpenter's reference to the existence of iron in the manganese, it was a remarkable fact that a very small quantity of iron caused much greater magnetic effects than would be supposed, possibly through exerting an effect upon the structure of the metal and also upon its behavior under magnetic force.

Mr. F. Heap was of opinion that a metallographic research would throw more light upon the magnetic properties of these alloys than chemical examination would.

Mr. E. L. Rhead gave the results of investigations of the effect of the addition of various metals to manganese. Zinc and cadmium gave a considerable magnetic value, while the action of antimony and bismuth was much more marked. The results appeared to indicate that the probable effect of all these metals on manganese was to alter the structure of the material in such a way as to render it magnetic. Phosphorus and arsenic had a similar effect, but to a less extent.

Mr. J. S. G. Primrose mentioned that in the last report of the alloys research committee Dr. Rosenhein stated that alloys which were found to be magnetic could be grouped to a very great extent by their structural differences.

Heat Treatment of Brass.

Messrs. G. D. Bengough and O. F. Hudson presented a paper on "The Heat Treatment of Brass: Experiments on 70:30 Alloy." In the discussion Dr. C. H. Carpenter asked if the authors had found any second constituent in the burnt specimens? It was important to have that point cleared up. His own research indicated that in the 70:30 brass there was an inversion at about 460° C., and that change had also been detected pyrometrically.

Prof. T. Turner thought that Dr. Carpenter's explanation was correct. In the case of steel it was manifest that separation occurred under the conditions in question. However, the matter demanded careful investigation, for there was no doubt that questions connected with annealing presented the greatest mysteries of metallurgy at the present time.

Mr. F. Johnson said that manufacturers wanted a ready means of recognizing the results which might be expected from heat treatment at various temperatures, and the danger of over-annealing. Tests at high temperatures would be valuable.

Mr. H. H. Greer asked if the tubes which were burnt in annealing could be restored to their original condition? That was a point of importance for the manufacturers to study so that workmen's mistakes might be rectified.

Sir Gerard Muntz agreed that a change did occur in brass at about 450° C., and that might possibly explain some occurrences of which there had been no adequate explanation hitherto.

Mr. Hudson, in his reply, said that he had not observed any signs of the second constituent mentioned by Dr. Carpenter.

The Effect of Impurities on Copper.

Mr. F. Johnson read a paper on the effect of silver, bismuth and aluminium upon the mechanical properties of "tough pitch" copper containing arsenic, in concluding which he stated that the experiments had demonstrated that silver, bismuth and aluminium exercised very marked effects on the mechanical properties of arsenical copper and that those effects were more apparent in testing the hot-rolled than in the cold-worked and annealed material.

The discussion was opened by Prof. T. Turner, who said the paper illustrated the combination of practice with theory, concerning which a great deal had been said lately. The author's statements as to the effect of bismuth were important. The difference between the effects of 0.02 per cent and 0.05 per cent of bismuth was of great importance in the case of pure copper, but in tough pitch copper the amount might be increased to more than double the lower limit mentioned.

Dr. Carpenter considered that the theory put forward in the

paper was most probably correct. He had no doubt that the real explanation of some of the defects mentioned was the liberation of gases during solidification.

Mr. E. L. Rhead asked if antimony alone had such a deleterious effect on copper as was generally supposed, and whether the addition of arsenic modified the effect of antimony as well as of bismuth.

Sir Gerard Muntz said a great effect was produced on copper both by arsenic and oxygen, and variations in their amounts produced widely different results. It seemed to him that, before any definite conclusion could be arrived at either the arsenic or the oxygen must be kept constant in quantity. He advocated the use of metallic arsenic. He believed it was a fact that liberation of gases did occur at the moment of solidification. He did not see the necessity for casting copper under pressure, for no gases ought to be in the metal when it was finished.

The Efficiency of Scientific Research.

An article under this heading from an engineering correspondent in the *Times* of Sept. 14 called attention to the inefficiency and loss arising from the conditions under which research work is carried on in England at the present time, and from the reluctance to publish the results arrived at, especially when these results appear fruitless. The author says: "Men are afraid to mark the blind alleys of their searchings with a notice board. They pass on to other paths, and others time and again repeat the same futile cycle"; and he considers that the want of publication of inconclusive research is the chief cause of inefficiency. As a remedy for the large amount of duplication and the synchronous waste of effort arising from independent competent investigators working on the same subject, both in the experimental departments of large firms and in private laboratories, the author advocates the formation of a "Research Co-ordination Committee" to receive and publish reports, to allocate special spheres to the laboratories of universities and technical schools, and to disseminate information. Finally, the competition of endowed and subsidized laboratories with the work of the independent professional man is condemned as tending to extinguish the services rendered to scientific progress by the private consultant by forcing him to direct his energies into other channels.

Commenting on this article the *Times* expressed the opinion that all who have experience of the mental and physical labor and the material cost involved in scientific research will in some measure agree with the general principles advanced by the author of the article.

Market Prices.

September, 1910.

Copper has had a downward tendency throughout the month, principally owing to the heavy stocks now carried constituting visible supplies. It opened at £55 15 0, falling slowly to the 21st, £54 16 0, recovering slightly on the 23d and closing at £57 16 0.

Tin has been very unsettled, rising sharply from the 2d to the 7th, when it had gone from £158 to £166. With fewer fluctuations it fell by the 14th to £156, thence recovering to £159 (27th), and closing at £158 10 0.

Lead, English, has been steady, with slight depression from 8th to 19th, closing at £13 2 6.

Iron, Hematite opening at 65/3, dipped sharply about 5th to 64/2, thence recovering, fell away on the 12th to 64/-, ultimately recovering to 65/-.

Scotch Pig has fallen off throughout the month, opening at 55/5½ and closing at 54/10½.

Cleveland Warrants have had a downward tendency, opening at 49/5 and sloping off gradually to 48/10. The industrial situation has affected iron prices.

	£	s.	d.
Alum, lump, loose, per ton	5	12	5
Antimony, black sulphide powder	24	0	0
Borax, British, refined crystal	16	0	0
Sulphate of ammonia, f.o.b. Liverpool, per ton	12	15	0
Sal-ammoniac, firsts, delivered U. K., per ton	42	0	0
Copper sulphate, per ton	18	0	0
Camphor, 1-oz. tablets		1	7
Caustic soda, ash, 48 per cent ordinary, per ton	5	10	0
Shellac, standard, T. N., orange spots, per cwt	4	1	0
Sulphur, recovered, per ton	5	0	0
Carbolic acid, liquid, 97/99 per cent, per gal			11½
Hydrochloric acid, per cwt		5	0
Creosote, ordinary good liquid, per gal			2½
Naphtha, solvent, 90 per cent at 160° C., per gal			10½
Petroleum, Russian, spot, per gal			5¾
Platinum, per ounce, nominal	6	17	6
Copper ore, 10 to 25 per cent. unit	8/10½	to	9 10½
Tin ore, 70 per cent, per ton	£95	to	97
Mica, small slab, in original cases, per lb.	6d.	to	1 0
Litharge, flake, per ton	15	0	0
Quicksilver, per bottle	8	7	6
Red lead, genuine, per ton	15	10	0
Bleaching powder, 35 per cent, per ton	4	5	0

India rubber, Para fine, opened at 7/11 and had not dropped much in price by the 21st, when it was 7/3½d. During the last week, however, this has again fallen off, closing at 6/6.

In the chemicals it will be noted that sulphate of ammonia has advanced 19/-, copper sulphate declined, shellac slightly so, naphtha 2½d, while platinum continues to appreciate.

Synopsis of Metallurgical and Chemical Literature.

Gold and Silver.

Filtration of Slime.—The filter press has long been the favorite means of handling slime in Australia, but the vacuum filter is becoming a serious rival. With a view to stimulating interest in the cheaper working of the filter press, M. W. von Bernewitz has contributed to the current *Proceedings* of the Australasian Inst. of Min. Eng. some of the details of filter pressing slime at the Associated Northern Blocks. The average value of the ore is $6.50 per ton. It is dry crushed in Krupp mills, roasted in Merton furnaces, ground in pans with weak KCN solution, the slime thickened, agitated and pressed in Dehne presses. Press filling and washing are done with pumps. Each press holds 4.75 tons dry slime. The pressure used in filling is 60 lb. and the time required is 15 min. Washing is done at 74 tons pressure, forcing about four tons of water through in half an hour. Tests showed that at least 20 min. is required thoroughly to displace the cyanide, after which the residue has a value of 30c. per ton. Continued washing would reduce the value to 20c. per ton on slime from ore that had been well roasted. On ore that was high grade or that had been poorly roasted the difference due to long washing is greater; with a residue assaying $1.90 per ton the washed samples would be about 70c., and on a 95c. residue the washed sample would be worth about 48c. per ton. This shows the variations that can occur in washing. After washing the press is dried for 2 min. with air at 90 lb. pressure, using 2640 cu. ft. free air. The residue contains 22 per cent moisture. Two men dump 11 presses in 8 hours, being paid 64c. per press. The cycle of operations requires time in minutes as follows: Filling, 15; washing, 30; drying, 2; dumping, 30; total, 77 min. During 1909, 44,163 tons were treated, giving an average extraction of 93 per cent, and costing for agitating and cyaniding 31c., and for filter pressing 25c. per ton. Cost for disposal of residue, 10c. per ton.

The author calls attention to the need of some method of discharging the presses without opening them and mentions a style of press devised by one of the managers, in which pressure water can be directed against each cake, causing it to disintegrate and flow out through a launder formed in the press plates. From the brief description, the idea seems to follow that of the Merrill press which was developed at the Homestake, and has since been extensively used in this country and in Mexico.

Zinc Precipitation.—A series of experiments undertaken to ascertain just what occurs in each compartment of zinc boxes, with a view to possible economy in the use of zinc, is communicated to the July, 1910, *Journal* of the Chem. Met. & Min. Soc. of S. Africa, by F. D. Phillips. The first experiment was made on strong extractor boxes of seven compartments. Precautions were taken in getting fair samples, 80 cc. of inflow and outflow solution being taken every 15 min. for a period of eight hours.

The following table shows the results of sampling a box two days after filling with fresh zinc, with precautions to prevent the formation of channels. Flow, 1.54 tons of solution per cu. ft. of zinc in 24 hours. Solution, 0.00209 per cent KCN; 0.005 per cent NaOH.

Comp. No.	Entering dwt. per ton.	Leaving dwt. per ton.	Extracting dwt. per ton.	Per cent extraction.	Per cent ext. of original value.
1	2.56	0.48	2.08	81.2	81.2
2	0.48	0.14	0.34	70.8	13.3
3	0.14	0.06	0.08	57.1	3.1
4	0.06	0.03	0.03	50.0	1.2
5	0.03	0.01	0.02	66.6	0.8
6	0.01	0.01	..	..	..
7	0.01	0.01	..	..	..

If the excellent work of the first compartment had continued through the other compartments the precipitation would have been complete in four compartments, whereas six were necessary. Why does precipitation fall off in the lower compartments? It appeared that the seventh compartment might be dispensed with, and this was done with successful results as shown by analyses made later.

Experiments with a lighter pack of zinc (330 lb. per compartment, or 18.3 lb. per cu. ft., as against 420 lb. per compartment, or 23.3 lb. per cu. ft.). Again the first compartment did all the work, giving an extraction of 71 per cent; the succeeding compartments precipitated the same amount each, but the assays showed that the lighter pack of zinc gave too high tailing value.

Similar experiments with slime extractor boxes showed that the first compartment did 90 per cent of the precipitation. The temperature of the slime solution is raised by waste steam heat, and this is believed to play an important part in the precipitation. In an effort to see whether it would pay to try recovering gold from tailing solutions, leaving the extractors with 0.02 dwt. gold, it was found that it would be profitable to put on an additional zinc compartment in each box.

Reinforced Concrete for Stamp Mills.—The heavy gravitation stamp has developed a number of new problems, among which is that of overcoming the effect of increased vibration on the foundations and frame. In the *South African Mining Journal* this phase of the subject, among others, is discussed by G. A. Robertson. The first notable departure from the old timber king posts was at the City Deep mill, South Africa, where reinforced king posts were constructed. Prior to that, reinforced concrete had been used in the mortar box foundations. The success of the City Deep king posts leads the writer to suggest that a single monolithic structure comprising foundations, king posts and other frames of reinforced concrete should be successful. He points out that the lack, hitherto, of adequate means for testing the effect of vibration on different forms of reinforcement has now been supplied by the invention of machines for the purpose.

An interesting point is brought out in connection with the manufacture of reinforced concrete parts, viz., in the use of molds attached to a vibrating table which is operated when the molds are being filled. The vibration, combined with the sudden arresting of the motion, has the effect of producing a rapid solidification of the particles prior to the initial set of the cement. It is claimed that great density and homogeneity are thus obtained, and that cavities are entirely eliminated. This is in agreement with the observation that the vibration of a stamp mill produces a very dense concrete when made during the operation of the mill. After the initial set of the cement the vibration has no effect on the strength of the material

Removing Iron from Ore Pulp.—The presence of particles of iron in pulp is a disadvantage and any means of removing it easily is desirable. The *Transactions* of the South African Inst. Elec. Eng., for May, 1910, contains a description of a device for this purpose, by W. C. Brown. It consists of a magnet with a circular pole, placed at an angle of 35 deg. under the classifier. As the pulp flows from the classifier it spreads over the pole of the magnet and any particles of iron are retained and brushed off later. The magnet revolves at the rate of 6 r.p.m. It has a cast-iron hollow core 6 in. in diameter and 10 in. long. The exciting coil consists of 5120 turns of No. 18 copper wire. The exciting current is 2½ amp, 110 volts, d.c. The pole plate is 20 in. in diameter and turned perfectly flat on top. Only the very finest iron is lost, this being due to the fact that the flow of pulp prevents the finest iron particles coming in contact with the magnet pole.

Slime Settler.—Description of a cheap and efficient settling tank is given by R. E. Huntley, in the *Journal* of the Chamber of Mines of Western Australia. It consists of a galvanized iron tank, 20 ft. in diameter and 5 ft. deep. A false bottom of wooden sectors is arranged in the shape of an inverted cone, the apex of the cone being at the center of the tank and the base about 12 in. above the true bottom of the tank. The sectors are of 1 in. wood, ½ in. wide at one end and 5½ in. wide at the other, and are spaced ⅛ in. apart, so that slime may build up under them to form a solid conical bottom.

The overflow of the sand collecting vats is run to this settler, where it is received in a funnel placed above the middle of the tank. The funnel is 9 in. in diameter and extends 6 in. below the level of the pulp when the tank is full. The funnel is surrounded by a bucket 15 in. in diameter, the overflow of which is 3 in. below the surface. This arrangement gives quiet and even flow of slime into the tank. Clear water overflows the rim of the tank. When a tank is full, it is allowed to settle for 30 min., after which any clear water is siphoned off, and the sludge is pumped to the agitation vats by a 4-in. centrifugal pump. When the settler is being emptied a jet of weak cyanide solution is added to thin the sludge.

The overflow water contains 4 to 8 per cent solids, and the slime is thickened to 50 or 70 per cent solids. Time for emptying a vat containing 30 to 40 tons dry slime is 1½ hours. The agitation and aeration caused in the pumping tend to improve the extraction.

Lead and Copper.

Electrolytic Lead Refining.—In an extended description of the Trail (B. C.) lead smelter and refinery, in the Sept., 1910, issue of *Mines and Minerals*, J. M. Turnbull gives details of the electrolytic lead refining. The tank room contains 240 tanks, 3 ft. x 8 ft. x 3½ ft., made of fir lined with asphalt, arranged in six double rows in a series of cascades. The electrolyte is kept in circulation by pumps. Each tank holds 20 anodes which are placed in position by a crane carrying a spacing frame. The anodes weigh 370 lb. each, and are cast with shoulder lugs to rest on the copper current bars. They are spaced 4½ in. apart.

The cathodes are sheets of pure lead made on a special machine invented in the refinery. The cathodes are 1/16 in. thick and are hung from copper bars between and outside of the anodes, there being 21 in each tank. The electrolyte is lead fluosilicate with free fluosilicic acid, averaging 12 per cent SiF_4 and 5 to 6 per cent lead. The tanks are operated with a current density of 16 amp per sq. ft. of cathode area; voltage, 0.32 volt per tank, which is reduced 0.02 volt by the contacts. The scrap from the anodes, after washing, amounts to 15 per cent, and is returned for remelting.

The electrolyte is made from fluorspar, silica and sulphuric acid, mixed in a cast-iron pan. The fumes are condensed in towers with water sprays, the acidified water being used over and over again until the strength is about 30 per cent fluosilicic acid

It is proposed to add 180 tanks which will raise the plant capacity from 75 to 100 tons per day, and permit the lowering of the current density to 12 amp, which has been found more desirable.

An average analysis of 2000 tons of shipping lead is as follows: Arsenic and bismuth, nil; zinc, .0005 per cent; silver, .0013; copper, .00075; lead, 99.9938; iron, .00075; tin, .0001; antimony, .0028. The anode slime analyzes: silver, 35 per cent; antimony, 25 per cent; arsenic, 20 per cent; copper, 8 per cent; iron, bismuth, silica, tellurium and selenium in smaller quantities and traces.

The slime is refined, and dore metal running 960 to 975 fine is obtained for further refining.

Roasting and Sintering.—At the twenty-third annual meeting of the Montana Society of Engineers F. M. Smith presented some data on the sintering of ores. After reviewing the history of roasting machines and the development of the various blast roasting processes, the speaker gave some data on the operation of the Dwight-Lloyd sintering furnace as used at the East Helena smelter.

The machine is a straight-line type, 28 ft. long over all, with an effective area of 30 in. wide by 150 in. long, or 31.25 sq. ft.

A 10-hp motor serves to operate the machine, the pug mill in which the charge is mixed and the platform elevator which raises the charge to the feed hopper. The sintering machine itself requires but 1 or 2 hp. The ore for roasting is crushed to ⅛ to ¼ in. diameter, mixed with about 4 per cent water in the pug mill and fed to the roaster. The product is a spongy material carrying about 4 per cent sulphur.

The machine is more flexible than the H-H process, and matte and flue dust are treated easily. Two characteristic mixtures used at East Helena are as follows: 68 per cent lead concentrate and slime; 12 per cent raw matter; 20 per cent miscellaneous sulphide ore and fine concentrates. The average analysis of these two mixtures is:

	SiO_2	Fe	Mn	S	Zn	Pb	Cu
No. 1	15.4	17.2	0.9	14.4	5.2	29.6	0.8
No. 2	11.5	16.9	1.0	13.0	2.7	30.0	1.0

These mixtures were sintered at the rate of 40 tons per 24 hours, and made excellent sintered products analyzing:

	SiO_2	Fe	Mn	S	Zn	Pb	Cu
No. 1	16.0	18.2	1.0	3.8	4.6	35.5	0.5
No. 2	13.6	18.9	1.0	4.0	6.3	35.0	0.8

In igniting the charge, gasoline is used under 80 lb. pressure. Two additional machines will be installed at this plant, each having a capacity of 100 tons per day. The present machine, with its 40-ton capacity, has about four times the capacity of one of the reverberatory furnaces with 840 sq. ft. of hearth area.

Impurities in Tough-Pitch Copper Containing Arsenic.—A paper presented by Mr. F. Johnson before the (British) Institute of Metals deals with "the effect of silver, bismuth and aluminium upon the mechanical properties of "tough-pitch" copper containing arsenic. The results of this investigation show that all three of the impurities—silver, bismuth and aluminium—have a very marked effect on the mechanical properties of arsenical copper, effects which are more apparent in the tests of the hot-rolled than in those of the cold-worked and annealed material.

The effect of silver in the proportions usually found in commercial "tough" arsenical copper may be considered as beneficial on the whole as regards mechanical properties and negligible as regards hot-working properties.

The effect of bismuth on the mechanical properties of "tough" arsenical copper which has been rolled is not serious up to 0.10 per cent; but no commercial arsenical copper could be regarded as fit for working at a red heat which contained so much bismuth. With 0.02 per cent, although the hot-working properties would be noticeably worse than if no bismuth were present, the copper would not be ruined. Any crude copper containing over 0.01 per cent bismuth should be regarded with suspicion, since the copper might contain traces of other impurities—*e.g.* nickel—which, while intensifying the injurious effect of bismuth, would hinder the corrective action of arsenic.

(The influence of bismuth on the mechanical properties of arsenical copper at high temperatures has not been investigated in this research.)

Arsenic and oxygen both aid in improving the mechanical properties of copper containing bismuth. In addition to the function exercised by cuprous oxide in guarding molten copper from the inroads of reducing-gases and their subsequent devastating influence, there may be ascribed thereto also the function of protecting the solid metal from the intercrystalline weakness engendered by metallic bismuth.

The effect of aluminium in arsenical copper is to destroy its hot-working properties and to impair its physical properties in the cast, and its mechanical properties in the rolled condition. (From advance sheets.)

Alloys.

Heat Treatment of Brass.—A paper read before the Institute of Metals by Messrs. G. D. Bengough and O. F. Hudson describes the results of an investigation in which the effects of heat treatment were studied on rolled and drawn 70:30 brass of varying size of section and of somewhat varying character, as made by different manufacturers. The experiments were also intended to throw some light on the question of "burning." For the purposes of the paper a sample of brass is considered to be burned when its ductility, as measured by the percentage elongation, has been lowered by heat treatment.

From the detailed study of the effects of various annealing temperatures upon the mechanical properties of cold-tested brass bars of commercial quality it appears that the best combination of strength and ductility is obtained for annealing temperatures between 600° C. and 700° C.

The true elastic limit was determined from one of the alloys used. This alloy did not appear to have a very well defined yield point, such as is exhibited, for example, in the case of mild steel.

The effects of varying the annealing period were also investigated. This factor has an even more important effect on the mechanical properties than the temperature factor. A temperature but a few degrees below the solidus, if only maintained for half an hour, will not seriously burn a bar of pure brass. A temperature nearly 100° C. lower will burn the brass if it is maintained for a sufficiently long time. Even small quantities of impurities have an important effect in increasing the tendency to burn. The atmosphere of the annealing furnace, whether oxidizing or reducing, does not appear to affect in any well-marked manner the properties of brass.

A few determinations of the loss of zinc by volatilization during the annealing process were made. This loss is appreciable, but is mainly a surface effect. The authors do not consider this to be the cause of burning.

Tests were made on the mechanical properties of brass wire at high temperatures. It was found that the maximum stress falls off continuously. The elongation and contraction of area, on the other hand, fall off to a very low value at temperatures between 300° C. and 400° C., after which a recovery of ductility occurs, and is practically maintained even at 590° C., the highest temperature employed. The suggestion is made that any handling of the material at a slightly higher temperature than usual may cause a permanent deterioration in its properties.

The authors studied the structure of annealed, over-annealed and burned brass by microphotographical methods and record new facts as to the crystal growth in brass at low annealing temperatures. (From advance sheets.)

Defects in Alloys.—A paper read before the Institute of Metals by Dr. Cecil H. Desch gives a classification and discussion of "some common defects occurring in alloys." The intended composition is assumed to be a proper one for the purpose for which the alloy is to be used. The first class of defects includes those which are present in the original casting, and are classified as (1) sponginess, due to the enclosure of gas; (2) brittleness or weakness, due to the presence of (a) oxides or other drossy matter, (b) intercrystalline metal or eutectic alloy.

The author then discusses (3) defects, due to inequalities of composition These may be due to (a) imperfect mixing of the ingredients; (b) separation by gravity during solidification; (c) segregation in the mould.

(5) Defects due to wrong thermal treatment may be caused by (a) quenching from a high temperature; (b) heating at too high a temperature or too long a time during annealing; (c) burning in the case of copper and zinc; (d) unequal thermal treatment of different parts of the same casting or forging.

The author then discusses (6) defects due to molecular change, other than that produced by mechanical stress; allotropic changes, tin plague, etc

A great variety of defects are due to mechanical rather than to chemical causes. These include (7) shrinkage cracks; and (8) defects due to molecular change produced by mechanical deformation: (a) brittleness of cold-worked metals (1. season-cracks, 2, fire-cracks, 3. strain-disease or forcierkrankheit, (b) chilling cracks, (c) cracking during hot-working.

Finally (9) corrosion and (10) erosion are briefly discussed in the paper. (From advance sheets.)

Shrinkage of Antimony—Lead Alloys.—An Institute of Metals paper by Mr. Donald Ewen and Prof. T. Turner deals with the shrinkage of antimony-lead alloys and of aluminium-zinc alloys during and after solidification. The method of Ewen's experiments was the same as in M. T. Murray's former work on copper-zinc alloys, the changes of length of a cast bar being observed, by means of an extensometer, from the moment of solidification down to the ordinary temperature.

In the work of Murray on the copper-zinc series of alloys it was shown that where a continuous succession of solid solutions separate out, the extensometer curve closely follows the solidus. Ewen now shows that in the antimony-lead alloys, where no solid solutions exist, the extensometer curves give negative results, no single bar of the series having expanded more than the original antimony. The expansion due to the formation of the solid eutectic has, however, been noted. In the aluminium-zinc series, on the other hand, where solid solutions exist, Mr. Ewen finds that the expansion curve closely follows the liquidus curve from 0 to 50 per cent of aluminium, when allowance is made for the expansion noted at the eutectic point (From advance sheets.)

Nitric Acid from Air.

The fixation of atmospheric nitrogen by means of electric discharges is the subject of five interesting papers published in the *Zeit. f. Electrochemie*, Oct. 1, and dealing with investigations carried out in Professor Haber's physico-chemical laboratory of the Institute of Technology of Karlsruhe.

F. Haber, A. Koenig and E. Platou report on experiments made with a high-tension arc with artificial cooling. The first series of experiments was made with direct-current arcs, the object being the determination of a maximum concentration at reduced pressures. The arc was produced in a fused-quartz tube, water-cooled from the outside. The curve of concentrations produced shows a decided maximum (8.16 per cent NO) at an atmospheric pressure of 150 mm mercury (voltage 1400, current 0.27 amp). In a second series of experiments the authors determined the apparent phase-difference in an alternating-current arc and the yield obtained. With a power consumption of 0.7 kw with a concentration of 3.4 per cent NO 57 grams HNO_3 were obtained per kw-hour; this gives a yield of 500 kg HNO_3 per kw-year. In view of the high concentration this figure is fairly satisfactory, although it does not represent the maximum obtainable with cooled arcs.

Proof of this is given in a paper by W. Holwech and A. Koenig, who obtained higher yields with a changed experimental arrangement using direct current. The arc was contained within very narrow cooled metallic tubes through which the air was pumped at high speed. Direct current of 1 amp at 6000 volts or 2 amp at 3000 volts was available. The best results were yields of 80 grams HNO_3 per kw-hour with a concentration of almost 2.5 per cent NO.

High-frequency alternating-current discharges were employed in experiments of F. Haber and E. Platou. The frequency does not have any great influence on the yield as long as care is taken to use a large amount of energy for the discharge and to extend the arc by blowing it out. If these precautions are not taken, yields are obtained which are below those obtained in commercial operation in European plants. It may be of interest to give these "commercial" yields. They are 50 grams to 75 grams HNO_3 with a 1.0 per cent to 2.5 per cent concentration of NO_2 and NO with preheated air. If the air is not preheated the "horn" discharge apparatus yields 50 grams HNO_3 per kw-hour.

While reduced atmospheric pressures have formerly been studied by Haber and his co-workers, F. Haber and W. Holwech have now also tried an increased pressure. But the results seem to show that no improvement of the yield can be obtained by an increase of pressure.

Finally A. Wolokitin reports at considerable length on the formation of nitrogen oxides in the hydrogen flame and the effect of pressure. The paper is chiefly of theoretical interest.

Recent Metallurgical and Chemical Patents

Copper.

The chemical treatment of copper ore containing gold and silver has always been open to the objection that the methods for profitably extracting the copper were not applicable to the precious metals, and vice versa The object of a process recently patented by William E. Greenawalt, of Denver, Col., is to extract all these metals in one operation The principles on which the process is based are these: Both sulphuric and hydrochloric acids act readily on copper in some of its combinations, chlorine acts readily on gold and silver; sulphur dioxide precipitates these metals from chloride solutions; sulphur dioxide combines with chlorine in the presence of water to form sulphuric and hydrochloric acids. The first step in the process is to form these two acids as indicated, and the next is to add an excess of chlorine to the acid solution This solution will now dissolve gold, silver and copper in ores, the metallic chlorides formed holding the silver chloride in solution. When the solution is precipitated with sulphur dioxide, the acids are regenerated and the cupric chloride is reduced to cuprous chloride and precipitated. It can be withdrawn from the solution, melted at low red heat, cast into anodes and subjected to electrolysis Several patents have been issued on various features of the process and the manner of carrying it out (968,651-2, 968,845, Aug. 30, 1910.)

Gold and Silver.

An intermittent slime filter of novel construction has been patented in the United States by John F. Webb, of London, England As illustrated in Fig. 1, which is a side elevation of the device, it consists of a rectangular filter medium which is alternately lowered, inverted, into the slime tank, and raised

to an inclined position for washing and discharge of the cake. The slime tank is represented at *f*, with the dotted lines showing the position of the filter when in the tank. The filter is shown raised for washing the cake prior to discharging. It consists of a metal box, one face of which is perforated and supports the filter cloth. The operating mechanism is a worm sector, *d*. Arms *b*, which are rigidly attached to the shaft of the sector, are pivotally attached to the filter. A connecting rod c^6, is also pivotally connected with the filter and a central support. In the revolution of the sector, the arms *b* support the filter, and with the aid of the connecting rod c^6 eventually guides the filter into its horizontal position in the slime tank. After the cake has formed the reverse of the operation lifts the filter into the position shown for washing and discharge of the cake. The weight of the filter is counterbalanced by the device shown, *c*, c^1, c^2. (968,720, Aug. 30, 1910.)

Sodium Amalgam.—The object of a recent invention by Herbert P. Ewell, of Detroit, Mich., is to manufacture sodium amalgam and maintain it in a liquid state so that it can be used in the riffles or wells of gold amalgamating devices. In the manufacture of the amalgam, the inventor uses a receptacle for the mercury, over the surface of which he maintains a stratum of paraffine to exclude the air. The sodium, in a suitable container, is introduced into the mercury beneath the paraffine, and in order to prevent the formation of a hard amalgam the sodium is kept in constant agitation until dissolved. By the agitation the mercury is always in excess of the sodium at the point of solution, and the resulting amalgam is liquid and may be conveniently introduced into amalgamating machines. (969,853, Sept. 13, 1910.)

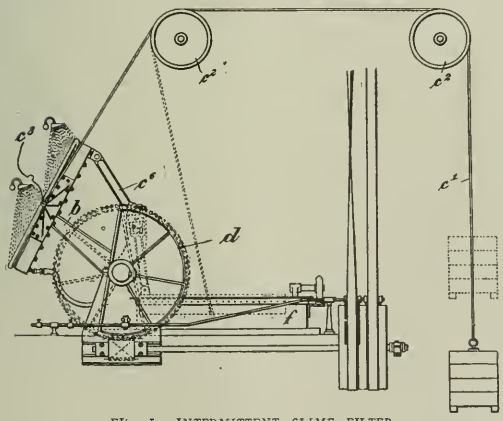

FIG. 1.—INTERMITTENT SLIME FILTER.

The solution of gold in cyanide solutions containing potassium cyanide and an oxidizing agent has been made the basis of some patents granted to John C. Clancy, of New York, and referred to in these columns in our issue of June, 1910, page 364. Recently Mr. Clancy has been granted an additional patent, the main point of which is contained in his single claim. "The method of treating pulverized ore containing gold, which consists in subjecting said ore to the action of a solution containing a soluble iodide and a substance capable of yielding nascent oxygen in the presence of said solution." It will be noted that the term gold is used instead of precious metals, as in former patents; also that no mention is made of a cyanide solution. The inventor claims that the process is applicable to unroasted ores, that it will take place in neutral, alkaline or acid solutions, and that the only addition of iodide necessary is to compensate for mechanical loss. The oxidizing agents used are sodium persulphate, ozone and nitrogen peroxide. The formula for one solution is 1 lb. of potassium iodide, 3 lb. of sodium persulphate and ½ lb. of lime or soda in one ton of water, using the solution in the proportion of two parts of solution to one of ore. (971,252, Sept. 27, 1910.)

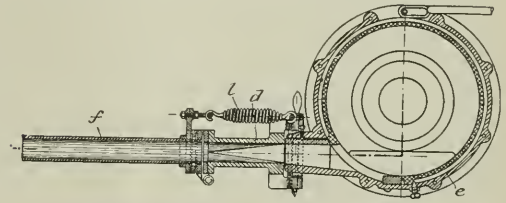

FIG. 2.—RETORT-CHARGING APPARATUS.

Zinc.

Furnace Charging Apparatus.—The convenient charging of retorts in zinc furnaces is accomplished by means of the apparatus illustrated in Fig. 2. It is the invention of Emile Dor-Delattre, of Liége, Belgium. It comprises a propeller wheel with radial blades, revolving within a casing *e*. The charging tube *f* is jointed thereto, so as to yield or move angularly in the event that the charging apparatus fouls with any part of the furnace in being moved along the line of retorts. The tube *f* and the coupling sleeve *d* may be moved horizontally when the device is not in use. The outer end of the charging tube *f* may also be moved downward vertically on a horizontal hinge, and horizontally to either side on two vertical hinges. Normally, the spring *l* holds the tube in the position shown, but in case of the tube fouling with any other part of the furnace during use the tube will yield instead of breaking. (969,254, Sept. 6, 1910.)

Mercury.

Reduction Furnace.—A continuous process for reducing mercury ores is the basis of a patent granted to Ira Hageman, of San Francisco, Cal. As illustrated in Fig. 3, the furnace is of the revolving type. It is oil-burning, as shown at 34 and 35, and has special devices to control the entrance and discharge of ore, as shown at the respective ends of the furnace. The furnace cylinder is represented by *a*, which revolves between fixed chambers *b* and *d* at the ends of the tube. The novel features lie in the means for continuously feeding and regulating the quantity of ore in the cylinder. The feed is by means of a

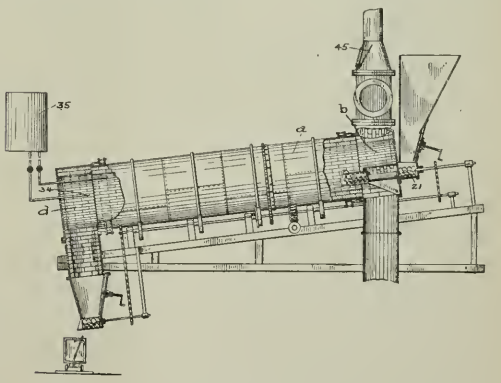

FIG. 3.—MERCURY FURNACE.

screw 21, operating in a cylinder connecting with the feed hopper. The end of the cylinder in which the screw operates is closed with a cap which is hinged so as to close the opening in the cylinder by gravity when the cap is not pushed away from its position by entering ore. The feed of ore becomes so packed in the cylinder that it also prevents the exit of fumes from the furnace. The discharge of the spent ore is controlled

by means of a hopper at the opposite end of the furnace. It will be noted that in each hopper, feed and discharge there is a controlling plate operated by a hand screw in each case, for the purpose of regulating the quantity of material delivered to the screws. Thus the quantity of ore in the furnace and the length of time it remains in the furnace may be controlled. Provision is made, as shown at 45, to connect the furnace with a stack until the proper running conditions have been obtained, after which the fumes from the ore can be directed through condensation chambers of such pattern that the fumes have to pass the entire length of the chambers before they can be discharged. There is also provided a suitable automatic water sealed safety valve on the condensing chambers. (970,705, Sept. 20, 1910.)

Roasters.

Wilfley Roasting Furnace.—The conditions necessary for successful roasting preliminary to magnetic separation are decidedly different to those obtaining in ordinary desulphurizing roasting. A patent recently was granted to ARTHUR R. WILFLEY, of Denver, Col., on a roasting furnace designed for this special kind of work. It is illustrated in plan in Fig. 4. It is a rotary-hearth, stationary-rabble furnace, with firebox, feed hopper and a special device for removing the roasted ore from the hearth after it has traveled the full length of the hearth and has been subjected to the effect of the heat for a sufficient length of

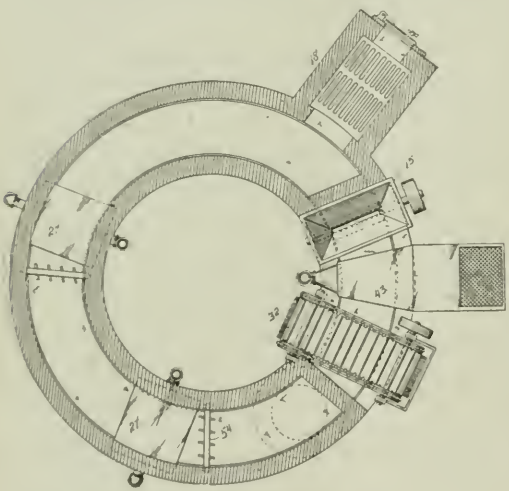

FIG. 4.—ROASTING FURNACE.

time. The firebox is located at 18, the feed hopper at 15, the endless brush, for removing the ore from the furnace, at 32, and the housing over the air blast, which is also used to remove ore from the hearth, at 43. Stationary rabbles are indicated at 54, and cooling jackets supported above the hearth at 27. In operation the ore is fed onto the hearth from the hopper and travels with the hearth past the firebox, passing later under the cooling jackets, in which water circulates constantly, and finally comes to the endless brush which removes it from the hearth. The air blast is used last to remove any particles, however small, which the brush has missed. The brush and air blast may be used separately or in combination. The rabbles may be dispensed with if the feed from the hopper is sufficiently thin to require no rabbling. The complete removal of the ore from the hearth at the end of the operation is highly essential to the subsequent operation of magnetic separation, as if the ore is allowed to remain in the furnace after the magnetic roast has been completed the subsequent heating will defeat the object of the process. (969,927, Sept. 13, 1910.)

Dust in Gas.

Settling Flue Dust.—Many schemes have been tried in the settling of dust from the flue gases of metallurgical furnaces, but usually these schemes have embodied the idea of causing the gas to flow in a tortuous path, both to reduce its speed and to provide baffles for retarding the solid particles which it is desired to settle. MR. HENRY HOWARD, of Boston, Mass., has been granted a patent on the idea of using superposed horizontal or inclined shelves in the flue, for the purpose of providing a means of settling the dust and retaining it. He has found that when the stream of dust-laden gas is confined in close proximity to a dust-retaining surface, such as metallic shelves separated by a vertical distance of 2 or 3 in., or less, a surprising result is secured. The chamber acquires practically the character of a filter, retaining even the finest particles of dust. The increased efficiency of the operation is believed to be due largely to the retardation, by the dust bed lying upon the shelves, of the lower portion of the shallow confined stream of dust-laden gas. It is observed in practice that the inlet end of the shelves quickly becomes obstructed and nearly closed by the accumulated deposit of dust, which slowly extends to the opposite end of the shelves. It is quite possible that those obstructions set up eddies in the current of gas, which aid in settling the dust particles. At suitable intervals the dust may be removed by some mechanical means. (970,053, Sept. 13, 1910.)

Electric and Electrolytic Furnaces.

Combined Resistance and Arc Furnace.—Mr. JOHANNES HÄRDEN, of the Gröndal-Kjellin Company, of London, England, patents a furnace in which resistance and arc heating are combined. Since the Gröndal-Kjellin Company is the owner of the induction furnace patents, it may be noted that in Mr. Harden's new combination furnace heating by induction is not employed. For resistance heating composite terminal plates are used as in the Roechling-Rodenhauser furnace, while arcs play on top of the bath and heat the refining slags to a high temperature. However, with a furnace of this combination the following difficulties are experienced, according to Mr. Härden: The metallic terminal plates are faced with conductors of the second class which become electric conductors when heated. When this takes place they become very hot and the metallic terminal plates are liable to be destroyed by melting. This would represent a serious source of danger. The above combination is, therefore, "not able to fulfil the requirements for dealing with large quantities of metal, since to avoid the destruction of the terminal plates the bulk of the electrical energy has to be conveyed to the charge by means of the electric arc or arcs, and when depriving the arc or arcs of electric energy the supply of energy to the whole furnace must be discontinued to prevent the terminal plates from being destroyed by overheating." Mr. Härden overcomes this difficulty in the following manner: In the composite terminal plate he provides a very steep temperature gradient between the face exposed to the charge of the furnace and the rear face of the plate, this rear face being kept comparatively cool. For this purpose the terminal plate is made of layers of material of increasing electric conductivity. At least three layers are employed. The metallic plate itself is artificially cooled. But between it and the highly refractory facing of magnesite, chromite or zirconite an intermediate layer of graphite or other carbonaceous material is packed. Such combination furnaces are claimed to be suitable for the treatment of "large quantities of steel, say, 30 tons or 50 tons." (967,909, Aug. 23, 1909.)

Metallic Sodium.—Mr. PAUL LEON HULIN, of Grenoble, France (assigned to Société d'Electro-Chemie), remarks that in the manufacture of metallic sodium by electrolysis of fused caustic soda the quantity of sodium produced decreases somewhat after a short time. In such cases the electrolytic cell has been dismantled and refilled with fresh caustic soda. Mr. Hulin states that this is unnecessary and that it is suffi-

cient to take the cathode at intervals out of the bath, bring it into contact with the water or an aqueous solution, and then replace it in position. This remedy is stated to be effective. (971,198, Sept. 27, 1910.)

Electrolytic Processes.

Sodium Chloride Electrolysis.—A special feature of the familiar Townsend cell is the subject of a patent recently granted to Mr. CLINTON PAUL TOWNSEND (though first applied for on April 12, 1902). It will be sufficient to quote here two of the claims. The first claim relates to the "process of electrolyzing solutions, which consists in passing an electric current through the solution to an electrode and recovering a product of electrolysis by means of a liquid which is substantially immiscible with water or an aqueous solution and inert toward said product." The ninth claim relates to the "process of electrolyzing solutions, which consists in passing an electric current through the solution and a diaphragm to an electrode, recovering a product of electrolysis by means of a liquid which is substantially immiscible with water or an aqueous solution and inert toward said product, and circulating said liquid in contact with said electrode by means of a gaseous product of the electrolysis." In the commercial Townsend cell the "immiscible liquid" is oil and the "gaseous product of electrolysis" used for circulation is hydrogen gas set free at the cathode. (972,947, Oct. 18, 1910.)

Barium and Strontium Hydroxides.—According to a process of Mr. ADOLF CLEMM, of Mannheim, Germany, barium chloride and strontium chloride are treated by electrolysis for the production of the hydroxides and chlorine. The electrolytic decomposition is carried to that point only where the chlorine solution is nearly saturated with the hydroxide (which is, for example, the case when electrolyzing barium chloride at a temperature of from 20° C. to 25° C. with an amount of from 35 grams to 40 grams of barium hydroxide within 1 liter). During electrolysis the cathode liquid is preferably heated to make use of the greated solubility of the hydroxide at higher temperatures (60 grams to 70 grams of barium hydroxide being carried in 1 liter at a temperature of from 50° C. to 60° C.). The solution thus obtained is then intensely cooled and the larger crystallizing part of the hydroxide in the solution is obtained in a pure state by centrifugal action and washing. The remaining liquid contains slight quantities of hydroxide which is turned back with the chloride into the process or which can be deposited in form of pure carbonate by means of carbonic acid. (973,171, Oct. 18, 1910.)

Magnetic Separation.

Detinning Tin Scrap.—An interesting application of magnetic separation is made in a detinning process of Mr. CHARLES J. REED, of Philadelphia, Pa. The tin scrap, mixed with finely divided magnetite, is charged into a rumble, in which it is subjected to heat and agitation until the tin softens and is scraped off from the pieces of sheet iron by the magnetite particles. The pieces of sheet iron are then separated from the magnetite and tin powder by means of a screen. Finally the magnetite is removed from the tin by magnetic separation. The process is also stated to be applicable to the treatment of galvanized-iron scrap, electroplated metal scrap, metal-carrying solder etc. (971,405, Sept. 27, 1910.)

The **W. S. Tyler Co.**, of Cleveland, Ohio, has just issued a handsome catalog of 95 pages on "double crimped" wire cloth and mining screen. Several pages are devoted to information relative to the proper gaging, definition of mesh and the selection of suitable screen for various services and well executed full-page plates facing the text show various sizes, gages and types of wire cloth, together with suggestions as to its uses. The advantages of the "double crimp" in preserving accurate spacing and in presenting a weaving surface with crimps or arches of the same height, in both warp and shoot, give to this cloth an exceptionally long life.

Bakelite.

Through the recent incorporation of the General Bakelite Company the manufacture of bakelite has reached the large-scale commercial stage. The General Bakelite Company has been formed by a few men exceedingly well known as successful manufacturing chemists and some other well-known business men.

The General Bakelite Company controls all bakelite patents of Dr. Baekeland except the European patents, which are the property of the Bakelite Gesellschaft, 33 Lutzowstrasse, Berlin, Germany.

The following is a list of the United States bakelite patents: 939,966, 941,605, 942,699, 942,700, 942,808, 942.809, 942,852, 949,671, 954,666, 957.137. Other patents are pending.

The General Bakelite Company has been incorporated in New York State with a paid-up capital of $800,000. Dr. Leo H. Baekeland is president, Mr. Jacob Hasslacher is first vice-president and treasurer, Dr. Hans Foersterling second vice-president, Mr. Hugo Dubois secretary. Two other directors are Mr. Samuel T. Peters and Mr. R. H. Williams, both of New York City.

The manager of the General Bakelite Company is Mr. Herbert S. May, a graduate of the Massachusetts Institute of Technology.

A plant is now being equipped at Perth Amboy, N. J. Until it is finished goods are being turned out in Yonkers.

The company will undertake the systematic introduction of bakelite for different purposes and manufacture the raw materials therefor, supplying users of bakelite with different grades of preliminary products, leaving the mechanical side of the process to other manufacturers, who will produce the goods from them.

The offices of the company are at 100 William Street, New York City, where there is an interesting exhibition of samples of bakelite, illustrating the exceedingly manifold applications of this new substance.

These various applications of bakelite have been discussed at various times in this journal (see especially our Vol. VII, pages 111 and 276, and Vol. VIII, page 328). But the time seems now opportune to give a concise and full summary of the nature and properties of bakelite, indicating the many fields of its usefulness.

Bakelite is a new substance which, in its different forms, offers the advantages of hard rubber, Japanese lacquer, celluloid, and in many respects excel the properties of these products. Bakelite is not merely a mixture or so-called "compound" like so many rubber, shellac or other compositions, but a well-defined chemical substance of specific properties; it thus adds an important member to the industry of plastics.

In the synthesis of bakelite we have a more recent instance of a laboratory reaction, which seems to run along lines parallel to those of the delicate physiological changes which occur in plant life.

This synthesis has some relation to the subtle process which in the Japanese lacquer tree engenders the substance known under the name of Japanese lacquer. Up till now the latter has been a product of limited application and an article of luxury on account of its excessive price and scant supply. Henceforth a new material has become available which can be manufactured in practically limitless quantities and at a price which allows a wide range of uses.

The laboratory method is an improvement on nature's process, not only because the reaction is quicker and less expensive, but the resulting artificial product is decidedly more resisting to chemical and physical influences.

Dr. L. H. Baekeland, the inventor of this process, has a record of more than a quarter of a century of research in the field of pure and applied chemistry. To those outside of the chemical profession he is more particularly known through his inventive and industrial achievement in the industry of so-called

gaslight photographic papers, of which velox is the best known representative.

He presented the first general account of the synthesis of bakelite and its industrial application before the New York Section of the American Chemical Society on Feb. 7, 1909, several years after he had started his investigations on this subject and after he had obtained due protection by filing the necessary patents here and abroad. This paper was printed in our Vol VII, page 111.

The word bakelite was selected as a simpler trade name for the longer and less expedient chemical name oxybenzylmethylenglycolanhydride.

For the layman it is somewhat startling that in the formation of bakelite two strong smelling liquid substances (carbolic acid and formaldehyde) should, under certain well defined conditions, react chemically upon each other and solidify to a transparent amberlike solid substance entirely devoid of odor and taste.

The various uses of bakelite will be better understood if we mention summarily some of its properties.

The final product, bakelite C, is infusible and can resist temperatures of 572° F. (300° C.) or over. It is insoluble in all solvents and can withstand strong chemicals, oil, hot water, steam, etc. All this in conjunction with its dielectric properties makes it an excellent electrical insulator.

Bakelite can be compounded with various filling materials and shaped or molded with great accuracy to articles of unusual strength. Or it can be sawed, turned and polished.

It can be used to impregnate wood and other porous bodies, rendering them harder and more resisting to chemical and physical influences. Or it can be obtained as transparent as glass, or colored to suit special requirements.

It can be purchased at a reasonable price, the latter varying with its different qualities.

Although bakelite is stronger and harder than celluloid or hard rubber, it lacks the unusual flexibility of both these plastics. Rods or plates made of it, or its compositions, are somewhat flexible, but considerably less so than celluloid or hard rubber.

On the other hand, it withstands incomparably better high temperatures and in general all chemicals and physical influences, and has the further advantage of not being attacked by solvents or by most chemicals. It does not emanate sulphur vapors like hard rubber, nor nitrous products like celluloid, which have a very disturbing influence in some electrical applications. If heated in a flame it does not suddenly catch fire like celluloid nor melt like rubber; it simply chars, then burns with difficulty.

During the act of hardening, which engenders the final product, there is a shrinkage of about 2 per cent in diameter, or in length. This shrinkage can be lessened by the judicious incorporation of filling materials and if the latter be used in large amounts the decrease in volume can be reduced to such a point as to make it hardly perceptible. Its specific gravity is about 1.25.

The bakelite process involves the utilization of three distinct and well defined varieties of bakelite, designated bakelite A, bakelite B and bakelite C. Bakelite C is the final product, whereas A and B are the transition products which enable us to mold or otherwise apply the material before its final hard condition.

Bakelite A is the initial raw material and exists in liquid, pasty or solid condition. Every variety of A, if heated at a sufficiently high temperature, changes into B, then into C. All varieties of A, whether they be liquid, pasty or solid, are still soluble in alcohol or acetone or in caustic soda, and behave as true resins. *Solid A* is characterized by the fact that it is solid at ordinary temperatures, but melts if moderate heat is applied, while stronger application of heat makes it infusible and insoluble by transforming it into B or C.

Bakelite B is an intermediate solid product. It is neither so hard nor so strong as C and may be easily mistaken for solid A, but it is different from the latter on account of its insolubility. Furthermore, it is infusible, although the application of heat may temporarily soften it, without, however, bringing it into fusion. Solvents do not dissolve it, but some of them, like phenol or acetone, may soften it or swell it without bringing it into solution. B is specially characterized by the fact that, although it is infusible, it will soften under the action of heat and then will mold and weld together if pressed in a hot hydraulic press. The latter fact differentiates it clearly from C.

Bakelite C is the final product, resulting from the application of heat on A or B, and may be considered as a chemical polymer of B resulting from a multiple molecular grouping. It is no longer a resin, because it lacks all the chemical characteristics of a resin, although physically it may resemble amber. In pure form, and if made from the first member of the phenol group, its chemical formula is represented by $C_{18}H_{18}O_3$. It has maximum strength and maximum resistance to chemical influences.

Although the preparatory work connected with the industrial side of the process has been going on with strenuous application since 1907, it was thought advisable not to jeopardize everything by a too hasty, broadcast introduction before each industrial use had been studied for a sufficiently long time on a practical commercial scale.

For this reason bakelite has been used during the last two years "on the quiet," especially for electrical purposes, while undergoing the test of time and the critical observations of every-day practice under the eyes of specialists. This work has been conducted on a sufficiently large scale to remove all doubts as to the practical side of the question. It can thus be stated without hesitation that bakelite is no longer a laboratory experiment but a product well tried in several industries.

Special conditions may, from time to time, arise which will necessitate changes in the directions to be followed for the manufacture of bakelite products. Further, the field for the application of bakelite being so unusually large, new uses sanctioned by preliminary trial will come up from time to time. All such special fields and new fields will be worked out by the General Bakelite Company, and its policy will be to abstain from recommending any applications of bakelite until first sufficient data have been gathered to make sure of its practical merits.

Bakelite electrical insulators as well as bakelite pump valves, valve disks and other molded goods are now obtainable from several of the licensees of the company.

Automatic Annealing Furnace.

Automatic operation has been extended as far as practically possible in a new annealing furnace placed on the market by the Hawley Down Draft Furnace Company, of Chicago. It was invented by its general manager, Mr. Cliff Bleyer.

All the operations are performed by electric motors, controlled by an electric clock. The furnace (Fig. 1) is heated by oil burners and after the proper temperature has been attained in starting the furnace no further attention is required except an inspection every eight or ten hours, which is sufficient to keep the temperature of the furnace uniform. But an electric pyrometer permits to read the furnace temperature at any time.

This furnace is especially suitable for light castings, like cartridge shells, and for any kind of small gray iron or metal castings requiring a short annealing period. The charge is automatically weighed into the furnace, kept there for the length of time required by the work and controlled by the clock, then automatically discharged by tilting the furnace (Fig. 2); the truck which carries the annealed castings away operates automatically on a track, discharging its load at any point for which the mechanism is adjusted; the truck then returns to its station under the furnace

The furnace is heated by two fuel oil burners of the type used in metal melting furnaces. The flames enter the bottom of the furnace, but are deflected toward the outlet (Fig. 1) so as not to make contact with the castings.

The electric clock controls the motors in such a way that at the specified moment either the minute or the hour hand makes contact with one of a set of plugs placed in the face of the clock and thus closes an electric circuit which operates the motor.

The furnace is charged from an overhead hopper of sufficient capacity to keep the plant in continuous operation. From the hopper the castings pass to an automatic scale which can be adjusted to weigh charges in any amount from 50 lb. to 500 lb. When a lot of castings has been discharged from the furnace usually a short interval is allowed to let the furnace recover any lost heat. Then a new charge is introduced into the furnace, the castings sliding from the scales into the furnace.

To control this sliding movement the wedge-shaped brick projecting upward from the bottom of the furnace at the center is of importance. As seen in Fig. 1 the face of this brick is curved, so as to act as a slide and permit the castings

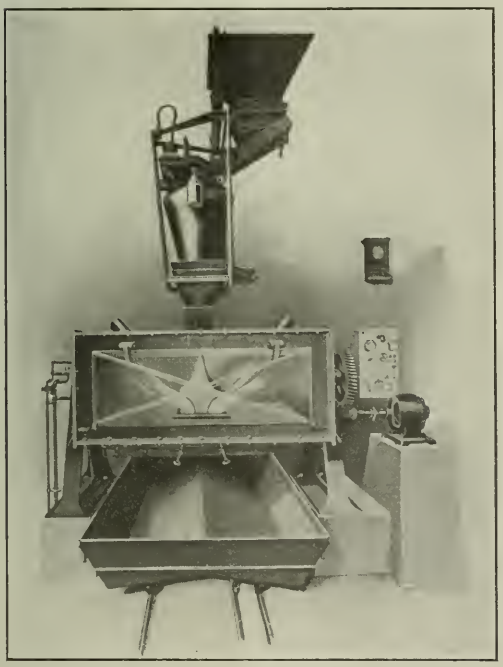

FIG. 1.—GENERAL VIEW OF AUTOMATIC ANNEALING FURNACE.

to be deposited in the furnace without any sudden jar. This brick also serves as a flame deflector to protect the castings from direct contact with the flame.

The oblong shell of the furnace is formed by six heavy cast-iron plates bolted together. It is lined with from 4 in. to 8 in. of firebrick, and between the iron shell and the firebrick lining an interlining of asbestos varying in thickness from ½ in. to ⅞ in. is interposed for heat insulation.

The tilting of the furnace is accomplished by gears operated by two electric motors, both of which are controlled by the electric clock. One of the motors controls the discharge of the castings from the furnace and its return to the position in which it receives a new load, through an angle of 90 deg. The other motor tilts the furnace forward and backward through an arc of 45 deg. and turns the castings. The object is to change at regular intervals the position of the castings in the furnace so that they are subjected to a uniform heat treatment on all sides. For instance, the castings may be turned around once in the middle of the heat.

Every charge weighed into the furnace is recorded and the amount of material that is passed through the furnace can be seen at any time, so that a check is obtained on the foundry and finishing departments and also an exact record of the amount of material handled.

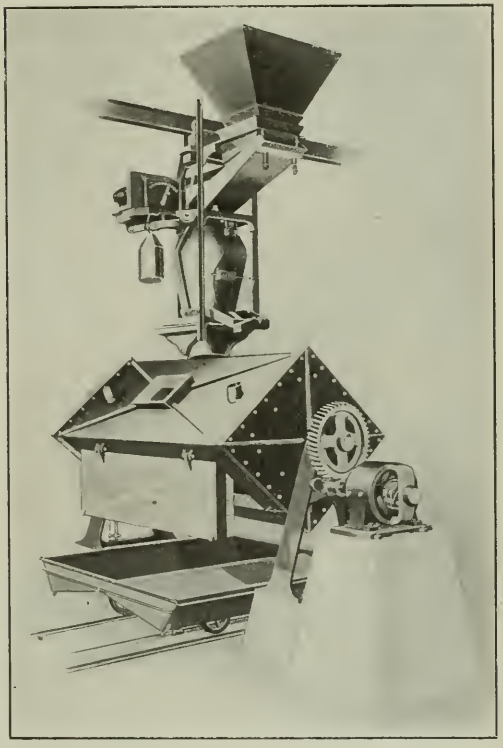

FIG. 2.—DISCHARGE OF AUTOMATIC ANNEALING FURNACE.

Volatile Matter of Coal.

"The Volatile Matter of Coal" is the title of the first bulletin just issued by the new Federal Bureau of Mines. The authors, Horace C. Porter and F. K. Ovitz, conducted their investigations at the Pittsburgh station while it was under the Technologic Branch of the Geological Survey, the work being a continuation of the fuel investigations begun several years ago at the Louisiana Purchase Exposition, St. Louis, Mo.

The results obtained at that plant showed that the work of determining the fuel values of the coals and lignites in the United States with a view to increasing efficiency in their utilization would be incomplete if it did not include systematic physical and chemical researches into the processes of combustion. Hence in their later investigations the authors carried on such researches, concentrating attention on those lines of inquiry which promised results of greatest economic importance. This bulletin is therefore a report on an investigation of the volatile matter in several typical coals—its composition and amount at different temperatures of volatilization.

Quoted directly the authors say: "The investigation has already shown that the volatile content of different coals differs greatly in character. The volatile matter of the younger coals found in the West includes a large proportion of carbon dioxide, carbon monoxide and water, and a correspondingly small proportion of hydrocarbons and tarry vapors. The older bituminous coals of the Appalachian region yield volatile matter containing large amounts of tarry vapors and hydrocarbons,

FIG. 1.—LARGE STORAGE TANKS, TOWER PIPE AND RECEIVERS.

difficult to burn completely without considerable excess of air and a high temperature. Coal of the Western type, moreover, gives up its volatile matter more easily at moderate and low temperatures than that of the other type. The volatile matter produced at medium and low temperatures is rich in higher hydrocarbons of the methane type, such as ethane and propane, which contain a larger portion of carbon than is present in methane.

FIG. 2.—MILL ROOM, WITH 200-GAL. JAR.

"These facts help to explain the difficulty of burning Pittsburgh coal, for example, without smoke, the low efficiency usually obtained in burning high-volatile Western coals, the advantage of a pre-heated auxiliary air supply introduced over a fuel bed, and the advantage of a furnace and boiler setting adapted to the type of fuel used. They bear directly also on the question of steaming 'capacity' of coal for locomotives, the design and operation of gas producers for high-volatile fuels and also on the operation of coke ovens and of gas retorts.

"The results show further that certain bituminous coals of the interior and Rocky Mountain provinces give promise of good yields of by-products of coking, notably ammonia and high-candle-power gas, comparing favorably in these respects with the high grade coking coals of the Eastern province.

"They show also that inert, noncombustible material is present in the volatile products of different kinds of coal to an extent ranging from 1 to 15 per cent of the coal."

The bulletin will be of interest to fuel engineers, designers and builders of gas producers, gas and coke manufacturers, superintendents of power plants, railway master mechanics and those engaged in the suppression of smoke. The bulletin may be obtained by applying to the director of the Bureau of Mines, Washington, D. C.

FIG. 3.—DRYING FLOOR.

Acid-Proof Stoneware Tanks.

The making of acid-proof stoneware tanks, particularly in sizes larger than commonly used, has always been rather difficult as the piece is liable to crack or warp while drying and burning.

The shapes and capacities required are so varied that it is usually impracticable to attempt to carry "stock sizes," as a user would then have to take the nearest size to what he wants and make it do or would have to pay an additional charge for the variations. Most stoneware companies are prepared to furnish without delay special pieces according to blue-prints submitted. As most tanks and special pieces have to be made by hand, anyway, the consumer may as well have exactly what he needs, according to his specifications.

Each manufacturer has mixtures and methods which he does not care to divulge, but the process, in general, consists in tempering different clays, adjusting the proportions to give the body strength and ability to resist temperature changes as well as the action of acids. In the composition for chemical purposes almost pure silica plays an important rôle. The piece is shaped and allowed to dry for a few days, then the acid-proof glaze is applied in spray or soft paste form and further dried. It is then baked for several days at about 2500° Fahr. to insure a uniform vitreous body, which, even without the enamel, is a good resistor of acids. The kiln is then allowed to cool slowly and uniformly to insure proper annealing.

The adjoining illustrations give views of several parts of the plant of Mr. A. J. Weeks, of Akron, Ohio. Fig. 1 shows the manufacture of large storage tanks, tower pipe and receivers. Fig. 2 is the millroom, showing machines and a 200-gal. jar. Fig. 3 shows a drying floor of the works, and Fig. 4 the drying of chemical rings with partitions for chemical towers.

Among the tanks recently made by Mr. Weeks are several for the American Steel & Wire Company. They are intended for pickling wire in sulphuric acid during the galvanizing process, the strands of wire running under large depressor spools which rest on hubs fastened to the tanks. The tank dimensions are 4 ft. by 5 ft. by 14 in. Several of these tanks, as well as flux jars from 50 gal. to 500 gal. capacity and flux pitchers, have been furnished by Mr. Weeks to the steel industry.

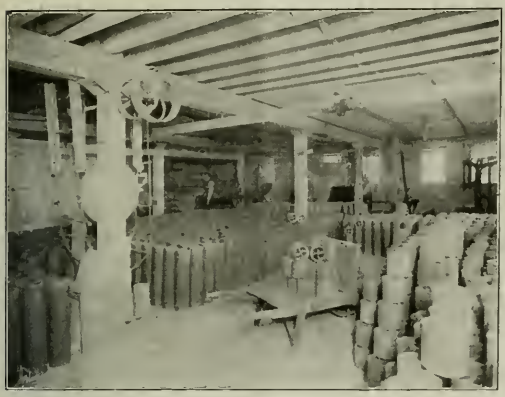

FIG. 4.—DRYING CHEMICAL RINGS.

Metallurgy in the Dutch East Indies.

Tin.

The government tin mines and smelters at the Island Banka produced during the year 1908-1909 (from Feb. 2, 1908, to Jan. 21, 1909, e. g., during 356 days) 204,190.70 pikols tin, against 191,930.22 pikols during 1907-1908 (1 pikol = 65 kg = 143 lb.). The laboring force consisted of 20,126 men (all Chinamen), against 19,106 in 1907-1908. The total production cost of 1 pikol tin, including freight to Tandjong Priok during the year 1908-1909, was Fl. 47.34. The average price made was Fl. 96.10. In the years 1907-1908 and 1906-1907 these figures were Fl. 44.61, Fl. 123.94, Fl. 35.67 and Fl. 128.26, respectively; 20,509.92 pikols impure tin had to be refined by liquating them. The cost of refining, including the smelting of the liquation residue and liquating the resulting tin, was Fl. 0.90 per pikol.

The Billiton Maatschappy, which operates on the Island Billiton, produced during the year 1908-1909 (ending April 1, 1909) 66,680 pikols tin, 29,900 of which were shipped to Singapore in the form of ore to be smelted there. The laboring force consisted of 13,608 men, e. g., the production per man was 4.9 pikol; 36,383.36 pikol were sold at Batavia at an average price of Fl. 93.29 per pikol. The price made at Singapore corresponded to Fl. 93.90 per Java pikol. The net profit amounted to Fl. 2,299,958.52, of which 5/8, e. g., Fl. 1,437,474.07, goes to the Dutch government. 144.79 pikols mixed wolframite tin ore concentrates were produced by concentrating ores mined the preceding year. This company contemplates at the present time to break with their old smelting method in van Vlaanderen shaft furnaces (these furnaces are also used at Banka) and to erect modern gas-fired reverberatory furnaces of the Schmidt & Desgraz type, which met with considerable success in European tin smelting.

The Singkep Tin Maatschappy operates on the Island Singkep (Lingga Archipel; Riouw). The production during the year from July 1, 1907, to June 30, 1908, was 6,441.18 pikols tin and 200 pikols wolframite. The ore is all shipped to Singapore and smelted there. The same was sold for $66.99 per Straits pikol, e. g., Fl. 95.78 per Java pikol of 65 kg against 134.13, Fl. last year's price. The net profit of the company was Fl. 65,855.05 against Fl. 199,835.305 last year's net profit. This company had to stop working its sea concession by Tiking as it did not pay the expense. A dredge is ordered which will be able to raise the ore from 40 ft. under the sea level.

Gold and Silver.

Lebong Donok Concession (Benkoelen, Sumatra) of the "Mynbouw Maatschappy Redjang, Lebong) during the year 1908 1058 m of tunnel were driven, increasing the available amount of ore to 319,249 metric tons at Dec. 31, 1908, with an assay value of 19.6 dwts. to the ton. The third level was opened up over the entire length and the South shaft had been sunk to 5 m below the third level as preliminary work for a fourth level. The number of stamps was increased to 70 and the number of rotating comminutating cylinders to 6, four of which were in operation: 12.82 per cent of the gold and 1.33 per cent of the silver contained in the ore was extracted by amalgamation and 84.99 per cent of the gold and 77.09 per cent of the silver contained in the sand were extracted by cyaniding the sand (40,454 tons of sand were treated). The slime plant treated 46,287 tons slime, 91.27 per cent of the gold and 76.68 per cent of the silver contained in the slimes were extracted by cyaniding.

Altogether 89,773 tons of ore containing 20.69 dwts. gold and 139.25 dwts. silver were treated, and 82,873.4 oz. gold and 471,743.1 oz. silver obtained, representing a value of Fl. 4,761,-000. The net profit over 1908 was Fl. 2,940,675.

Lebong Soelit Concession (Benkoelen, Sumatra) of the Mynbouw Maatschappy Ketaboen: 1790 m of tunnel were driven during 1908, increasing the available ore to 162,600 tons, with an average of 12.02 dwts. gold to the ton. The number of stamps was increased from 30 to 40. The stamp mill treated 33,029 tons of ore, the ball mill 6672 tons. The extraction by amalgamation was in the stamp mill 54 per cent and in the ball mill 20.4 per cent of the gold contained in the ore; 24,965 tons of sand were treated by cyaniding. The average extraction was 82.8 per cent of the gold and 59.3 per cent of the silver contained in the sand; 12,791 tons of slimes were treated by the cyanide process and an extraction of 76.9 per cent of the gold and 39.8 per cent of the silver contained in the slimes was obtained.

The total production was 26,177.7 oz. gold and 45,168.5 oz. silver, representing a value of Fl. 1,377,823. The total cost of production was Fl. 20.81 per ton of ore and the company's profit Fl. 541,500.

Concession "Totok" (Res. Menado) of the Mynbouw Maatschappy "Totok."—This company proceeded in 1908 working the deposits Haies, Liang and Tehosi by hydraulic mining. The coarse material goes to the stamp mills (two of 20 stamps each, which are direct-coupled with Pelton wheels); the fine material to concentration and amalgamation tables. The tailings of the stamp mill go to the dump; 70,334 tons of ore were treated and 318.8 kg of gold and 123 kg of silver, representing a value of Fl. 536,973, were extracted.

Concession "Paleleh" (Res. Menado) of the Mynbouw Maatschappy Paleleh: 2880 m. of tunnel and shaft were driven during 1908, increasing the amount of available ore to 87,000 tons. Thirty-six thousand nine hundred and eighty-five tons of ore, containing 10.07 grams of gold to the ton were treated and 260.158 grams of gold and 71.283 grams of silver were extracted by amalgamation, e.g., a gold extraction of 70.5 per cent. Thirty stamps were in continuous operation and 20 new ones were put up.

The Consolidated Mining & Smelting Company, operating copper and lead smelters at Trail, B. C., has, during the past year, erected an ore testing plant to determine the methods of economically treating ores, and to devise improvements on the present methods of concentrating ores from its mines.

NOTES

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in July, 1910, was 72. There were milled 1,865,318 tons of ore; 9826 stamps were in operation with an average duty of 7.07 tons per 24 hours, 170 tube mills were at work. Comparison with last year shows an increase in the tonnage milled, number of stamps and number of tube mills in operation. The yield for the month was 638,714 fine ounces, being the largest yield for the year 1910, and larger than for 1909 up to the same month.

The Colorado Iron Works Company, Denver, has sent us pamphlets Nos. 24, 26 and 27, describing, respectively, the Akins classifier, the Rothwell continuous thickener and filter, and the Akins-Rothwell agitating tank. These are devices which will interest those engaged in the cyanide industry.

The **International Acheson Graphite Company** has opened a Southern branch in Atlanta, Ga. The company is greatly pleased with the appreciation shown for its lubricants—Gredag, Oildag, Aquadag and its pure powered graphite—by Southern industries.

The **Wood Turpentine Association** held a meeting at Jacksonville, Fla., on Sept. 24, which was attended by representatives of a large number of plants engaged in the manufacture of high-grade wood spirits of turpentine and pine oil. The main object of the association at the present time is to improve the quality of their products and to establish uniform standards to which all the members of the association will bring their products before placing them on the market. The officers of the association are: Dr. J. E. Teeple, of New York, president, and Mr. A. J. Grant, of Orlando, Fla., secretary.

The **Noble Electric Steel Company** is reported to have plans prepared for the installation of five additional furnaces at its large smelter at Heroult, at a cost of about $300,000. The company is developing several chrome mines near its smelter. Since the demonstration in July of the electric process the furnace has been closed down owing to the Northern California Power Company being unable to supply electricity during the dry season to operate the plant.

The **Moler brick,** which has been mentioned repeatedly in this journal as a new refractory for certain metallurgical purposes (see especially vol. VII, p. 474), is being made by the Fredericksholms Tegl & Kalkvaerker in various shapes. This brick is used largely for partition walls in steel buildings because of its lightness. Messrs. Carl J. Michaelsen & Co., Ameliegade 26, Copenhagen, Denmark, are the sole exporters.

The **American Association of Commerce and Trade,** Berlin, Germany, is an American Chamber of Commerce, founded seven years ago by Americans and is run by Americans on American lines for the purpose of promoting American trade with Germany and German trade with the United States. It was organized especially for assisting American business firms to start branches in Germany. The organization with offices at the Equitable Building, 59-60 Friedrichstrasse, Berlin, has the largest and most complete American reading-room in Germany, where thirty daily American papers and 150 trade publications are available, besides all United States Government reports and statistics, all the directories of the leading American and German cities, all the principal telegraph codes. This reading-room is open to American business men and travelers visiting Berlin.

Remarkable Properties of Cobalt-Chromium Alloys.—While nickel-chromium alloys have come into extended use in recent years in form of a non-oxidizing resistance wire for electric furnace and electric heating purposes, considerable attention has recently been drawn to the sister alloy cobalt-chromium through the work of Mr Elwood Haynes, of the Haynes Automobile Company, of Kokomo, Ind. An alloy of 75 per cent cobalt and 25 chromium, properly prepared, is very tough and malleable and can be hammered out into exceedingly thin sheets at a bright red heat without cracking. Blades of this alloy have been used with good success for razors. Tensile tests have given an elastic limit of 70,000 lb., a tensile strength of 96,000 lb. and an elongation of 3 per cent. By charges of the proportions and addition of other elements alloys can be produced which are practically not attacked at all by nitric acid and which are sufficiently soft and malleable to be worked cold. This would seem to open a field for the use of this alloy in vessels and appliances for chemical work if the cost is not too high. The most useful property of the alloy is its extreme resistance to corrosion, in which respect it is said to be equal to gold and platinum. Sulphuric acid and hydrochloric acid attack the alloy to some extent, according to strength of acid and temperature. The alloy seems extremely suitable for the manufacture of small cutting instruments and especially of table cutlery, for which purpose the property of taking an edge comparable to that of tempered steel, together with its silver-steel-like color, its high luster and its non-corrodibility, form a combination of most desirable properties.

Electrometallurgy of Iron and Steel.—According to a report received by the London *Electrical Review* from Stockholm, the Swedish Electro-Metal Company has formed a new company by the registration of the Electro-Metal Company in London. The object is the exploitation of the inventions of the engineers, Messrs. Lindblad, Gronwall and Stalhane (well known from Dr. Haanel's last Canadian report), in England, the British Colonies and North America. The capital of the new company is 550,000 crowns ($150,000) and two Swedish engineers are at the head of the undertaking. It is added that the Swedish Electro-Metal Company has commissioned the Iron Industry Bank to erect works using the electro-metal furnace near Trollhatten and the bank has sanctioned the expenditure of 250,000 crowns ($67,000) for this purpose and has made an arrangement with the owners of the Trollhatten water power for the supply of 3000 hp. The Swedish iron and steel producers are also interesting themselves in this type of electric furnace. The Swedish company, which established a Norwegian Electro-Metal Company at Christiania a year ago, still possesses the patent rights for the inventions in other countries. Mr. Eyde (of Birkeland-Eyde fame), who has withdrawn from the management of the Norwegian Power Company for the utilization of the waterfalls in Telemarken, has been elected on the board of directors, but will otherwise devote himself to the interests of the Norwegian Electrochemical Company, in which he and some Norwegian capitalists have secured the majority of the shares. It is now the intention of Mr. Eyde to take up the manufacture of iron and steel by means of electric smelting, according to the patents of Mr. Gronwall, which were recently acquired for this purpose. The scheme aims at the erection, it is said, of works at all places on the coast where the conditions are favorable.

Experimental Testing Plant in Canada.—An announcement published in the *Canada Official Gazette* gives the following information regarding a national experimental testing plant, concerning which those desiring further information may address Dr. Eugene Haanel, Dept. of Mines, Ottawa, Ontario, Canada. "The Mines Branch of the Canada Department of Mines is installing in Ottawa an experimental plant for the concentration of magnetic ores. When completed the plant will consist of a standard Grondal concentrating unit, comprising an ore crusher, ball mill, and two Grondal magnetic separators operating in tandem, the capacity of the plant being from two tons to four tons of crude ore per hour. The plant is being installed for the purpose of testing low-grade magnetic iron ores and sands, with a view to proving their amenability to concentration by the Grondal wet system. Tests will be carried through for the determination of the following points: The extent of crushing required to raise the iron content of the crude to 65-70 per cent in the concentrate, the extent of crushing required to depress sulphur, phosphorus

or titanium contents to percentages acceptable to furnacemen; the number of tons of crude ore required to produce one ton of concentrate; the power consumed per ton of crude ore concentrated; the cubic feet of water used per ton of crude ore concentrated. It is expected that the plant will be ready for operation by the first week in November, 1910. Tests will be made free of charge on Canadian ores, but it is required that shipments shall be delivered, carriage paid, at the testing plant at Ottawa. Shipments of ore for testing purposes should be not less than 5 nor more than 20 tons, and it is particularly requested that shipments be made in sacks containing not more than 100 lb. of ore each."

PERSONAL.

Dr. **E. R. Buckley**, consulting engineer of the Federal Lead Company, Missouri, has gone to Idaho, where he will spend several weeks.

Mr. **Gelasio Caetani,** metallurgical engineer of San Francisco, Cal., has gone to Nicaragua on professional business.

Mr. **John Dern**, president of the Consolidated Mercur Gold Mines Company, Utah, was elected president of the American Mining Congress at its recent meeting in Los Angeles.

Mr. **E. Harms,** superintendent of the smelting works at Torreon, Mexico, has been visiting the metallurgical works of Colorado, Utah and Montana.

Dr. **Joseph A. Holmes,** Director of the Bureau of Mines, was in Denver recently en route to Washington after attendance upon the session of the American Mining Congress in Los Angeles.

Messrs. **Arthur Lakes** and **Arthur Lakes, Jr.,** announce that they have opened offices at 701-3 Gas & Electric Building, Denver, Col., for the practice of mining and metallurgical engineering.

Mr. **H. A. Megraw** has permanently located in San Luis Potosi, Mexico, where he is engaged in metallurgical practice.

Mr. **C. W. Merrill,** metallurgical engineer of San Francisco, has gone to London.

Sir **Alfred M. Mond,** of the Mond Nickel Company, recently inspected the company's property at Sudbury, Ontario.

Dr. **E. F. Northrup,** of the Leeds & Northrup Company, of Philadelphia has been appointed professor of physics at Princeton University.

Mr. **Irving E. Slater,** of Los Angeles, is on a professional trip into British Columbia.

Mr. **Maximilian Toch,** of New York City, specially known among the chemical fraternity by his work in the chemistry of paints and the publication of a treatise on the subject, which has become one of the standard books of this important branch, has been invited to lecture on the subject of new paint materials at the University College in London, under the auspices of the Paint and Varnish Society, on Nov. 3. He left New York on Oct. 25 by the steamship *Kronprinz Wilhelm.*

Dr. **Herbert G. Torrey** has resigned as chief assayer of the New York assay office, which position he has held for 37 years. He will devote his time to consulting metallurgical work.

The wedding is announced of Baron Dr. **Karl von Vietinghoff-Scheel,** editor of the *Chemiker Zeitung* of Berlin, Germany, and Miss **Zuleikha Hegeler,** of La Salle, Ill.

Digest of Electrochemical United States Patents Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ELECTRIC FURNACES (Continued).

685,042, Oct. 22, 1901, William T. Gibbs, of Buckingham, Canada.

Resistance type, enabling fusion and reduction to be effected in an atmosphere free from carbon-vapor or dust. The furnace comprises an inclosed firebrick chamber having an arched roof, beneath and adjacent to which is a thin horizontal carbon rod resistor, extending between massive carbon blocks set in iron casings in the side walls. The material to be heated is placed on the hearth of the furnace beneath and at some distance from the resistor, being heated by radiation and reflection. The chamber has charging and discharging openings and a gas-outlet. A modified furnace is shown employing three superposed resistors connected in series.

685,717, Oct. 29, 1901, Oscar Frolich, of Steglitz, near Berlin, Germany, assignor to Siemens & Halske, of Berlin, Germany.

Arc type, especially for the production of calcium carbide. The furnace comprises a crucible mounted on a truck, having brick sides and a conical bottom constituting one electrode, with a large central tap-hole. The other electrode is a depending carbon tube. Pipes lead from the upper end of this tube downward to four burners opening radially into a chamber beneath the crucible. Air is also introduced at these points to burn the carbon monoxide drawn from the smelting zone; or the gas may be burned by vertical tubes opening from the top of the hollow electrode. Beneath the tap-hole is a conical protected hearth to receive the carbide, which is gradually lowered as the product forms. For the production of calcium carbide the mixture of lime and carbon is finely pulverized and forms a gas-tight inclosure around the arc, directing the gas into the depending electrode.

685,965, Nov. 5, 1901, Charles Paul Bary, of Paris, France.

Resistance type, for the production of stannic acid from tin. The furnace comprises a base of agglomerated magnesia, containing a narrow groove joining transverse rectangular recesses receiving the electric terminals. In use, molten tin fills the groove and recesses, and the tin in the groove is heated to the point of volatilization. Upon the base is a dome-shaped cover, having a central slit for the escape of the vaporized tin and partitions which extend downward into the molten tin in the electrode-recesses. As the tin vapor escapes through the slit, it is oxidized by air and is carried on by an air blast through settling pipes into a dust-collecting chamber lined with cloth and having staggered vertical partitions.

686,525, Nov. 12, 1901, James H. Morley, of Springfield, Mass.

Arc type, especially for the production of calcium carbide. The smelting chamber is a vertical clay tube, resting upon a carbon-block electrode. The other electrode is a square carbon rod depending into the tube and having a central passage. Two chutes for the supply of mixed lime and carbon extend downward to opposite sides of the clay tube. A hood serves for the collection and removal of gases and dust. The depending electrode is given an intermittent upward movement and a reciprocating rotary movement, to assist in stoking the material. The escape of gases from the settling zone is said to induce a current of air through the depending electrode. Two furnaces are illustrated, carried on opposite ends of a horizontal revoluble arm, carbide in one being allowed to cool while the other is in operation. With a current of 400 amperes at 60 volts, the upper electrode is 4 in. square and is raised 8 in. per hour. The arc is started by a carbon pencil initially placed in a central recess in the end of the lower electrode.

686,551, Nov. 12, 1901, Albert Simon, of Bordeaux, France.

Electrolytic furnace, for the production of iron, manganese and ferro-manganese. The apparatus comprises a crucible, having a hearth of crushed ferro-manganese in which are imbedded horizontal carbon cathodes. The upper depending anode consists of four or more carbons, inclosed in an envelope of crushed (manganese) slag compressed with tar and resin. The anode is surrounded by a brick inclosure having openings and passages for the escape of gases. The electrolyte, consisting of lower oxides of manganese and oxides of iron dissolved in fused calcium fluoride, is gradually melted by heat from two small carbon-block resistors, initially placed between the carbon

cathodes and the lower end of the anode. The normal current-density at the anode is 40 to 50 watts per square centimeter. The reduced metal is tapped off.

686,830, Nov. 19, 1901, Marcus Ruthenburg, of Philadelphia, Pa.

Resistance type, for fusing fine iron ore into globules or lumps. The electrodes are tubes of cast iron, constituting the cores of electromagnets and having opposed outlets in their adjacent polar tips. The charge feeds downward through the cores and out into the space between the poles, where it is held magnetically as a bridge, being here melted by the current traversing it. The poles are water-jacketed. Carbon plates may extend transversely through the poles, constituting floors on which the charge feeds out into the gap. Modifications are illustrated, comprising horizontal electrodes containing screw-conveyors, and duplicate pairs of electrodes.

687,505, Nov. 26, 1901, Marcus Ruthenburg, of Philadelphia, Pa.

Resistance type, for fusing magnetite into lumps fitted for subsequent smelting, or for reducing a mixture of iron oxide, charcoal or coke and flux. One electrode consists of a vertical conical tube of cast iron, the lower discharge end of which is water-jacketed. Beneath the outlet is a horizontal cast-iron roller, constituting the other electrode. The material falls into the space between the lower end of the tube and the roller and is fritted or smelted by the current passing through it. The roller is gradually rotated, contact being made by a brush.

NEW BOOKS.

Arnold, J. O. On a fourth recalescence in steel. 12 pages, with illustrations and tables. Price, 6 pence. Sheffield: J. W. Northend.

Bain, H. Foster. More recent cyanide practice. Reprints of 77 articles by various authors, published between 1907 and 1910. 418 pages, illus. Price, $2. San Francisco: Mining and Scientific Press.

Bayley, W. S. Elementary Crystallography. 242 pages, illus. Price, $2. New York: McGraw-Hill Book Company.

Bowles, Oliver. Tables for the determination of common rocks. 64 pages. Price, 50 cents. New York: D. Van Nostrand & Company.

Burali-Forti, C., and Marcolongo, R. Eléments de Calcul Vectoriel Translated from the Italian into French by A. Lattes. 230 pages. Price, in paper cover, 8 francs (retail price in New York $2.50). Paris: A. Hermann et fils.

Goldschmidt, Karl. Ein Schulfall socialdemokratischer Verhetzung. Der Kampf um die Pensionskasse von Th. Goldschmidt in Essen. 36 pages. Halle a. S.: Wilhelm Knapp.

Noyes, William A. A Text Book of Organic Chemistry. Second edition. 540 pages. Price, $1.50 net. New York: Henry Holt & Company.

Peddle, John B. Construction of Graphical Charts. 100 pages, 58 illustrations. Price, $1.50 net. New York: McGraw-Hill Book Company.

Frazer, Persifor, and Brown, Amos Peaslee. Tables for the determination of minerals by physical properties ascertainable with the aid of a few field instruments based on the system of the late Professor-Doctor Albin Weisbach. Sixth edition, revised to date. Price $2.50. Philadelphia: J. B. Lippincott Co.

Hale, W. Jay. Calculations of general chemistry, with definitions, explanations and problems. Second edition revised. Price $1 net. New York: D. Van Nostrand & Co.

Parr, S. Wilson, and Wheeler, Wilfred Forrest. Weathering of coal. Series of 1909. 43 pages, illustrated. Paper gratis. Urbana, Ill.: University of Illinois.

Stillman, Thomas B. Engineering chemistry. A manual of quantitative chemical analysis for the use of students, chemists and engineers. Fourth edition. 744 pages with 174 illustrations. Price $5. Easton, Pa.: Chemical Publishing Co.

BOOK REVIEWS.

Principles of Metallurgy. By **Charles Herman Fulton, E.M.,** President and Professor of Metallurgy, South Dakota School of Mines. Pages xii + 544; 187 illustrations. Price, $5 net. New York: McGraw-Hill Book Company.

A book to be highly commended. It fills a "gap" which has been somewhat of a "thorn in the flesh" to English-speaking metallurgists—begging pardon for the mixed metaphor.

The subject is treated from the broad standpoint of accurate scientific information. It is not very elementary; it is intended for serious students well prepared in chemistry and physics. It is admirably adapted for use in technical schools and universities, and for the practical man who wants to be abreast of the times in the scientific basis of his profession.

The keynote of the work is accurate, quantitative information about metallurgical principles, along with a clear description of the nature of the principles themselves. Our only criticism is that the work lacks a little in *balance*, that some parts of it are rather brief while others are gone into quite extensively; but all that is given is well given.

It is a book which should be in the hands of every metallurgical student and in every metallurgist's library.

J. W. Richards.

* * *

The Journal of the Municipal School of Technology, Manchester. Vol. II. 298 pages. Manchester: The Municipal School of Technology.

This is "a record of investigations undertaken by members of the teaching staff of the several departments of the school during 1908." The papers included in this volume are of quite a diversified character and were presented before various societies. The following should be of special interest to the readers of this journal. W. Cramp and B. Hoyle, on the electric discharge and the production of nitric acid; by W. W. Haldane Gee, on electrolytic corrosion; by E. Knecht, on the mercerization of cotton yarns and on the action of oxalic acid on cellulose; and others.

* * *

A Pathfinder—Discovery, Invention and Industry. With Edward G. Acheson's portrait as frontispiece. 143 pages. New York: The Press Scrap Book.

This most interesting volume is the record of the life and work of Edward Goodrich Acheson. It starts with the quotation from Shakespeare: "There's a divinity that shapes our ends, rough-hew them how we will."

The largest part of the work is taken from Dr. Acheson's autobiography, originally written for his family and not intended for publication. This makes the volume the more valuable and interesting. Acheson's work in electrochemistry, carborundum and artificial graphite, aquadag and oildag has long been recognized as revolutionary and fundamental. But this volume brings out the man as man in all his characteristic traits. It brings a great modern inventive and engineering genius nearer to the knowledge and appreciation of the general public. To ambitious youth this life story of Acheson should be an inspiration.

After introductory chapters on the reward of the Perkin medal to Dr. Acheson and on his ancestors, there follow six chapters of the autobiography, dealing with his boyhood and youth in Western Pennsylvania, his association with Edison at Menlo Park, his work for the Edison interests in Europe, and carrying his life story up to the thirty-fifth year. Then follow three chapters dealing with Acheson's monumental work on carborundum, graphite, egyptianized clay, direct reduction of aluminium and silicon, production of siloxicon, lubricating graphite, aquadag and oildag. The story is full of interesting details not heretofore known. It is a masterly human document.

Metallurgical and Chemical Engineering

A Consolidation of Electrochemical and Metallurgical Industry and Iron and Steel Magazine.

Vol. VIII. New York, December, 1910 No. 12

PUBLISHED MONTHLY BY THE

Electrochemical Publishing Company

239 West 39th Street, New York.

Pittsburgh Office, 201 Oliver Building.
Western Office, 1510 Court Place, Denver, Colo.
European Office, Hastings House, Norfolk St., Strand, London, Eng.

J. W. Richards, Ph.D., President.
Curtis E. Whittlesey, Treasurer. E. F. Roeber, Secretary.
Telephone Call: 4700 Bryant.

E. F. Roeber, Ph.D., Editor.
H. C. Parmelee, Western Editor.

Yearly subscription price for United States, Mexico and United States dependencies, $2.00; for all other countries, $2.50 (European exchange, 10 shillings, 10 marks, 12.50 francs).

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Entered as Second-Class Matter at the Post Office at New York, N. Y., under the Act of Congress, March 3, 1879

New York, December, 1910.

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Monazite in Idaho.

It is possible that Idaho will contribute materially to the future domestic production of monazite, valuable for its thoria content and used in the manufacture of incandescent gas mantles. The monazite sands occur there in placer deposits as usual, and their present recovery is an interesting example of retreating material which already has been worked for another purpose. The placers formerly were worked for gold, and in that operation the monazite was concentrated in the fine tailing and went to waste with such gold as escaped. Now these tailings represent the most valuable and easily worked deposits of monazite, although the untouched placer fields will, of course, be worked both for gold and monazite. The practical assurance that monazite existed in commercial quantities in this district was one of the results of the tests on black sand made by the United States Geological Survey at Portland during the exposition in 1905.

The milling of monazite sand involves special treatment in wet dressing, subsequent electromagnetic treatment and cleaning. Screening to remove the coarse gravel is the first step, followed by table concentration and electromagnetic treatment to remove magnetic iron, titanic iron and garnet. The concentrate thus obtained is then cleaned electromagnetically, the product averaging about 95 per cent monazite. The monazite content of the Idaho placer deposits will average at least 0.1 per cent. Determinations of the average thoria content of the Idaho monazite show from 4.75 per cent to 5 per cent, which is a commercial grade. The recovery of gold from the sand should be a great aid in this development if it is to prove commercially successful.

The Condensation of Zinc Vapor.

In the abstracts of Current Metallurgical and Chemical Patents in this issue mention is made of three proposals for the improved metallurgical treatment of zinc ores. The metal zinc, as has often been pointed out in these columns, is among commercial metals "sui generis." In the first place, because of its extreme tendency to form soluble compounds it is a fact that geologically zinc presents divagations and disseminations that no other metal does. Thus there are complex zinc-lead ores, complex zinc-fluorspar ores, complex zinc-copper ores, complex zinc-barytes ores and complex zinc-iron ores, for in geology iron, like the poor, is always with us. Furthermore, in the reduction of the roasted and oxidized zinc concentrate in the spelter plant a marked metallurgical idiosyncracy is noted. This is the fact that the reduction of zinc oxide to the metal by carbon takes place only at a full 100 ° C. above the temperature at which the metal boils in a manner strictly analogous to the manner water boils. Thus zinc is divagatous chemically as well as geologically. This chemical peculiarity is quite natural and related to its geological wandering tendency. For, like those

ence whom society cannot quite understand, but of whom society is somewhat in awe, zinc is "unusual"—an adjective used in this connection with a slowly pronounced intonation and with the connotation and suggestion of a vague and vast incomprehensibility.

The attempt of Mr Frederick W. Gordon to harness by a blast furnace this "unusuality" is a grand and noble idea. Mr. Gordon is an old-time practical engineer and scientist, a man who has designed since 1870 over half a century of iron blast furnaces. He was, if not the innovator, at least a pioneer in the introduction of the water-cooled tuyere and the water-cooled "bosh-plate." Mr. Gordon 20 years ago was working out "heat-balance sheets," which was quite unusual at that time. It may be surmised, however, that his present attempt to attach a condenser to a blast furnace and to make a sort of glorified and magnified "zinc-stuhl" on the pattern of the zinc-stuhl of the medieval Harz "shaftofen" is the last attempt that ever will be made to reduce and condense zinc in a blast furnace. Zinc vapor does not seem to harmonize with heated air. For the zinc molecule above 500° C. has a strong affinity for the atoms of oxygen, even when those atoms are tied up securely to carbon or hydrogen as carbon dioxide or aqueous vapor. Thus it is difficult to see much chance for success for Mr. Gordon to do what Hempel and others have failed to achieve. While Mr. Gordon's argument as to the condensation of zinc in liquid form in the condenser attached to his blast furnace seems strictly correct if the conditions for such liquid condensation have once been established and are properly maintained in continuous operation, yet it would seem that there is grave danger that before these correct conditions can be established the shaft of Mr. Gordon's furnace will be clogged up with zinc oxide and his condenser will be clogged up with blue-powder and a miscellaneous mess. To be concise his proposal seems like the last charge of the Old Guard in the metallurgy of zinc. As such it is admirable and deserves close attention, whatever may be the practical outcome

Mr. Hiram W. Hixon is a younger man, but he is also noted as a doer of things in the business of making money out of ores. It was Mr. Hixon who over 20 years ago made copper converting practical at Aguas Calientes and Anaconda. Lately he has reduced to a practical basis for the Mond Nickel Company the converting of a low-grade copper-nickel matte to an "80-percent matte." In turning to zinc metallurgy, Mr. Hixon has with better judgment, we opine, taken to the electric furnace as the most promising metallurgical means to control the unusuality of zinc. Mr Hixon's ideas as disclosed in his current patents have to do with the eduction of zinc vapor from the furnace to the condenser, and the charging of the electric zinc furnace. The proposals by such an eminent man are worthy of consideration, especially as a veteran of a proved practical turn of mind applies himself to these practical sides of the problem.

The last of the three proposals is that of Mr Woolsey McA Johnson. Mr Johnson is a much younger man and less well known than either Mr Gordon or Mr Hixon, but he has a reputation for originality of conception and perseverance in proving the value of his conceptions. Mr Johnson comes of a family of engineers, for his grandfather, the late William J M Maine among other notable achievements rendered practical the caisson system of building sub-aqueous foundations. His proposal is to increase artificially by eduction the gases emanating from the reduction of low-grade zinc ores through a porous electrically heated carbon filter and thereby render innocuous such weak oxidizers as carbon dioxide, which would hinder the condensation. Mr. Johnson seems to take a philosophical view of the field and to unite the broad commercial side with the scientific and practical side of the question in a quite effective manner.

These three proposals and the attempts of such other workers as FitzGerald, de Laval, Cote Pierron and Queneau (all of which have been noticed before in this journal) give the impression that the atmosphere in the metallurgy of zinc is supersaturated. It may be safely hoped that some of these ideas will condense into commercial form directly. Indeed, the condensation of zinc vapor on a large scale as well as the condensation of ideas seems to be imminent. With labor conditions such as they are now, the 50-lb. zinc retort accordingly is an anachronism in this twentieth century. We predict that in this age of electricity zinc, the unusual metal, will be tamed by that kindly yet powerful force, the electrothermic effect of the electric current.

The Iron Industry's Insatiate Thirst for Capital.

The changing conditions of demand in the iron and steel industry do not remove its thirst for additional capital. In the great period of expansion in productive capacity which may be said to have started with the famine in material in 1899 enormous quantities of capital were invested. From the present outlook it appears that productive capacity has outrun demand and that it will be several years before there is need of any important increases for the purpose of averting a famine in material, but the demand for capital continues. As the investments of the past 10 years have been largely for the purpose of increasing capacity, the money has naturally been spent largely by mill managers and superintendents who were unusually busy with routine duties. They have done their best, but it would be presumptuous to infer that fully as good results, in the introduction of economies, have been attained as would have been the case if the money had been spent for the sole purpose of effecting economies in operation. There is no need for arguing that matter, for if the newer half of the industry represents a great step forward, the older half has the step still to make, since there has been little capacity definitely abandoned.

It is one of the interesting phases of the tendency in the affairs of men to find a balance, to effect a compensation, that while capital is most easily found when prices are high and profits large the necessity for investing capital in improvements is greatest when prices are low and profits small or entirely absent. If the need for expansion in capacity has been removed for a period of years, it follows that the need for improvements so as to reduce costs at many points is presented. The need of capital is not felt alone in the case of entirely new methods, like the use of the electric furnace for steel refining, the duplex process, or the dry air blast. Since the days of water-power the steam engine has been in use, and yet of late blooming mill engines have been introduced which are computed as likely to save their cost in three years, when compared with a type replaced and which nevertheless is not

generally regarded as antiquated. The tin-plate industry in the United States is not 20 years old, yet its methods of driving have undergone great changes. The pioneer American tin-plate manufacturers shrank from the nightmare not then uncommon in Wales of an engine direct connected to the hot mills, with a flywheel speeded up by gearing, but halted gearing the engine down to the mill and connecting it direct. Later the rope drive appeared to solve the problem and became almost general, but the latest tin-plate plant erected contains 12 tin mills direct connected to a twin-tandem compound Corliss engine, the rope-drive principle being relegated to the cold mills. With blast-furnace gas engines for blowing and electric power generation and exhaust-steam turbo-generators making their mark, but still introduced only to a limited extent, the sums which ought to be spent in new power-generating plant are evidently enormous in themselves.

Indeed, the needs for capital run the whole gamut of iron and steel operations. Starting with the coke, it is obvious that the retort oven must eventually supplant the beehive oven, but the moderate showing which the retort oven makes in production relative to the whole is due wholly to expansion; there has as yet been no supplanting. In ore, the early operations on the Mesabi range, where mining began just under the roots of the trees, are playing out one by one and must be replaced by operations involving stripping of 100 ft. or more, frequently involving expenditures running into seven figures for a single operation. In the blast furnace there has just come the thin-wall stack, and if half what is hoped of it proves true this curious device will command its introduction. In the fundamental methods of steel production the following figures, in three years of substantially equal totals, are amply suggestive:

Years.	Pig Iron.	Gross tons. Bessemer Steel.	Open-Hearth Steel.
1906	25,307,191	12,275,830	10,980,413
1907	25,781,361	11,667,549	11,549,736
1909	25,795,471	9,330,783	14,493,936

One Bessemer steel plant during the period covered, but another was built and the trend from Bessemer to open-hearth is too marked to promise full employment for Bessemer capacity unless, indeed, all steel-making capacity is required. We have made only the most cursory survey to suggest and by no means to catalog, but coming down at the end to the electric refining furnace we find a line of work in which even the barest outline of the requirements of the next few years cannot be drawn. If the steel industry sometimes enjoys a princely income, it must at all times spend like a prince.

Patent Law and Common Sense.

Discovery and invention are in the language of patent law to all intents and purposes synonymous. To evaluate in terms of use to society any successful invention is the first principle of patent equity. Of late there has come into greater and we believe more deserved prominence the doctrine that invention depends on the practical effort of the inventor to show and demonstrate the success of his process or device. Broadly speaking, the language of Paley, a commonsense English theologian of the eighteenth century, "He alone discovers who proves," exactly fits the case. An analysis of the essentials of an invention in the light of Paley's maxim is of use to the metallurgical or chemical engineer in whose industry the old order ever yieldeth, giving place to the new. The conception of an invention can come from either a flash of intuitive thought or from a long course of hard reasoning, or it can come from an artistic blending of the two, a combination of the purely feminine and the purely masculine character of mental action. Before he can act efficiently and successfully the inventor must see clearly and think correctly. By the building up of a structure of countless mental bricks, each one of which is a strong unit of the aggregate "see-think-and-act," by proceeding from the simple homogeneous unit to the production of the complex, harmonious, heterogeneous whole, the inventor reduces his invention to practice and his original and novel idea has become a useful reality. In short, the value of an invention is measured by the standard of William James' pragmatism—"Utility is the test of truth." The invention of an inventor is comparable to a play of a writer. The invention of a great inventor is comparable to a play of Shakespeare's.

Shakespeare was often accused of plagiarism, especially in the play of Hamlet, the greatest tragic history of a noble soul brought to ruin by a complex of circumstance and environment that the world has ever read. Just in the same way Edison has been accused of inventing "the perfectly obvious." Shakespeare wrote Hamlet, first adopting it from a play of Kyd, then he rewrote it and greatly improved it. Kyd had taken his plot and a good many of his scenes from a French historical novel. This had been taken in turn from the original Scandinavian story. Shakespeare made use of the forms of tragedy brought to perfection by Marlowe from the old miracle plays of the monks. Finally, he made use of many old phrases, rhymes and conceits, of many old tricks of stagecraft. Looked at in a narrow sense there is hardly anything new in it, and on this perverted view the partial truth "there is nothing new under the sun" has indeed a verisimilitude. But any literary critic, in fact, any man of literary taste and ordinary judgment, would dismiss at once the charge of plagiarism from Shakespeare. Shakespeare did not steal Hamlet.

Shakespeare is comparable to the great inventor, for he was a good actor as the great inventor is a fair mechanic; he was a good playwright with a keen sense of the public's wants, as the great inventor has a keen sense of the needs of the industry; he was fairly successful financially just as the higher type of an inventor like Bessemer, Siemens, Bell and Edison have had their share in the financial rewards. In short, they both unite the real with the ideal in a sensible sort of manner. Shakespeare used inferior and old materials and produced a superior article. Bessemer did precisely the same. Just as Shakespeare was original so is the inventor original who proves his invention by intelligently directed hard work. For as old Dean Paley said, "He alone discovers who proves." In the welter and toil of developing an invention the "merely obvious" is usually discovered with difficulty. In fact, what hindsight calls the obvious is usually the antonym of obvious. For, if the obvious was so eternally simple and infernally apparent, why did not someone else discover and prove its existence and value? This commonsense doctrine is seen in every one of the patent decisions of the Supreme Court of the United States. To be a bit disrespectful, law is, after all, only long-winded commonsense.

International Congress of Applied Chemistry of 1912.

In the following we give a list of the presidents and vice-presidents of the different sections of the International Congress of Applied Chemistry to be held in this country in 1912. In each one the president's name is first given. No officers have yet been elected for section 11a on law and legislation affecting chemical industry.

The first meeting of the presidents was held on Nov. 21 at the Waldorf Astoria, under the presidency of Dr. Wm. H. Nichols, president of the congress, and Prof. Morley, honorary president.

As will be seen, there are 23 sections and subsections.

1. Analytical Chemistry: Dr. W. F. Hillebrand, Washington, D. C., and Prof. H. P. Talbot, Boston, Mass.

2. Inorganic Chemistry: Prof. C. L. Parsons, Durham, N. H., and Prof. Charles Baskerville, New York City.

3a. Metallurgy and Mining: Prof. J. W. Richards, South Bethlehem, Pa., and Dr. J. B. F. Herreshoff, New York City.

3b. Explosives: Prof. C. E. Munroe, Washington, D. C.

3c. Silicate Industries: Dr. A. S. Cushman, Washington, D. C., and Dr. Karl Langenbeck, Boston, Mass.

4. Organic Chemistry: Prof. M. T. Bogert, New York City, and Prof. W. A. Noyes, Urbana, Ill.

4a. Colors: H. A. Metz, New York City, and Eugene Merz, New York City.

5a. Industry and Chemistry of Sugar: Dr. W. D. Horne, Yonkers, N. Y., and Dr. F. G. Wiechmann, New York City.

5b. Industry and Chemistry of India Rubber and Other Plastics: Dr. L. H. Baekeland, Yonkers, N. Y., and C. C. Goodrich, New York City.

5c. Fuels and Asphalt: Dr. David T. Day, Washington, D. C., and Dr. F. Schniewind, New York City.

5d. Fats, Fatty Oils and Soaps: David Wesson, New York City, and Dr. Martin H. Ittner, Jersey City, N. J.

6a. Starch, Cellulose and Paper: A. D. Little, Boston, Mass., and Dr. T. R. Wagner, Chicago, Ill.

6b. Fermentation: Dr. Francis Wyatt, New York City, and Dr. Robert Wahl.

7. Agricultural Chemistry: Dr. F. K. Cameron, Washington, D. C., and Prof. H. J. Wheeler, Kingston, R. I.

8a. Hygiene: Prof. W. T. Sedgwick, Boston, Mass., and Dr. L. P. Kinnicutt, Worcester, Mass.

8b. Pharmaceutical Chemistry: Prof. J. P. Remington, Philadelphia, Pa., and Prof. V. Coblentz, New York City.

8c. Bromatology: Dr. W. D. Bigelow, Washington, D. C., and Dr. A. L. Winton, Chicago, Ill.

8d. Physiological Chemistry and Pharmacology: Prof. J. J. Abel, Baltimore, Md., and Prof. W. J. Gies, New York City.

9. Photochemistry: Prof. W. D. Bancroft, Ithaca, N. Y., and R. J. Wallace, St. Louis, Mo.

10a. Electrochemistry: Prof W. H. Walker, Boston, Mass., and Prof. C. F. Burgess, Madison, Wis.

10b. Physical Chemistry: Dr. W. R. Whitney, Schenectady, N. Y., and Prof. Alexander Smith, Chicago, Ill.

11a. Law and Legislation Affecting Chemical Industry:

11b Political Economy and Conservation of Natural Resources: Prof. J. A. Holmes, Washington, D. C., and Prof. C. R. Van Hise, Madison, Wis.

New York Meeting of the American Institute of Chemical Engineers.

The third annual meeting of the American Institute of Chemical Engineers will be held in New York City from Dec. 7 to 10. The hotel headquarters are at the Hotel Astor.

The first meeting will be held on Wednesday, Dec. 7, at the Hotel Astor beginning at 9.30 a. m., with an address of welcome by Mr. John Purroy Mitchell, president of the Board of Aldermen of New York City. This will be followed by a business section and by the reading and discussion of the following papers:

Report of the Committee on Chemical Engineering Education, by Dr. F. W. Frerichs.

The Development of the Chemist as an Engineer, by Dr. Fred W. Atkinson.

The Training of Chemical Engineers Which Meets the Requirements of Manufacturers, by Prof. M. C. Whitaker, Columbia University.

The Fitzgibbons Boiler, by Mr. Jerome Alexander.

In the evening a meeting will be held at 8 p. m. at Columbia University, when the following papers will be presented:

Manufacture of Hydrated Lime, by Mr. Richard K. Meade.

Address of retiring president, by Dr. Charles F. McKenna, on The Evolution of Portland Cement Processes.

On Friday, Dec. 9, the morning meeting will be held at the Hotel Astor. After a business session the following papers will be presented:

Manufacture of Lignite Briquettes, by Mr. Henry S. Renaud.

Bleaching Oils with Fullers' Earth, by Mr. David Wesson.

Action of Fruit Juices on Metallic Containers, by Dr. Edward Gudeman.

Vacuum Distilling Apparatus, by Mr. Phillip B. Sadtler.

On the evening of Friday a joint meeting will be held with the New York Section of the American Chemical Society at the Chemists' Club, Prof. C. Baskerville presiding. The following papers will be read and discussed:

The Principles of Sewage Disposal, by Mr. George C. Whipple.

Sewage Disposal in Europe, by Dr. Rudolph Hering.

Sewage Disposal in New York, by Dr. G. A. Soper.

Sanitary Conditions in Their Relation to Water Supplies in the Vicinity of New York, by Mr. N. S. Hill, Jr.

The Unsolved Problems of Sewage Disposal, by Prof. Charles E. A. Winslow.

A quite attractive program of visits and excursions has been arranged.

On Wednesday afternoon, Dec. 7, the Marx & Rawolle Glycerine Refinery, Brooklyn, N. Y., will be visited, where Wood's multiple effect stills are in operation.

On Thursday morning an excursion will be made to the chamber acid plant of the Standard Oil Company at Bayonne, N. J. Through the courtesy of the Standard Oil Company the members of the Institute and their guests will be conveyed by a steam lighter to Constable Hook and return. The boat will leave Pier 4, North River, at 9:30 a. m. Luncheon will be taken at Hugot's Restaurant, New Brighton, S. I. In the afternoon of Thursday a visit will be paid to the candle house of the Pratt Works of the Standard Oil Company and to the grease works of the same company at Blissville, L. I. On the evening of Thursday a dinner will be held at the Hotel Astor.

A Symposium on the Fixation of Atmospheric Nitrogen.

The first meeting of the season of the New York section of the American Electrochemical Society (Mr. E. A. Sperry being the chairman), will be held at the Chemists' Club on Friday, December 16, at 8:15 p. m. The program is a symposium of papers on the fixation of atmospheric nitrogen in all its phases, including the production of nitric acid and nitrates by means of electric discharges through air, the production of calcium cyanamide from nitrogen and calcium carbide, and the synthesis of ammonia. There will be an introductory paper by Dr. E. F. Roeber, a paper by Mr. Charles S. Bradley, the pioneer of the industry in this country, a paper by Mr. John C. Clancy on the use of cyanamide in the metallurgy of gold, and discussions by various other speakers.

Electric Pig Iron.

In connection with our former numerous notes on the large experimental and semi-commercial work now being done in Sweden and Norway on the reduction of iron ore in the electric furnace, we take pleasure in reproducing a photograph of

ELECTRIC PIG IRON PLANT OF THE JERNKONTORET AT TROLLHÄTTAN, SWEDEN.

the outside of the Jernkontoret's electric furnace pig-iron plant at Trollhättan, Sweden, as it appeared under construction this summer. The building to the right is the charcoal storehouse.

According to reports just received the furnace was expected to make pig iron by Nov. 15.

The Western Metallurgical Field.

Cyanidation at the Alaska-Treadwell.

The ores mined on Douglas Island, Alaska, at the properties of the Alaska-Treadwell, Alaska-Mexican and Alaska-United companies, are very low grade, averaging during the year 1909-1910 $2.95 per ton. The ore is treated by stamp-amalgamation-concentration process, the concentrate being shipped by boat to Tacoma for smelting. As the concentrate represents about 42 per cent of the total annual yield, or about $875,000, its metallurgical treatment is a matter of concern. Until 1899 it was treated by chlorination on the company's ground, but subsequent to that time it was contracted to the Tacoma smelter, Tacoma, Wash., where it was sent in 1000-ton lots.

Several years ago the management of the company instituted experiments looking toward cyaniding the concentrate and erected a 10-ton plant, in which the concentrate was ground in a tube mill, run over an amalgamating table, thence to agitating tanks, where it was agitated by compressed air, and finally to filtering apparatus. Precipitation of the gold-bearing solution was effected by zinc dust. Tests on roasting the concentrate indicated that this procedure was unnecessary, as the unroasted material gave excellent extraction.

The success of the experiments is indicated by the fact that a cyanide plant and refinery are among the improvements contemplated at the property. This will eliminate the freight and treatment charges on the concentrate, which have been about $10 per ton, and as the probable cost of cyaniding will not exceed $5 per ton, a saving of $5 to $6 per ton will be gained by the company. This will amount to about $150,000 per annum, which is the proposed cost of the plant. The residue will still have a commercial value when a market is found for it.

Buying and Selling Ore.

It frequently has been intimated that the seller of ore is not well enough versed in the details of the buyer's end of the business to enable him to deal intelligently with the buyer and understand the reasons for treatment charges and schedules of settlement. At the recent meeting of the American Mining Congress Mr. S. E. Bretherton presented some excellent hints to ore shippers, in which he tried to explain the "give-and-take" points in the treatment of smelting ores. He explains the reasons for the manner of taking moisture samples; for penalizing certain elements in the ore and paying a premium for others; for deducting a percentage of lead and copper from the correct assay; for paying a reduced price for the metals; for the variation in treatment price of low and high grade ores and many other points which the shipper should know, and which, if he did know, would create a better feeling between him and the buyer.

Moisture samples usually include more than an average of the fine material, as the coarse contains moisture which would not be eliminated in the time taken for drying. The manner of obtaining an accurate sample of the ore for assay is one that is not always easily explained to the shipper. Hand sampling may be preferred by him, but his particular material may be far better suited to automatic mechanical sampling and vice versa. Again, he does not always understand the disposition of metallics in his ore, sometimes believing that he is defrauded out of their value, when the fact is that the metallics are assayed separately and their value calculated and added to the total value of the ore. In the matter of "salting," the risk is almost altogether on the side of the smelter, as it is far easier to increase the value of a lot by accidentally mixing a small quantity of high-grade ore or metallics, than it is to reduce the value of high-grade ore by the accidental admixture of low-grade material.

Metallurgical losses account for some of the misunderstood points in treatment charges and settlement schedules. High-grade gold ores suffer a greater metallurgical loss than low-grade and the treatment charge is therefore higher. On the other hand, low-grade ores offer less gold on which to make a profit, and the price paid per ounce of gold is lower than with high-grade ores. Silver and lead are sources of large metallurgical loss, not only in smelting, but in the subsequent sale of the product, such as matte, if the smelter has not his own means for refining.

In addition to the losses on the metals, the smelter has to take into consideration the other constituents of the ore, such as silica, barium sulphate, zinc, alumina and arsenic, on which he must ask the seller to share with him the expense attached to putting these materials through the furnace. On the other hand, there are such substances as iron, manganese, lime and magnesia which the smelter is glad to credit to the seller if his ore contains them, as otherwise it might be necessary to purchase such fluxes at a higher cost.

Zinc.

The unquestioned value of the recently discovered deposits of zinc carbonate and silicate at Leadville has directed attention to the inexplicable reason for overlooking them in years gone by. Noted geologists, such as Emmons and Irving, have given the matter their attention, but without offering any satisfactory reason for the oversight, except that at the time the deposits were first opened it was probable that the miner was so intent upon the discovery of gold and silver that he neither regarded the zinc minerals as having value, nor did he have them analyzed to ascertain such a possibility.

Be that as it may, the deposits now known to exist and from which shipments have already been made, are of sufficient importance to warrant the Colorado School of Mines preparing a special bulletin on the genesis and extent of the deposits, the identification of the minerals and the assay of the ore. In view of the specially favorable market for zinc at the present time the mining and reduction of these ores must be of great benefit to the industry.

Colorado.

Mention has been made previously in these columns of the interest being aroused in the rejuvenation of the mining and

metallurgical industries in this State. Recently the Colorado School of Mines issued its *Quarterly* for October, 1910, and devoted the entire issue to an exposition by the faculty of the scope and progress of the mining industry in Colorado.

Special mention is made of the probable success of new processes of ore treatment and their undoubted effect on the production of low-grade and refractory ores that are not being mined at present.

Bearing on the same subject, Mr. Franklin Guiterman, Mr. T. L. Wilkinson, Dr. Victor C. Alderson and Dr. R. D. George recently gave their views on the situation at a luncheon of the Denver Chamber of Commerce. Mr. Guiterman recited the unquestioned decrease in production of smelting ores in the past decade, but found reason to believe that Colorado still offers a profitable field for mining and metallurgical ventures when guided by competent management and professional advice. Mr. Wilkinson reviewed the increase of other methods of ore treatment, including concentration, showing that in 1909 there were produced 165,000 tons of concentrates valued at about $5,000,000, as compared with about 525,000 tons of smelting ores valued at only $6,000,000. He found reason to believe that well-engineered plants for ore dressing would prove successful in this State, but laid stress on the need of employing the best engineering talent obtainable.

Dr. Alderson, president of the School of Mines, spoke briefly of the great value which the school's new ore-testing plant would be to the State in determining the best methods of ore treatment and in avoiding the unwise expenditure of such enormous sums of money as hitherto have been expended on poorly designed mills and ill-adapted processes. Dr. George, as State geologist, spoke strongly on the possibilities of reopening abandoned and partly abandoned camps in the State, where modern methods would accomplish the success which was not possible in former years. Altogether, an increasing feeling of optimism prevails as a result of the investigation of the actual conditions in the State and opportunities are seen which in late years have escaped notice.

Analysis of Zinc Retort Residues.

At the November meeting of the Colorado Section of Western chemists and metallurgists the analysis of zinc retort residues was under discussion. This material offers some special obstacles to the usual methods of analysis on account of the large quantity of carbonaceous matter contained and its general refractory nature. Mr J. C. Evans outlined the procedure which he had developed in the course of his experience. Silica is determined by making an insoluble residue which is subsequently fused with sodium peroxide, the melt being dissolved in HCl and evaporated in the usual manner. Iron is determined in the combined filtrates from the silica determination, being oxidized by HNO_3, precipitated by NH_4OH, filtered, dissolved in HCl and titrated in the usual manner. If Al_2O_3 is required, it may be determined in one-half of the solution containing the iron before titrating for that element. Lime is determined in the filtrates from the iron and alumina precipitations. Zinc requires special procedure. The sample is disintegrated in a platinum dish with HF and HCl and the former removed by evaporation. The chlorides are taken up with a little HCl, 2 cc of HNO, are added and the solution evaporated to dryness. Nitric acid and KClO, are then added and the solution again evaporated to dryness, from which point the assay is continued as usual. Wet lead and copper are made on the same sample, attacking the material with chlorate and nitric acid and several additions of HCl to decompose the chlorates. Finally 10 cc H_2SO_4 are added and evaporated to fumes. The lead and copper are then determined in the usual manner. For the gold assay the material is first roasted until glowing ceases.

The silver assay is best made by taking such quantity of the ore as will reduce the proper size button, using an excess litharge flux and no flour or nitre

The Non-Ferrous Metal Market.

The improved condition and advancing market of spelter was the only feature of the metal market for November. There was a slight advance in copper, but there was no feeling of certainty regarding the market and transactions, while fairly large in volume, were confined almost altogether to the needs of consumers. In general the market has been quiet.

Copper.—An advance to 13 cents for Lake copper occurred early in the month, but the differential between Lake and electrolytic remained smaller than usual. The copper statistics were variously construed, but in general may be said to have been favorable as the total stocks for Europe and America were reduced by over 22,000,000 lb. The average quotations were 12⅞ cents to 13 cents for Lake and 12.75 cents to 12.80 cents for electrolytic.

Lead.—The market has been unchanged and steady at 4.40 cents for New York and 4.25 cents to 4.30 cents for St. Louis The volume of business has been satisfactory.

Tin.—Trade in this metal was reduced to a minimum both at home and abroad, being confined to necessities. November tin was available at slightly lower quotations than for October, being about 36¼ cents.

Spelter.—In view of the demand for this metal in the face of a probable shortage in supply, as mentioned last month, the quotations have advanced. Producers are selling sparingly and only small quantities at any one price. New York quotations were 5.95 cents to 6 cents; St. Louis, 5.80 cents to 5.85 cents.

Other Metals.—There was a fair demand for other base metals during November. Aluminium advanced to 23 cents per pound; antimony showed a light demand at prices ranging from 7⅛ cents to 8 cents for various grades; quicksilver experienced a good business at $45 per flask of 75 lb. for large lots, with higher prices for small.

The Iron and Steel Market.

The November elections were unfavorable to the iron and steel market, not so much on account of a prospect being introduced that the tariff will be reduced on iron and steel products, as by reason of the general unsettlement to business which such a political change is generally assumed to involve. The iron industry, long held to be the barometer of general trade, depends even more than it did a few years ago upon the general business activity of the country. There is a closer relation than ever before between mill activity and general activity. Less iron and steel is going into large projects for which financial arrangements are made and against which contracts are placed far in advance of the completion of the work, and much more is going into the ordinary channels of daily consumption, being bought, made and delivered in a hand-to-mouth sort of way, so that iron and steel activity is more closely aligned with the ordinary business activities of the country than ever before. As to the tariff, it is recognized that nothing can be started for a year, and so much else can happen in that year that the iron trade is thinking very little of the tariff

The iron and steel industry as a whole operated during November at about 70 per cent of its total capacity Last February it was, roundly speaking, operating at about 95 per cent of capacity, so that it has been losing an average of about 3 per cent a month The progression has been continuous, but not uniform, there being a very slight contraction in operations in March and April, a rapid contraction in May and June, slower contraction again in July and August, and more rapid contraction again in the past three months.

Prices suffered little change in November The mills became more convinced than ever that no important gain could be obtained by reducing prices and no important contracts were in negotiation which would tempt cutting. Most finished steel products are subject to cutting at times, or by certain interests,

but, on the whole, there is little more of this than there was one month or two months ago.

There is no prospect of any improvement in general demand in the next two months. Production promises some slight further decrease, as the accumulation of old orders for heavy steel products is further reduced. For several months the current buying has been chiefly in the light products, the output of heavy products being based chiefly upon old orders.

The "unfilled orders" of the United States Steel Corporation showed a decrease during October of 286,153 tons to 2,871,949 tons. This total is the smallest reported, but it is stated that at one time the inter-company orders were included in the total and at some intermediate time this practice was discontinued so that if comparisons were made on the same basis the tonnage last reported would not be the lowest. This, however, is of relatively little consequence, for the reason that on any basis of compilation the present unfilled orders show the lowest relation to existing capacity which has ever occurred. The continued decrease in unfilled orders has been due to the working off of old contracts for heavy products, like rails, line pipe, fabricated steel, etc., and with this loss tapering off and production decreased, the November loss, which will be reported Dec. 10, promises to be small.

Books were opened early in November for rail orders for 1911 delivery, the Erie's order for 33,000 tons being the first placed. This has been followed by some other orders, and inquiries are pending, but, on the whole, the railroads are not ordering large tonnages. The railroads do not contemplate any extensive track laying next year and, some popular notions to the contrary notwithstanding, the annual wear of rails in service does not create a large demand. Available data are somewhat scant, but they indicate that the annual requirements, due merely to the wear of standard sections, aggregate in the neighborhood of 1,000,000 tons.

The Pittsburgh Steel Company will build two 400-ton blast furnaces at Monessen, Pa., where it operates eight 60-ton basic open-hearth furnaces, blooming, billet and rod mills, 200 wire-drawing blocks, 200 wire-nail machines and barb-wire and field-fencing departments. It operates also a hoop and cotton-tie mill at Glassport, Pa., nearby. The company has had outstanding $6,000,000 of stock and $3,750,000 of bonds, made up of $3,250,000 out of an issue of $4,000,000 in 1907 and $500,000 out of an issue of $1,000,000 in 1908, both at 6 per cent. It issues to the holders of the $6,000,000 an additional $1,000,000, for which $1,000,000 cash is paid into the treasury. To Speyer & Company, associated with the Corn Exchange Bank, it issues $7,000,000 of 7 per cent cumulative preferred stock in return for which Speyer & Company are to secure the outstanding bonds for retirement and pay a sum, not stated, into the treasury. For exchange purposes the stock is rated at 99, the first bonds at 110 and the second bonds at 102½, interest being allowed to Dec. 1 and dividends accruing from that date, so that about $4,150,000 preferred stock is involved in the exchange.

Pig Iron.

There has been a little more buying of pig iron, the needs of consumers increasing after an abstention of so many months from active buying. The buying is of a very conservative character and prices have, on the whole, continued to show a weakening tendency, although the actual declines have been light and there have been two cases in which prices have become firmer. The Buffalo furnaces, having become well sold up at $14 and $14.25 at furnace for No. 2 X, have advanced their asking price to $14.50, while basic iron in the valleys is quoted at not under $13.25, furnace, against $13 done in October. The Birmingham market stands at a minimum of $11, Birmingham, this price being made freely for early delivery, but by only a few producers for first quarter, the stand for $11.50 for next year not having been entirely abandoned. In the Philadelphia market $14.75, delivered, has been done on basic iron. The valley market, Pittsburgh delivered prices being 90 cents higher, stands as follows: Bessemer, $15; basic, $13.25 to $13.50; No. 2 foundry, $13.75 to $14; forge, $13 to $13.25; malleable, $13.25 to $13.50.

Steel.

The mills have taken a more decided stand in the matter of basing prices upon Pittsburgh in the case of Bessemer billets and sheet bars. The market had become decidedly unsettled by the naming of prices f.o.b. maker's mill, which led to sharp competition in delivered prices. Bessemer billets are quotable at $23, Pittsburgh, and sheet bars at $24.50, Pittsburgh, with full freight from Pittsburgh to destination. Similar uniformity has not been established in open-hearth steel, owing possibly to the greater number of producers. About the middle of the month a sale of 700 tons of open-hearth sheet bars was made by a Pittsburgh mill at $24 delivered Wheeling, the freight from Pittsburgh being $1. This is regarded as below the present market, which can be quoted at about $23.50, Pittsburgh, for billets and $24, Pittsburgh, for sheet bars. It is interesting to note that the consumer who had the refusal of the open-hearth sheet bars at $24, delivered, offered the order in preference to Bessemer steel producers at the same price, but could not place it. In either sheets or tin plates the preference of the mill is to roll Bessemer steel, open-hearth being used only when the customer requires the material or perhaps when, as in the present instance, a saving can be effected.

Finished Products.

Plates and shapes are being occasionally shaded by about a dollar a ton, and this is true also of wire products. Merchant steel bars are firmer than before. With the opening of order books for next year's delivery the rail mills began to quote per pound, converting their former prices of $28 for Bessemer and $30 for open-hearth into the corresponding figures per pound. It was desired to help discourage the tendency to compare billet and rail prices as, particularly with the more stringent specifications and inspection now in vogue, there is a wide difference in the cost of manufacture. Regular prices are given below, at Pittsburgh, delivered prices being at the Pittsburgh base plus rail freight to destination unless otherwise noted:

Rails, 1.25 cents for Bessemer, 1.34 cents for open-hearth, at mill, except Colorado.

Plates, 1.40 cents for tank quality.

Shapes, 1.40 cents for beams and channels, 3 in. to 15 in., zees and angles 2 in. x 3 in. and larger.

Steel bars, 1.40 cents, base; iron bars, 1.35 cents to 1.40 cents, Pittsburgh; 1.30 cents to 1.32½ cents, Cleveland; 1.30 cents to 1.35 cents, Chicago.

Wire nails, $1.70 per keg; plain wire, 1.50 cents; painted barb wire, 1.70 cents; galvanized barb wire, 2 cents.

Black sheets, 28-gage, 2.15 cents to 2.20 cents; galvanized, 3.15 cents to 3.20 cents; blue annealed, 10-gage, 1.60 cents to 1.65 cents; painted corrugated roofing, $1.55 to $1.60 per square; galvanized, $2.75 to $2.80.

Tin plates, $3.60 for 100-lb. cokes

CORRESPONDENCE

Defects in Refractory Brick.

To the Editor of Metallurgical and Chemical Engineering:

SIR—In METALLURGICAL AND CHEMICAL ENGINEERING for September appeared an article by the writer on the above subject which seems to have been misunderstood. Before explaining what I said I would like to call attention to the difficulty of reproducing any kind of clay work. We read in the history of pottery of one Bernard Palissy, who attempted to reproduce a white enamel which came to his notice. After spending all his fortune and the fortunes of a number of his friends he finally succeeded at the end of some twenty or thirty years.

To those who have had experience in clay work it is not sur-

prising that he should spend so much time and money at it. The surprising thing is that he succeeded at all. An ordinary man certainly would have failed, but Palissy was a beacon light in the history of pottery; a man who did many extraordinary things in his line, but this achievement probably excels all others.

Another great man who devoted his life to pottery was Seger; but Seger, with all his cunning and the financial backing of the German Government, failed to produce a certain pink glaze after many years of patient labor. Again, we read of a noted firm of English potters who had an order for a perfect loving cup. The price was no object, but the cup must be perfect. After spending $1,500 and producing many exquisite cups the manager of the plant decided that the perfect piece of pottery was yet to be made. This was not an attempt to reproduce anything, but simply to make a perfect piece of high-grade ware, which is much simpler.

There recently appeared in some of the magazines and trade journals a note to the effect that the X. Y. Company, after some years of experimenting, had finally succeeded in producing a fair imitation of a certain pottery. This company had spent thousands of dollars and considered itself well repaid when it had produced "a fair imitation" of the pottery in question.

Coming a little closer to refractories, let us consider the brick manufacturers, to whose skill we owe much of the beauty and worth of our cities. How many of them will attempt to match the color of a brick? To be sure, there are certain stock colors that they can and do match with great accuracy, but the experimental kilns are ever producing colors that differ radically from the standard. Sometimes these odd colors are of rare beauty and some of them are reproduced; but a large percentage of our most successful brick makers would no more guarantee to match all of them than a bird fancier would guarantee to match a white blackbird. Can any and all of these rare colors be reproduced? Certainly they can. So can the pink glaze on which Professor Seger spent so much time and money, but many centuries have failed to produce the man to do it.

But there are in the United States to-day manufacturers who can reproduce anything in the line of refractories. Of course, if the sample is made of material that has been carefully hand-picked the consumer must be willing to pay art pottery prices, but if it is made of material that can be produced commercially, then any refractory can be reproduced with certainty and at a price that is not prohibitive.

With this introduction I will try to explain what I said or intended to say in your September issue. I stated that I had once visited a plant where the methods used were not the best, and, on asking the manager about it, was told that the methods in question were the very best for which the consumer would pay, that they were so good, in fact, that all he was getting out of it was the consciousness of giving one hundred cents worth for a dollar and the possibility of educating his customer to the intrinsic value and actual cost of a good refractory.

While speaking of things that have come under my observation let me relate another. One of the plants of the United States Steel Company was having trouble with the refractories used. A manufacturer studied their case and made a refractory to suit their need. The brick in question were 4½ in. x 4½ in. x 10½ in. and were furnished at $40 per 1000. They were made with great care and the manufacturer kept his finger on the pulse of that furnace, if I may use such a figure, ready to change the prescription at any time if it should become necessary. It was not necessary, however. The steel makers expressed themselves as well pleased with the brick, but after a time notified the manufacturer that they could get brick of that size for less money, and as he could not meet the reduced price without sacrificing quality they are now using a cheaper brick made by another firm.

A standard size brick was made here some years ago that sold at the factory for $70 per 1000. They cost in Leadville, where the writer used them, $96 per 1000, and they were worth it. An average firebrick could have been had at that time for $30 and when the Leadville smelters came into the control of non-resident stockholders the cheaper brick were used. Some of these same stockholders are now bewailing the good old days when they could buy good refractories.

Within the last month the writer saw a pile of fire clay that had been weathering for five years. This clay was pronounced by the United States Geological Survey one of the best in the United States just as it was, but the manufacturer, wishing to be ready for anything, is perfecting this lot. The weathering is now about completed, but before the clay is used all impurities will be carefully picked out. Will the consumer appreciate this or will he think the manufacturer is trying to rob him when he asks a moderate return for all this trouble and expense?

Now I do not wish to be understood to say that any manufacturer can and will produce any refractory that is desired. The manufacturers may be divided into three classes. The first class is not competent to produce anything better than an ordinary fire brick. This class will seek other lines when the market demands something better than the present average, which average, however, was made to meet a demand. The second class is composed of business men who prefer to handle the best seller. The product of this class will keep pace with the demand. The third class consists of a few men who follow art for art's sake, who spend their spare time—and money—perfecting a product for which they hope some day to find a buyer. May their hope not be long deferred.

Clay working is not a pure science. There is much empiricism in the practice of the most scientific man that handles clay in any form, whether he be a potter, a maker of refractories or a chemist. The metallurgist with his work on slimes and the Portland cement maker have thrown much light on the subject, and in spite of the fact that for more than forty centuries the ceramist has not been sure of his foundation, he hopes the time is not far distant when he will be able to reproduce, with absolute certainty, anything, whether it be the most delicate glaze, the rarest building brick, the most refractory fire brick or the toughest abrasive. He has not yet reached that degree of perfection; neither has the paint maker, the Portland cement man, the metallurgist nor a host of others.

A short time ago the writer stood on top of an iron furnace. Carloads of ore, flux and fuel were dumped into the furnace in rapid succession. Rivers of molten iron flowed out a hundred feet below. Thousands of tons of raw material were changed into finished products in this plant every day with scarcely a touch of human hands. A few minutes' walk took us to one of the best zinc smelters in the world. Here we saw men laboriously roasting ore by hand, after which it was mixed with carefully prepared fuel and a few shovelfuls put into an expensive clay retort, placed in the furnace and, at the expense of more time and fuel, a little zinc was distilled off. In spite of this outlay of labor and capital the output was only a little over 85 per cent. As we looked across the valley at the majestic stacks of the iron works it seemed as if the wheels of progress had suddenly turned backward many centuries. To the zinc men, however, the outlook was different; they claimed that their output was the greatest that had ever been produced. They did not exalt themselves, however, above their fellows, but admitted that their record-breaking output was due to the superior refractories which they were using.

Returning to refractories, the writer is still optimistic enough to believe that the time is near when the consumer and manufacturer will get together with a full knowledge of the needs of the one and the difficulties of the other, much to the benefit of both.

Golden, Cal. JOHN C. BAILAR.

Some Calculations Pertaining to the Reactions Occurring in a Blast Furnace.

BY W. D. BROWN.

By a comparison of the volumes of the constituent gases of blast-furnace gas much is learned that is not readily seen when, as is usually the case, calculations are based on the weights of the constituents of the gas. Moreover, the calculations are much simplified if based on the volumes or percentage volumes of the constituent gases. This follows from Avagadro's law that a given volume of any gas contains the same number of molecules as the same volume of any other gas; and, since reactions of gases are between molecules, it follows that the volumes of gases entering any reaction bear a simple ratio to each other. For example, we have the union of carbon monoxide and oxygen to form carbon dioxide, $2CO + O_2 = 2CO_2$. From this it is seen that two molecules or volumes of carbon monoxide combine with one molecule or volume of oxygen to form two molecules or volumes of carbon dioxide.

Extent of Oxidation of the Carbon in the Furnace.

It is believed that a quantitative investigation as to the amount of carbon existing in the gas as carbon dioxide may throw some light on the reactions occurring in a blast furnace.

It is a well-known fact that any carbon dioxide produced at the tuyeres is quickly reduced to carbon monoxide, part of which, on rising through the charge in the furnace, reduces iron oxide, the final reaction being as follows: $Fe_2O_3 + 3CO = 2Fe + 3CO_2$. From this it is derived that 111.7 tons of iron are reduced by the carbon monoxide containing 36 tons of carbon, or that 1 ton of iron requires for its reduction the carbon monoxide containing 0.322 ton of carbon; the carbon dioxide formed likewise contains 0.322 ton of carbon. Pig iron contains about 93 per cent of iron and the carbon dioxide formed from the reduction of the iron in 1 ton of pig iron contains 0.300 ton of carbon. Iron may also be reduced by solid carbon, the carbon oxidizing to carbon monoxide. This carbon monoxide may or may not reduce other particles of iron, the iron reduced by carbon monoxide being here considered regardless of the source of the carbon monoxide.

The carbon dioxide formed during the reduction of iron by carbon monoxide is, however, not the only source of carbon dioxide in the gas. It is formed when the carbonates of the charge are decomposed, but most of it is reduced by solid carbon to carbon monoxide ($CO_2 + C = 2CO$). There is yet another source of carbon dioxide in the gas; it is formed when carbon is deposited, the reaction being $2CO = C + CO_2$.

It would seem that with these three uncertain sources of carbon dioxide little can be learned. Yet it is readily seen that if the deposited carbon reduces iron the result both as to the resultant gas and as to the amount of heat liberated is the same as if the iron were reduced by carbon monoxide. This is shown by the following reactions:

Reduction by CO.	Carbon Deposition and Reduction by Deposited Carbon.
$2Fe_2O_3 + 6CO = 4Fe + 6CO_2$	$6CO = 3CO_2 + 3C.$ $2Fe_2O_3 + 3C + 3CO_2 = 4Fe + 6CO_2.$

The final products are the same in both cases. In fact, if some of the deposited carbon impregnates the pig, or is burnt before the tuyeres, or reduces carbon dioxide from the flux to carbon monoxide, or decomposes water, and the same quantity of solid carbon of the coke reduces iron, the result is the same as if iron were reduced by carbon monoxide. In reality no matter what the source of carbon dioxide it shows the extent of the oxidation.

By a comparison of the carbon contained in the carbon dioxide formed per ton of pig iron with the amount required if reduction is by carbon monoxide we may learn with what degree the reduction is made by carbon monoxide. The total compounds of carbon found in the top gas are derived from the total carbon gasified and their percentage by volume represents this weight of carbon. This, after deducting the carbon lost in flue dust and in "slips," is the sum of the carbon in coke (c) and in limestone and other carbonates (s), less that entering the pig (p), and may be represented by $(c + s - p)$. Now, since each molecule of each of the compounds of carbon found in blast-furnace gas contains but one atom of carbon and any given volume of any of the carbon compounds contains the same weight of carbon as does the same volume of any other carbon compound found in blast-furnace gas, it follows that the weights of carbon gasified are in proportion to the volumes of the resultant compounds of carbon. Therefore, *the ratio of carbon dioxide to the sum of the carbon compounds is the ratio of carbon in carbon dioxide to the carbon gasified.*

For years the ratio of carbon dioxide to carbon monoxide has been looked upon as an indication of the reduciton going on in a furnace. A more definite indication of the working of the furnace is the ratio of carbon dioxide to the sum of the carbon compounds. This, the percentages by volume of the gases being represented by their formulas, is expressed as $\frac{CO_2}{CO_2 + CO + CH_4}$ and, as shown in the preceding paragraph, is the ratio of carbon existing as dioxide, the higher state of oxidation, to the carbon gasified; and from it we may determine the actual weight of carbon existing as dioxide per ton of pig iron by multiplying by the weight of carbon gasified $(c + s - p)$. It is readily seen that the greater the amount of carbon oxidized to carbon dioxide the more heat is liberated in the furnace, and, what is of more importance, the more carbon has descended in the furnace to unite eventually with air before the tuyeres, for the presence of carbon dioxide means that carbon of the coke has escaped oxidation by carbon dioxide or iron oxide, or carbon has been added to the charge by deposition. By a comparison of the carbon existing as dioxide, as found above, with the carbon required as carbon monoxide to reduce the iron in a ton of pig iron, we may arrive at a concrete knowledge of the reduction. This is best shown by example.

Example I.

Carbon dioxide in gas (CO_2)	12.3 per cent
Sum of carbon dioxide, carbon monoxide and methane ($CO_2 + CO + CH_4$)	38.3 per cent
Ratio of carbon in higher state of oxidation to carbon gasified $\frac{CO_2}{CO_2 + CO + CH_4}$	.3212
Carbon gasified per ton of pig iron	.898 ton
Carbon in higher state of oxidation per ton pig iron $.898 \times .3212$	.288 ton
Carbon dioxide formed from reduction by carbon monoxide of the iron in 1 ton of pig iron	.300 ton

It is seen here that the carbon existing as dioxide is sufficient to have reduced by carbon monoxide but 96 per cent of the iron. This does not show that the 96 per cent of the iron was actually reduced by carbon monoxide, but that more of it could not have been so reduced, and that at least 4 per cent of the iron was reduced by solid carbon of the coke; and if more than 4 per cent of the iron were reduced by solid carbon this amount of carbon escaped oxidation by the carbon dioxide of the limestone. The result as to the gas and the amount of heat liberated in the furnace is the same as if reduction were by carbon monoxide and the carbon dioxide of the limestone were reduced to carbon monoxide by solid carbon of the coke.

Example II.

In an experiment made by Mr. R. J. Wysor in conjunction with the writer (METALLURGICAL AND CHEMICAL ENGINEERING, July, 1910) we have conditions showing more carbon existing as dioxide than is sufficient to have reduced all the iron by carbon monoxide.

Carbon dioxide in gas ..... 13.3 per cent
$(CO_2 + CO + CH_4)$ in gas ..... 38.5 per cent
Ratio of carbon in carbon dioxide to carbon gasified, $\frac{13.3}{38.5}$ ..... .3455
Total carbon gasified per ton of pig iron..... .9725 ton
Carbon existing in gas as CO_2 per ton of pig iron $.9725 \times .3455$ ..... .336 ton
Carbon as CO required to reduce iron in a ton of pig iron ..... .300 ton
Carbon as CO_2 from other sources..... .036 ton

Thus in this test it is seen that the carbon existing as dioxide is more than sufficient to have reduced all the iron by carbon monoxide. Possibly some of this carbon dioxide may have come from the limestone unchanged and the iron reduced by solid carbon of the coke or by deposited carbon, but the result is the same as if the iron were reduced completely by carbon monoxide. There is, moreover, a surplus of carbon as dioxide which may have come from the limestone unchanged or from carbon deposition when the deposited carbon was not utilized in the reduction of iron.

Carbon Burnt before the Tuyeres.

In Gordon's "Translation of Gruner's Studies of Blast-Furnace Phenomena" the ideal working of a blast furnace is set forth as being attained when reduction is made by carbon monoxide alone. Reference is later made to the ideal working of the furnace as being attained when all the carbon of the fuel reaches the tuyeres. It does not follow, however, that all the carbon is burnt before the tuyeres even if reduction is entirely by carbon monoxide, for the carbon may be utilized in the furnace in other ways. Rather is the proportion of carbon burnt before the tuyeres an indication of the general working conditions of the furnace, for, as has just been said, it is influenced by other factors than the manner of reduction. Moreover, the amount of carbon burnt before the tuyeres is all important, for it is the sole source of heat in the smelting zone, and upon the quantity of heat developed here depends the grade of the pig iron produced.

A distinction should be made between the amount of carbon burnt before the tuyeres by dry air and by air containing moisture, as it is only by the former that heat is developed, in the latter case the moisture absorbing heat when decomposed by the carbon. In fact, the ratio of carbon burnt before the tuyeres by the dry air of the blast to the total carbon gasified (including the carbon in limestone) is an indication of the general working of the furnace. This follows from the fact that increasing the limestone, and therefore the amount of carbon used in the reduction of carbon dioxide, decreases the carbon burnt before the tuyeres and increases the total carbon gasified; the ratio decreases with increase of limestone, but not at the same rate as does the ratio of carbon burnt before the tuyeres to the carbon in coke. Moreover, this ratio, as will be shown, is readily calculated from the gas analysis alone.

In a blast furnace the carbon obtains its oxygen from the air and from the charge. The oxide of carbon formed when the carbon is burnt in air, i. e., before the tuyeres, is accompanied by nitrogen; this is the only source of nitrogen (disregarding the negligible quantity in the coke), and none is retained in the furnace. Therefore, by a comparison of the volume of nitrogen and the volume of the compounds of carbon in the top gas, the ratio of carbon burnt before the tuyeres may be found.

Dry air, a mixture of oxygen and nitrogen in the ratio by volume of 1 to 3.78, may be written $(O_2 + 3.78\ N_2)$. The action of dry air on carbon may be written thus: $2C + O_2 + 3.78\ N_2 = 2CO + 3.78\ N_2$. From this it is seen that one volume of oxygen, accompanied by 3.78 volumes of nitrogen, acts on carbon with the formation of two volumes of carbon monoxide, accompanied by the 3.78 volumes of nitrogen. the carbon monoxide is then $\frac{2}{3.78}$ or 0.529 of the nitrogen. The air supplied to a blast furnace is, however, never absolutely dry. For purposes of calculation (m) represents the ratio of the volumes of moisture and dry air in the moist air mixture, or the fraction of a cubic foot of moisture accompanying a cubic foot of dry air. Air may then be written $(O_2 + 3.78\ N_2 + mH_2O)$. The action of water on carbon is written thus: $C + H_2O = CO + H_2$, from which it is seen that one volume of water vapor acts on carbon forming one volume each of carbon monoxide and hydrogen. Then (m) volumes of water vapor cause the formation of (m) volumes of carbon monoxide. The action of moist air on carbon at the tuyeres is as follows:

$$(2+m)\,C + O_2 + 3.78\ N_2 + mH_2O = (2+m)\,CO + 3.78\ N_2 + mH_2$$

The products of partial combustion with moist air are, therefore, found to be $(2+m)$ volumes of carbon monoxide and (m) volumes of hydrogen to 3.78 volumes of nitrogen. The carbon monoxide formed from moist air at the tuyeres is therefore $\frac{2+m}{3.78}$ times the nitrogen.

The percentages of the constituent gases are here represented by their molecular formulas. Then the carbon monoxide formed at the tuyeres is $\frac{2+m}{3.78}\ N_2$ per cent of the top gas. The total compounds of carbon in the gas are $(CO_2 + CO + CH_4)$ per cent of the top gas, and represent a definite weight of carbon; namely, the total carbon gasified in the furnace $(c + s - p)$. Since the weights of carbon gasified are in proportion to the volumes of the resultant compounds of carbon the carbon monoxide burnt before the tuyeres $\frac{2+m}{3.78} \times N_2$ divided by the compounds of carbon in the gas $(CO_2 + CO + CH_4)$ gives the ratio of carbon burnt before the tuyeres to the total carbon gasified. The *carbon burnt before the tuyeres by moist air is, therefore,* $\frac{2+m}{3.78} \times N_2 \div (CO_2 + CO + CH_4) \times (c + s - p)$. The carbon monoxide formed at tuyeres by dry air being 0.529 of the nitrogen, *the ratio of carbon burnt before the tuyeres by the dry air to the total carbon gasified is:* $0.529 \times N_2 \div (CO_2 + CO + CH_4)$, or is 0.529 of the nitrogen divided by the sum of the carbon compounds present. The weight of carbon burnt before the tuyeres by the dry air of the blast is then $0.529 \times N_2 \div (CO_2 + CO + CH_4) \times (c + s - p)$

Example III.

In the experiment mentioned in Example II the carbon burnt before the tuyeres was determined as follows:

Moisture accompanying 1 cu ft. dry air of blast, m ..... 0.01 cu. ft.
Nitrogen in top gas, N_2..... 57.2 per cent
Carbon monoxide formed at tuyeres, $\frac{2+m}{3.78} \times N_2$, $\frac{2.01}{3.78} \times 57.2$ ..... 30.42 per cent.
Total compounds of carbon in top gas $(CO_2 + CO + CH_4)$ ..... 38.5 per cent
Ratio of carbon burnt before the tuyeres to total carbon gasified, $30.42 \div 38.5$ 0.790
Total carbon gasified per ton of pig iron..... 0.9725 ton
Carbon burnt before the tuyeres per ton of pig iron, 0.9725×0.790 ..... 0.7683 ton
Carbon in fuel per ton of pig iron..... 0.9550 ton
Ratio of carbon burnt before the tuyeres to carbon entering as fuel, $0.7683 \div 0.9550$ .. 0.804
Ratio of carbon burnt before the tuyeres by dry air to carbon gasified, $0.529 \times 57.2 \div 38.5$ 0.786
Carbon burnt before the tuyeres by dry air of blast, 0.9725×0.786 .7644 ton

It is noticed here that only 80.4 per cent of the carbon in the coke is burnt before the tuyeres, and under Example II it was found that the reduction was by carbon monoxide, supposing the carbon in limestone requires a like quantity to reduce the carbon dioxide driven off from the limestone. In this test the authors found that there was a surplus of hydrogen over and above that driven off from the coke and that derived from the moisture in the blast. This hydrogen must have come from water from some source, and carbon must have been used in liberating the hydrogen. In this test 6 per cent of the carbon in the coke must have been so utilized.

Example IV.

During an experiment on a blast furnace a continuous sample of gas was taken over a period of four and a half hours. Determinations of moisture in the air were also made by psychrometer at intervals of one hour. Coke was sampled continuously; the fixed carbon was determined in the sample, the value obtained being used as the total carbon in the coke.

The data and the calculations follow:

Gas Analyses.		Moisture in the Air. Gr. per Cu. Ft.	Vapor pressure in inches.
CO_2	12.3	7.	0.661
CO	25.8	7.5	0.707
CH_4	0.2	7.	0.661
H_2	4.6	7.	0.661
N_2	57.1	6.	0.555
		Average... 7	0.65

Pressure of dry air and vapor, by barometer.. 29.11 inches.
" " vapor ..... 0.65 "
" " dry air ..... 28.46 "

Ratio by volume of moisture to dry air, $\frac{0.65}{28.46}$ .. 0.023

Nitrogen in top gas.....57.1 % top gas.

Carbon monoxide formed at tuyeres, $\frac{2.023}{3.78} \times 57.1$ .....30.56% " "

Total compounds of carbon in top gas, ($CO_2 + CO + CH_4$) .....38.3% " "

Ratio of carbon burnt before the tuyeres to total carbon gasified, 30.56 ÷ 38.3.....0.798

Fixed carbon in coke (natural state)....84.1%

Coke per ton of pig iron.....2341 lb.

Carbon in coke per ton of pig iron.....1969 lb.

Carbon in limestone.....11.5%

Limestone per ton of pig iron.....1149 lb.

Carbon in limestone per ton of pig iron..... 132 lb.

Total carbon entering furnace.....2101 lb.

Carbon entering a ton of pig iron..... 89 "

Carbon gasified in furnace per ton of pig iron..2012 lb.

Carbon burnt before the tuyeres per ton of pig iron, 2012 × 0.798 .....1606 "

Ratio of carbon burnt before the tuyeres to carbon in the fuel, 1606 ÷ 1969.....0.816

Ratio of carbon burnt before the tuyeres by dry air of the blast to total carbon, 0.529 × 57.1 ÷ 38.3 .....0.789

Carbon burnt before the tuyeres by dry air of the blast, 2012 × 0.789.....1587 lb.

The Amount of Air Entering a Blast Furnace.

Ordinarily the volume of air blown is obtained from the number of revolutions of the blowing engine and the capacity of the tubs. However, it is well known that there is considerable leakage and it is advisable frequently to obtain the number of cubic feet entering the furnace. This is calculated from the weight of carbon burnt before the tuyeres by the dry air of the blast, as found above.

From the reaction of partial combustion of carbon ($2C + O_2 = 2CO$) it is seen that two atoms of carbon combine with one molecule or two atoms of oxygen. From a knowledge of their atomic weights it is found that 24 lb. of carbon combine with 32 lb. of oxygen, or that 1 lb. of carbon combines with 4/3 lb. oxygen. But it is desirable to state the number of cubic feet of oxygen rather than the number of pounds used. One cubic foot of oxygen at 62° and 30" mercury weighs 0.0843 lb. and 4/3 of a pound will occupy a volume of 15.82 cu. ft. (4/3 ÷ 0.0843), and therefore 1 lb. of carbon combines with 15.82 cu. ft. of oxygen. The nitrogen in the air is 3.78 times the volume of oxygen, or the dry air is 4.78 times the oxygen. Then 75.62 cu. ft. (15.82 × 4.78) of dry air are necessary for the combustion of 1 lb. of carbon, and the number of cubic feet of dry air entering the furnace is found by multiplying the number of pounds of carbon burnt before the tuyeres by 75.62.

The number of cubic feet of moist air may be obtained by multiplying the above by the fraction of a cubic foot of moisture accompanying 1 cu. ft. of dry air and adding the product to the number of cubic feet of dry air.

Example V.

Carbon burnt before the tuyeres by dry air of blast per ton of pig iron.....1712 lb.

Dry air at 62° and 30" mercury necessary for partial combustion of 1 lb. of carbon.... 75.62 cu. ft.

Dry air entering furnace per ton of pig iron, 1712 × 75.62 .....129,461 " "

Moisture per cubic foot dry air..... 0.01

Moisture in blast per ton of pig iron..... 1,294 " "

Moist air at 62° and 30" mercury per ton of pig iron .....130,755 cu. ft.

The Volume of Gas Coming Off from the Furnace.

The volume of dry gas may be found from the volume of dry air in the blast by comparing the percentages of nitrogen in the air and in the gas, as the volumes of dry air of blast and of dry gas vary inversely with the percentages of nitrogen contained therein. If the volume of dry air is calculated from the weight of carbon burnt before the tuyeres, the volume of gas derived from it should equal that obtained by the following method, based on the ratio of the carbon compounds in the gas to the carbon gasified.

One cubic foot of carbon dioxide, at 62° and 30", weighs 0.11591 lb. and 1 cu. ft. of oxygen weighs, under similar conditions, 0.08431 lb. Now, the molecules of carbon dioxide (CO_2) and of oxygen (O_2) contain the same number of atoms of oxygen and the same weight of oxygen. The oxygen contained in a cubic foot of carbon dioxide is then 0.08431 lb. The difference between 0.11591 lb. and 0.08431 lb. or 0.0316 lb. is the weight of carbon in a cubic foot of carbon dioxide. Analogous reasoning applied to the other compounds of carbon in the gas gives the same result—0.0316 lb. as the weight of carbon in 1 cu. ft., at 62° and 30", of the carbon compounds present in the gas. Then, by multiplying the percentage of carbon compounds in the gas by 0.0316 the weight of carbon contained in a cubic foot of the dry gas is obtained. The total carbon gasified being $(c + s - p)$ pounds per ton of pig iron, the number of cubic feet of dry gas is $(c + s - p) \div 0.0316$ ($CO_2 + CO + CH_4$).

The gas, however, contains moisture. This is usually stated as (m') grains per cubic foot of dry gas at 62° and 30". By multiplying the number of grains by 0.0030 the fraction of a cubic foot of moisture accompanying 1 cu. ft. of dry gas is obtained, and from it the cubic feet of moist gas per ton of pig iron is readily found.

Example VI.

Total compounds of carbon in the gas.....38.5 per cent.
Weight of carbon in 1 cu. ft. of carbon compounds (at 62° and 30") .....0.0316 lb.
Weight of carbon in 1 cu. ft. of gas, 38.5 × 0.0316...0.012166 lb.
Total carbon gasified per ton of pig iron.....2178 lb.
Volume of dry gas at 62° and 30", 2178 ÷ 0.012166.....179,056 cu. ft.
Volume of dry air entering furnace (Example V).....129,461 cu. ft.
Volume of dry gas, calculated from nitrogen, $\frac{79.1}{57.2}$ 129,461 .....179,028 cu. ft.

Summary.

The foregoing calculations are presented somewhat in detail and may be tedious to some. The results are here presented as formulas, by which calculations can be readily made.

Carbon gasified in the furnace after deducting the solid carbon driven off from the top of the furnace is the sum of the carbon in the coke (c) and in limestone (s) less that entering the pig, and may be expressed as $(c + s - p)$ pounds or tons per ton of pig iron. The percentages by volume of the constituent gases of the top gas are represented by their formulas; thus, N_2, CO_2, CO and CH_4 represent their percentages in the top gas. The moisture in the blast is represented by (m), the ratio by volume of moisture and dry air, or the fraction of a cubic foot of moisture accompanying a cubic foot of dry air.

Extent of Oxidation of Carbon.

Ratio of carbon as dioxide to total carbon gasified,

$$\frac{CO_2}{CO_2 + CO + CH_4}$$

Carbon gasified in furnace per ton pig iron, $(c + s - p)$ ton.

Carbon existing as dioxide per ton of pig iron,

$$\frac{CO_2}{CO_2 + CO + CH_4} \times (c + s - p) \text{ ton.}$$

Carbon required as dioxide if all iron is reduced by carbon monoxide, 0.300 ton.

See Examples I and II.

Carbon Burnt Before the Tuyeres.

Ratio of carbon burnt before the tuyeres by dry air of blast to carbon gasified,

$$0.529\,N_2 \div (CO_2 + CO + CH_4).$$

Carbon burnt before the tuyeres by dry air of the blast,

$$0.529\,N_2 \div (CO_2 + CO + CH_4) \times (c + s - p).$$

Ratio of carbon burnt before the tuyeres by moist air of the blast to carbon gasified, $\frac{2 + m}{3.78} N_2 \div (CO_2 + CO + CH_4)$.

Carbon burnt before the tuyeres by moist air,

$$\frac{2 + m}{3.78} N_2 \div (CO_2 + CO + CH_4) \times (c + s - p).$$

See Examples III and IV.

Amount of Air Entering the Furnace, as Calculated from the Carbon Burnt before the Tuyeres

Dry air, at 62° and 30", necessary for partial combustion of 1 lb. carbon, 75.62 cu. ft.

Dry air entering furnace per ton of pig iron is 75.62 times the carbon burnt before the tuyeres by dry air and is, when $(c + s - p)$ represents pounds, $0.529\,N_2 \div (CO_2 + CO + CH_4) \times (c + s - p) \times 75.62$

Moisture per cubic foot dry air is (m).

Moisture per ton of pig iron is (m) times the dry air entering furnace

Moist air entering furnace per ton of pig iron is the sum of dry air and moisture

See Example V.

Volume of Gas at Top of Furnace, Reduced to 62° and 30" Mercury

Weight of carbon in 1 cu. ft. (at 62° and 30") of CO_2, CO or CH_4, 0.0316 lb

Weight of carbon in 1 cu. ft. gas, $0.0316\,(CO_2 + CO + CH_4)$.

Carbon gasified per ton of pig iron, $(c + s - p)$ lb.

Volume of dry gas, $(c + s - p) \div 0.0316\,(CO_2 + CO + CH_4)$.

Moisture in gas reduced to grains per cubic foot dry gas at 62° and 30", m'.

Volume of moisture accompanying 1 cu. ft. dry gas, $0.003012\,m'$ cu. ft.

Number of cubic feet of moisture per ton of pig iron can be found by multiplying the number of cubic feet of dry gas by $0.003012\,m'$.

Number of cubic feet of moist gas is their sum.

See Example VI.

Laboratory, Duquesne Steel Works,
Carnegie Steel Company.

Notes on the Chemistry and Metallurgy of Vanadium.

By Warren F. Bleecker.

Introduction.

The constantly increasing importance of vanadium products in the steel industry, as well as in other branches of manufacture and arts, has stimulated investigation of the future supply of the element, and with the discovery of sufficient quantities of the ores the problem has been turned over to the metallurgist to make available the valuable content desired in the many combinations and forms by the various consumers. The problem is not an easy one, because the chemistry of vanadium is but little understood. Of the metallurgy, written in scientific language, there is no literature to be found, and, with a theoretical working knowledge of the chemistry of the element, the metallurgist is still at a loss to know how to proceed, because almost all writers have disagreed as to elementary facts, even as to what analytical group of elements vanadium belongs. A careful study of all the laboratory data obtainable reveals the fact that the problem is not to be solved in the mere attempt to produce reactions which are analagous to those of other elements, nor by a simple study of general relationship to other elements in the same or related groups. On the other hand, specially designed metallurgical methods may be developed from a more or less incomplete knowledge of the principal vanadium compounds, which methods can be and have been designed and put into operation with success.

The investigations recorded in this paper have extended over a period of two and a half years of continuous work, and while many facts have been definitely established, there is much yet to be done before some of the methods herein described can be used on a large scale without the supervision of an expert.

I have endeavored to divide the paper into three parts, dealing respectively with the chemistry of vanadium, vanadium analysis and some new metallurgical methods. Under the first heading I shall attempt to give such chemical data as will enable the reader who has not made a study of vanadium to understand without difficulty the methods of analysis and extraction that follow. Wherever possible I have examined the original references, and where I have not personal knowledge of the accuracy of experiments or statements I hope to have put the matter in such a manner as not to leave the reader in doubt as to the authority. Only the chemistry of such compounds is given as will have a bearing upon the following subject matter, or have for some reason properties of more than passing importance.

Under the second heading I have tried to discuss all of the useful methods of vanadium analysis, and a great many which are not useful, but important, either historically or because of the weight of authority that is still bearing upon them. I shall make criticisms freely, but not in a spirit of having said the last word. Rather, I am simply trying to state certain difficulties which I have encountered, and in each case I shall give reasons and authority for statements, and in addition try to

develop the correct formulas, factors, etc., by arguments based on facts given under the first heading.

Under the third heading I shall discuss in detail several methods for the extraction of vanadium from various crude ores, and methods of refining and producing compounds upon a commercial scale. I shall give some methods said to be in use with all the detail available. I shall criticise these methods, both favorably and unfavorably, and finally I shall endeavor to discuss my methods fairly, giving points of weakness as well as points of excellence.

In short, the object of this paper is to give information which is intended to be accurate and sufficiently in detail to give the reader an intelligent idea of what is being said so that he may properly judge its value. I hope that mere opinions given here are not taken as statements of fact, but I feel that opinions are valuable, because they may contain at least the germ of an idea, which unexpressed, might be forever lost.

The Chemistry of Vanadium.

Vanadium is a metallic element, never found free in Nature, but always combined with other elements forming complex minerals. It has an atomic weight of 51.2 and a valence of three and five. The specific gravity is given as 5.5, and the metal is not magnetic, nor are any of its compounds, excepting the alloys with magnetic metals.

Metallic vanadium may be prepared by heating the dichloride in a stream of dry hydrogen gas, the dichloride being made by passing dry chlorine gas over crude vanadium obtained by the reduction of vanadic acid with aluminium according to the Goldschmidt process. The vanadium obtained in this manner is said to be practically pure, and is in the form of a steel-gray powder, which is similar in appearance to tungsten metal.

Vogel and Tammann[1] claim to have made metallic vanadium assaying 99.07 per cent V by reducing the pentoxide with aluminium in a crucible lined with magnesite. The assay was made by the method of Rose[2] and the chief impurity was silicon. The remarkable statement is made that no aluminium could be observed in the final product. In view of the work of other experimenters and of my own experiments, I am inclined to believe that some error exists, especially since the method of Rose is not the easiest method with which to obtain accurate assay.

Prandtl and Bleyer[3] experimented with a reducing agent composed of a mixture of metallic calcium and aluminium, and obtained by the reduction of the pentoxide of vanadium metallic reguli which assayed as high as 94.45 per cent metallic vanadium. The purest regulus was very hard, crystalline, not brittle, and took a high polish which was stable. The color was steel-gray, and the metal was insoluble in concentrated hydrochloric acid.

I have reduced vanadium pentoxide by means of the electric arc in the form of a regulus, but the product contained much carbon and considerable oxygen, although not enough to account for all of the vanadium as either the monoxide or carbide.

Metallic vanadium is slowly soluble in sulphuric and nitric acids and in water solutions of the fixed alkalis. It is stable in the air at ordinary temperatures, but rapidy oxidizes upon heating in the presence of air.

Contrary to published statements I have never found vanadium to exhibit electro-positive properties in any solution, but always find it as an anode product, the VO_3 ion even finding its way through a porous membrane from the cathode compartment of a cell, the electrolyte of which is acid.

Vanadium, like antimony, arsenic and uranium, plays a double rôle. It unites with bases to form three series of salts known as vanadates, analagous to the salts of phosphorus—ortho, meta and pyro respectively. It also, when combined with one atom of oxygen, unites with acid radicals to form two series of salts known as vanadyl salts.

Vanadium unites with oxygen to form five oxides: V_2O_5, V_2O_4, V_2O_3, V_2O_2, V_2O, called the pentoxide, tetroxide, sesquioxide, dioxide and monoxide respectively. Of these the latter three are of academic importance only, while the tetroxide of vanadium is met with only in combination with other elements, excepting where it is especially prepared. The pentoxide of vanadium is a very important compound, and it is popularly known and spoken of as vanadic acid. It is known in three distinct physical forms, which are described as yellow, red and crystalline.

Vanadic Acid.

The yellow variety is the one sold by chemical supply houses put up in small phials and labeled "Highest Purity." In this particular instance this means less than 98 per cent. Since the product contains titanium it is difficult to obtain the exact percentage of purity, because, as I shall show later, titanium present in vanadium products is in part reported as vanadium. It is easiest made by the hydro-electrolysis of a fused copper vanadate ingot, and obtained in this way may be 100 per cent pure. It is soluble in water, a saturated solution containing 300 mg to 400 mg per liter, depending upon the position of ammonium metavanadate by heat, and sometimes results from the decomposition of vanadates by mineral acids. This yellow vanadic acid sometimes results from the decompresence of slight traces of mineral acids. Pure solutions usually do not contain quite as much as 300 mg. The water solution is strongly colored yellow, the minutest trace of the dissolved acid being apparent. Although the yellow vanadic acid may be dissolved in mineral acids, the best solvent is either sodium hydroxide, to which a little hydrogen peroxide has been added, or sodium carbonate. Alkalis dissolve the vanadium to a colorless solution, while concentrated sulphuric acid dissolves it to a deeply colored solution, blood red when very concentrated, but blue upon dilution.

The red variety of vanadic acid is the one most commonly met with in the laboratory, and, like the yellow acid, is amorphous and very light. It results from the action of mineral acids upon metallic and alkali vanadates as a rich looking, curdy precipitate which is easily filtered and washed. The vanadate must either be in the form of a dry powder, or in a very concentrated solution in the case of alkali vanadates. The mineral acid may be added in any desired degree of concentration, but too great a quantity will dissolve the vanadic acid completely in most cases. The precipitation is not in any case quantitative, but ordinarily, by careful manipulation, 25 per cent to 30 per cent of the vanadic acid may be removed from copper vanadate by decomposition with sulphuric acid. This reaction will be discussed more fully under a description of the commercial production of vanadic acid. The red vanadic acid deports itself similarly to the yellow variety, except that it is slightly more soluble, and a water solution of the pure salt may contain over 500 mg per liter of the acid. It is soluble in acids, out of which it may be precipitated by boiling if the concentration is right. The product formed in this way is never pure, but always contains a portion of the impurities in the solution.

Dissolved in hydrochloric acid the red oxide sometimes changes to the yellow oxide upon evaporation of the solution, but more often the crystalline form results from this treatment.

The dried oxide feels greasy to the touch, and discolors the skin to a slight extent. It rapidly takes on water to the extent of nearly 10 per cent. This water may be removed by careful heating to about 300 deg. C., a higher temperature resulting in a blackening and finally in fusion and transformation into the crystalline variety. The finely divided powder has an odor similar to that of rotten apples, and is unmistabable. All acid vanadium solutions have this odor as well.

[1] *Zeit. f. anorg. Chem.*, 1909, p. 225. R. Vogel u. G. Tammann. Zur Darstellung des Vanadins auf Aluminothermischen Wege.
[2] Analytical Chemistry, Treadwell-Hall, Vol. II, p. 231.
[3] *Zeit. f. anorg. Chem.*, 1909, p. 217. W. Prandtl u. B. Bleyer. Uber die Darstellung von Vanadin und anderen Metallen nach dem Thermitverfahren.

The crystalline variety of vanadium pentoxide is obtained from either of the other two by fusion in either porcelain or platinum. The mass becomes liquid at about 600 deg. C., and may be easily poured into molds. Upon cooling it decreases in volume about one-fifth, and is quite hard and very brittle. When broken, the ingot shows a remarkable crystalline structure which is in the form of needles arranged parallel and extending inward perpendicular to each surface. The ends of the crystals meet at 45 deg. and a vertical section has the appearance of three pyramids, each with the base extending outward. In certain cases where the vanadic acid is the result of an acid evaporation, the mass cannot be obtained in an ingot, on account of its boiling out of the mold at the time of cooling. No amount of remelting will enable the chemist to get a good pour, the mass acting in the same way each time. The action is probably due to the absorption and expulsion of gas, either oxygen or nitrogen. A familiar example of this phenomenon is that of the absorption of oxygen by pure molten silver and its subsequent expulsion upon cooling.

Small amounts of impurities affect the properties of fused vanadic acid to a marked degree. Potassium salts make the ingot yellowish in color and cause it to absorb water and disintegrate. Titanium makes the ingot extremely hard, while iron makes it brittle. The presence of copper does not seem to have any effect on its physical properties when cold, but it seems to make it more tractable while being melted and poured.

The oxide while molten is a very powerful reagent, attacking all containers, even platinum and fused silica. It attacks silica crucibles with the formation of silicates; it unites with iron to form a crystalline variety of vanadate of iron, which has the appearance of the natural phosphate of iron; it attacks carbon and graphite crucibles with the formation of vanadium carbide. Fused silica, and porcelain, which are attacked but slightly at first, cannot withstand the strains due to the cooling of the thin crust left after pouring. I have found platinum crucibles to undergo no loss after several melts, and again, to lose 75 mg during a single melt. It is possible that this loss may have been due in every case to the presence of some reducing agent, for it is a well-known fact that metallic vanadium will attack platinum very readily.

The thin coating of the oxide left in the crucibles after pouring presents a beautiful crystalline appearance, the crystals being arranged in clusters with radiating elements. This coating, although thin, adheres strongly to the crucible, and is removed only by continued treatment with either strong acids in large quantities or by boiling in a vessel filled with alkali.

Sir William Crookes[4] mentions that platinum crucibles after having been used for the ignition of vanadium pentoxide several times lose their shape, the crucible becoming distorted at the bottom. This is due to the strains set up in the structures of the oxide while cooling, and accounts for the fact that the best porcelain crucibles are shattered during the process of melting in a flame which does not heat the crucible evenly.

The fused oxide is a conductor of electricity, but I have never been able to cast an anode of it which would withstand the disintegrating effect of the electrolyte. The action of the electric current upon an anode made of pure vanadic acid in an electrolyte of a water solution of the acid also slowly causes the formation of the yellow variety of the pentoxide, which form does not adhere to the crystalline mass.

The remaining oxides of vanadium are at present of no particular importance to the metallurgist, except it may be noted[17] that investigators have mistaken a regulus of the monoxide for the metal itself.[18]

Other Vanadium Salts.

Besides the oxides the following compounds of vanadium are known, and have been investigated: Chlorides,[5, 6, 7, 8, 9, 16]; oxychlorides,[6, 7, 8, 9, 18]; bromides and iodides,[9, 16]; fluorides,[10, 11, 12, 16]; sulphates,[14, 15]; sulphides,[16]; selenides and selenates,[17]; nitrides,[16]; nitrates,[16]; phosphates,[16]; arsenates,[16]; borates,[16]; silicates,[16]; silicides,[8]; carbide. The vanadates of sodium, potassium, calcium, ammonium, lead, mercury, copper, silver, barium, molybdenum, tungsten, uranium, manganese, nickel, cobalt and iron are well known, and need no reference.

I have investigated vanadates of aluminium, zinc and gold in addition to those mentioned above, as well as vanadyl oxalate and acetate. It is probable that vanadium forms stable salts with magnesium, platinum, osmium and iridium, and the rarer metals, although little has been published regarding these compounds. Vanadium also forms alloys with the common metals, and these alloys, containing only a small percentage of vanadium, have for the most part distinct properties, some of which are very valuable.

I shall discuss only the chemistry of those compounds of vanadium which are of importance in metallurgical operations of extraction or incident thereto.

Three chlorides of vanadium are known[16] (p. 709), two of which, VCl_2 and VCl_3, are formed from the third, VCl_4. The tetrachloride is formed by passing dry chlorine over metallic vanadium at a red heat[16] (p. 710) [5] (p. 345).

Dammer mentions six oxychlorides of vanadium[16] (p. 710), three of which are said to be derived from the vanadium oxytrichloride. The statement is made that this compound, which has the formula $VOCl_3$, is prepared by heating a mixture of vanadium pentoxide and carbon in an atmosphere of dry hydrogen, allowing the mass to cool, and then heating in a stream of chlorine. Divanadyltetrachloride $(VO)_2Cl_4(H_2O)_5$ is formed by the action of hydrogen sulphide upon a hydrochloric acid solution of vanadium pentoxide with the separation of sulphur. The compound $2VO_2 . 4HCl . 3H_2O$ is said to result from heating vanadium pentoxide in a strong solution of hydrochloric acid, with the evolution of chlorine.

The action of concentrated hydrochloric acid upon vanadium pentoxide or upon a dried metallic vanadate, such as iron vanadate, results in the formation of vanadyl dichloride, $VOCl_2$, or upon vanadic acid, results in the formation of vanadyl dichloride $VOCl_2$ with an evolution of chlorine gas, according to the reaction,

$$2V_2O_5 + 12HCl = 4VOCl_2 + 6H_2O + Cl_2$$

The bromides and iodides of vanadium are analagous to the chlorides, and there are the corresponding oxysalts.

Melikoff and Kasanezky[18], and Ephriam[16] develop the following structural formula for a vanadyl ammonium fluoride which is of interest as showing the general character of all vanadium compounds which are in the highest state of oxidation

$$\begin{matrix} NH_4-O & & -F & -F & -NH_4 \\ & V & & & \\ NH_4-O & & -F & & \end{matrix}$$

Vanadium pentasulphide is probably formed by the addition of ammonium or sodium sulphide to an alkaline solution of vana-

[4] Sir William Crookes, "Select Methods in Chemical Analysis." Ed. 1905, p. 186.

[5] *Zeit. f. anorg. Chem.*, 1905, p. 315. J. Koppel, R. Goldman u. A. Kaufmann. Verbindungen des vierwertigen Vanadins.

[6] *Zeit. f. anorg. Chem.*, 1902, p. 456. F. A. Gooch u. J. B. Stookey. Die Reduktion der Vanadinsaure durch Chlorwasserstoffsaure.

[7] *Zeit. f. anorg. Chem.*, 1902, p. 66. F. Ephriam. Uber die Einwirkung von Chlorwasserstoffsäure auf Vanadinsaure.

[8] *Zeit. f. anorg. Chem.*, 1902, p. 71. F. Ephriam. Versuche zur Darstellung Chlorhaltiger Doppelverbindungen des Vanadinpentoxyds.

[9] *Zeit. f. anorg. Chem.*, 1904, p. 246. F. A. Gooch u. R. W. Curtis. Die Einwirkung von Halogenwasserstoffsäuren auf Vanadinsaure.

[10] *Zeit. f. anorg. Chem.*, 1903, p. 247. P. Melikoff u. P. Kasanezky. Uber Vanadinverbindungen.

[11] *Zeit. f. anorg. Chem.*, 1902, p. 80. F. Ephriam. Notiz uber die Konstitution der Vanadindoppelfluorid.

[12] *Zeit. f. anorg. Chem.*, 1904, p. 442. P. Melikoff u. P. Kasanezky. Uber die Konstitution der Fluorvanadin Verbindungen.

[13] *Zeit. f. anorg. Chem.*, 1905, p. 352. J. Koppel u. A. Kaufmann. Versuche zur Darstellung von Vanadinmetall und einigen Vanadinverbindungen.

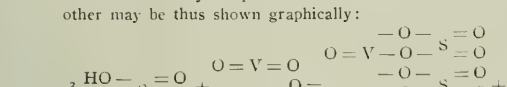
[14] *Zeit. f. anorg. Chem.*, 1902, p. 55. A. Piccini u. L. Marino. Uber einige Vanadinverbindungen von der Form VX_4.

[15] *Zeit. f. anorg. Chem.*, 1904, p. 152. L. Marino. Uber das electromotorische Verhalten des Vanadins.

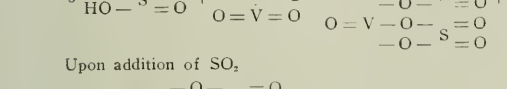
[16] Dammer, "Handbuch der anorganische Chemie."

dium pentoxide, out of which solution a mineral acid will precipitate the vanadium sulphide incompletely as a brownish colloidal mass, a portion of which can be collected upon a filter. Dammer[15] (p. 716) says that none of the sulphides are formed by the addition of H_2S to a solution of a vanadium salt, but that the precipitate formed by the addition of hydrogen sulphide to an alkaline solution is but a mixture of an oxide (he does not say which) and elementary sulphur. Just why an oxide should precipitate out under such conditions is not clear, and so far as I know the subject awaits investigation.

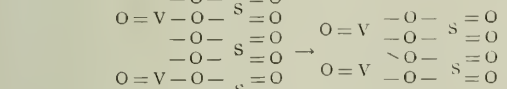
The vanadyl sulphates are important, there being five mentioned by Dammer[16] (p. 716). The usual reaction by means of which the vanadyl sulphates are converted from one to the other may be thus shown graphically:



Upon addition of SO_2



Containing V^4 and V^5 in equal parts. Upon addition of H_2O_2 or $KMnO_4$



According to the reaction:

$$5(VO)_2(SO_4)_2 + 2KMnO_4 + 8H_2SO_4 = 5(VO)_2(SO_4)_3 + K_2SO_4 + 2MnSO_4 + 8H_2O$$

Vanadium unites with silica chiefly in combination with other elements, but there is at least one compound containing only vanadium, silicon and oxygen. I have caused silica and vanadium to unite by mixing finely divided quartz with red vanadic acid, then fusing the mixture. The resulting mass is brown, or black, due to the uncombined vanadic acid. Upon grinding this fused mass and washing it with weak sodium carbonate solution, there remains a white residue which consists principally of quartz, but which contains vanadium. In using impure silica (free from vanadium) I have obtained upon washing with alkali a green residue which could not be distinguished with the eye from the original vanadiferous sandstone. This residue contained vanadium which was with difficulty extracted with nitric acid. I have also prepared a compound of vanadium and silica by treating an alkaline solution of sodium silicate with red vanadic acid added in the powdered form. The jelly-like precipitate which resulted was poured into a filter and washed until the wash water (containing a little alkali) showed no trace of vanadium. The product was white when dried, but various samples were not of constant composition. The precipitate when wet was distinguished from pure silicic acid by the readiness with which it could be washed on the filter. The same material probably results upon the addition of acetic acid to a solution of sodium vanadate containing also sodium silicate.

[17] *Zeit. f. anorg. Chem.*, 1907, p. 393. W. Prandtl u. F. Lustig. Über Vanadinselenverbindungen.
[18] *Zeit. f. anorg. Chemie*, 1902, p. 174. F. A. Gooche u. R. D. Gilbert. Die Fällung des Ammonium Vanadats durch NH_4Cl.
[19] *Zeit. f. anorg. Chemie*, 1902, p. 181. A. Rosenheim. Die Fällung von Ammonium Vanadat durch Chlorammonium.
[20] A. H. Low, "Technical Methods of Ore Analysis." Ed. 1905, p. 205.
[21] Brearly, "The Analytical Chemistry of Uranium." Ed. 1903, p. 31.
[22] *Zeit. f. anorg. Chem.*, 1907, p. 368. T. F. Rutter. Beitrage zur Kenntnis der Vanadinverbindungen.

Alkaline Vanadates.

The vanadates of sodium are of importance because of their formation in all of the proposed methods of roasting crude ore to effect extraction. There are nine of these sodium vanadates, of which Dammer[16] (p. 721) gives seven. Of these the metavanadate, $NaVO_3$, and the pyrovanadate, $Na_4V_2O_7$, are easily soluble in cold water, the remainder are soluble with more or less difficulty, the dihypovanadate and the hexavanadate being almost insoluble in water, either hot or cold. All of the simple sodium vanadium compounds are decomposed by mineral acids, with the formation of vanadyl salts, and, if the solutions are highly concentrated, with separation of vanadic acid.

The vanadates of potassium are for the most part analagous to the vanadates of sodium, but they are not of metallurgical importance because of the relatively great cost of the potassium salts from which they could be made.

There is a considerable number of compounds of vanadium, sodium and sulphur which I have prepared in large quantities but have not yet had the time to analyze. One compound in particular occurs in the form of bright red crystals, very difficultly soluble in water and slowly soluble in mineral acids. It carries about 78 per cent V_2O_5, the remainder being sodium, sulphur and oxygen.

An acid added to a water solution of sodium vanadate will form vanadyl salts if the solution is dilute, and will form vanadic acid if the solution is concentrated.

If to a concentrated solution of sodium vanadate, which has been electrolyzed in the anode compartment of a porous cell, ammonia is added, a complex sodium ammonium vanadate is precipitated, and may be readily filtered and washed. The salt is crystalline and of a bright red color. The precipitation is not quantitative, but seems to be an instance of the formation of a salt by the addition of a relatively weak to a strong base.

Dammer gives seven distinct ammonium vanadates, analogous to the vanadates of sodium, of which the metavanadate is by far the most important, owing to its insolubility in ammonium chloride.[16-18-19] Ammonium vanadates are decomposed with alkalis and with mineral acids, in the latter case with partial separation of vanadic acid if the operation is performed with care and with this object in view. Ammonium metavanadate is formed by the addition of a large quantity of ammonium chloride to a concentrated neutral solution of sodium vanadate. It is decomposed on heating, leaving a residue of vanadium pentoxide in a state of purity corresponding to the purity of the ammonium compound. By repeated resolution of ammonium metavanadate in alkali, and reprecipitation with pure ammonium chloride, practically pure vanadic acid may be obtained by heating the final product. I have never been able to obtain a quantitative precipitation, although this is possible.[5-22-23] The salt is pure white, crystalline, filters readily, and may be washed with a moderate amount of hot water without appreciable loss if the water contains a small amount of ammonium chloride.

Metallic Vanadates.

Of the metallic vanadates it may be said in general that they are formed by the addition of a soluble salt of a metal to a neutral solution of sodium or potassium vanadate. They all carry much water when pressed as dry as possible, and they are all very light when dried, but not ignited, with the exception of the vanadates of mercury, lead, gold and barium, all of which are slightly heavier in gravity than the others. They are all decomposed upon the addition of mineral acids, usually with the separation of vanadic acid, and are reprecipitated upon the addition of alkalis to neutrality. All of the metallic vanadates seem to be soluble in water to a greater or less extent, but for the most part they are insoluble in water containing a very small quantity of the precipitating agent. All of the metallic vanadates when precipitated chemically may be colloidal, and some of them are always so. This is especially true of the vanadates of iron, copper, zinc and aluminium. Those

least tending to form colloids are the vanadates of lead, mercury and calcium.

The vanadates of mercury, lead, copper and iron are fusible at a temperature of about 600° C., while the vanadates of aluminium, calcium, zinc and tungsten are infusible at temperatures obtained with a blast lamp under the conditions of my experiments. The fused vanadates of iron and copper were found to conduct electricity to a sufficient degree to warrant this property being taken into consideration in trying to solve the metallurgical problems. I am not at this time sure, however, that the pure compounds would act in this manner, although the melting has been conducted in such a manner as to preclude the possibility of a partial reduction of one or more of the elements to the metallic state, and the conductivity of the material was seemingly the same as that of vanadate fused in graphite crucibles.

The fused vanadates of iron and copper are very heavy, probably having a specific gravity equal to iron or thereabouts, and are extremely hard. In fact, a hack-saw is dulled after a few strokes across the ingot, and the best grade of steel drill becomes greatly worn during an attempt to obtain drillings. The ingots may be broken with a hammer perhaps a little more readily than hard quartz, and present on the interior a crystalline structure which is much harder than the outer layer, which does not appear to be crystalline. The crystals found in the vanadate of iron ingots are much larger than those found in the vanadate of copper.

Since the metallic vanadates are not fully discussed in any of the papers which I have read, I shall devote a small amount of space to describing briefly those which I have studied.

By far the most important metallic vanadate is the vanadate of iron. This is a general name given to the ortho, meta, pyro and tetra vanadates, as well as to a mixture of one or more of these with ferric or ferrous oxide. Vanadate of iron is prepared by the addition of a soluble salt of iron, usually the sulphate, to a solution of sodium vanadate. The precipitate may be either black, green or yellow. The precipitate is usually colloidal, and contains considerable vanadium combined with sodium, as well as iron, which salts are slightly soluble in water.

The salt when dried is in the form of lumps, which may easily be crushed to a light powder, which may be either green, yellow, brown or red, the olive green variety being of the highest grade, and probably either the tetra or the ortho vanadate, or a mixture of the two. The more nearly the vanadate approaches red in color the lower its grade, the V_2O_5 content in some of the very red material I have assayed carrying as low as 18 per cent V_2O_5.

Vanadate of iron is acted upon by mineral acids of all degrees of dilution, but less readily when calcined or fused. The separation of vanadic acid is slight, owing to the difficulty of dissolving the oxides of iron which seem always to be present. It is acted upon to some extent by strong solutions of the fixed alkalis with the formation of sodium or potassium vanadate and hydroxide of iron.

Vanadate of copper is of considerable importance in metallurgical operations. It may be formed chemically only under favorable conditions, and when so formed is usually very impure. The impurities are difficult to remove by the ordinary methods of washing, and further, the precipitation is usually not quantitative. If to a solution of sodium vanadate a solution of copper sulphate be added, copper vanadate will be precipitated. The precipitate is usually blue, although it may be yellow or even green, depending somewhat on the purity. The best method of producing vanadate of copper is the electrolytic method which will be described in detail later on.

Vanadate of copper is readily decomposed by the mineral acids, with the separation of vanadic acid. It is not easily decomposed with alkalis, being but slightly soluble in these reagents. The dried powder is light, and is very irritating to the nose and throat. It may be easily melted and cast into ingots. A graphite crucible is used for this work with fair success. The crucible is steadily attacked, and the average number of melts is about four per crucible. The copper vanadate is reduced, the copper appearing as metal and the vanadium as carbide and silicate. It is a remarkable fact that the reduced copper contains but a trace of vanadium, the greatest amount ever found being under 1 per cent. Copper containing even this small amount of vanadium appears to be considerably toughened, and is much harder under the file than ordinary copper.

Vanadate of mercury is of no especial importance except as to its use in a method of gravimetric analysis. It is formed by the addition of mercurous nitrate to a neutral solution of sodium vanadate. In the process of analysis the mercury is removed by distillation, but it is a difficult matter to remove it entirely except by prolonged heating at a fairly high temperature. It is readily decomposed by acids.

The vanadate of zinc may be made in at least two ways. It is formed when a solution of neutral zinc salt, such as the sulphate, is added to a neutral solution of an alkali vanadate. The precipitate is usually yellow, though sometimes almost white. It is flocculent and quite gelatinous. It may be washed with water by means of a suction pump, and when dried is of a light yellow color, and, as characteristic of all of the metallic vanadates, excepting lead and mercury, it has a very low specific gravity. Vanadate of zinc is readily decomposed by acids with a separation of vanadic acid. No commercial importance is attached to this method of vanadic acid production, since none of the reactions are clean, and the resulting product is of a low grade, being contaminated with zinc. Vanadate of zinc is formed also electrolytically from solutions of alkali vanadates.

Vanadate of silver is formed in a manner similar to vanadate of zinc. It is slightly more stable in the presence of mineral acids, but in general the reactions are of no especial importance.

Vanadate of gold is formed chiefly by precipitation from solutions of alkali vanadates with gold salts. It is formed with extreme difficulty by electrolytic methods.

Calcium also unites with vanadium to form a number of vanadates, of which the metavanadate, $Ca(VO_3)_2$, is the one usually met with. These vanadates are formed by the addition of soluble salts of calcium to solutions of alkali vanadates. The metavanadate is white and very porous when dry, but contains over 90 per cent of water when removed from the filter press in cakes. It is readily decomposed by mineral acids with a separation of vanadium pentoxide, and partially decomposed upon treatment with a strong solution of sodium carbonate. It is unaffected by the hottest flame of a Teclu burner, and it does not readily absorb moisture from the air. In cases where it is necessary to add lime to an electric furnace product for fluxing, the vanadate is a very convenient form. Vanadates of lime are found in mill solutions at all times unless tedious steps are taken to remove them, and are present in all residues.

Vanadate of aluminium is made both chemically and electrolytically, and has the same appearance when dried as vanadate of lime. It is soluble in the mineral acids with the separation of vanadic acid, and is decomposed to a slight extent by strong alkaline solutions. It is infusible at the ordinary temperatures, and is of no importance metallurgically. It is only interesting because it exists at all.

Vanadate of lead may be made both chemically and electrolytically from solutions of alkali vanadates. It is yellow and heavy. When fused it may be cast into ingots, but the fused mass is a non-conductor of electricity. Fused, it is of a dirty gray color, very hard and quite brittle. It is decomposed readily by all mineral acids without the formation of vanadic acid. When decomposed with sulphuric acid the resulting solution contains vanadyl sulphate, which may be evaporated and fairly pure vanadic acid recovered. The decomposition of

lead vanadate with sulphuric acid is not complete unless the acid is dilute and unless a large excess of the acid be used. The residue of lead sulphate is colored yellow and the last traces of vanadium can be removed only by solution of the lead sulphate in ammonium acetate, and making a second separation.

The precipitation of vanadate of lead chemically from solutions is not a satisfactory one, except under very favorable conditions. It is best accomplished in an acetic acid solution by the addition of lead acetate. Under these conditions the vanadate is precipitated quantitatively, and may be readily filtered and washed. The best solvent for vanadate of lead is nitric acid, in which both the vanadium and lead go into solution. The lead may be removed by any one of a number of methods, and the resulting vanadium solution is of a high degree of purity. I shall deal with the electrolytic production of vanadate of lead later on.

Necessity for Uniform Conditions in Work with Vanadium.

Investigators have stated with undoubted truth that vanadium, when it cannot unite with any other group to form salts, will unite with itself, with the formation of vanadyl vanadates.

The deportment of the various vanadates and vanadium compounds with reagents is at first very confusing, because a very slight change of conditions will cause the formation of salts in different degrees of oxidation, and the chemist may get a precipitation when he expected a solution, or vice versa, or what is still more troublesome, a quantitative result in neither instance. It has been my observation and experience that this is the rock upon which almost all metallurgical methods for vanadium treatment have been wrecked. One of the first requisites for a successful metallurgical method is therefore a means of keeping the vanadium in one or the other state of oxidation during a given operation, and if possible, to predetermine what the stage of oxidation shall be. Unless this is done there can be no uniformity of results and success will be extremely uncertain.

A standard work on technical analysis directs the chemist to dissolve the vanadium in ammonium carbonate to effect the separation of vanadium from iron, etc.[20] Another writer[21] advises "pouring the filtered HNO_3 solution of the mineral (ore) into sodium carbonate, which causes the vanadium to go down with the iron." Now the action of sodium, ammonium and potassium carbonates upon compounds of vanadium is practically the same under the conditions of the directions quoted above. But if vanadium solutions are concentrated, or contain much iron or alumina, or are in a reduced (V_2O_4) condition, these reagents will cause a more or less complete precipitation, especially if the alkali is not added in large excess. Indeed, under the most favorable conditions of dilution, oxidation, etc., some of the vanadium may remain with the precipitate of iron and alumina; but only when the molecular quantity of iron or alumina or lime exceeds that of the vanadium can the vanadium be precipitated quantitatively, and then only when the solution is barely neutral.

The statement is met with among those who have not carefully studied the reactions of vanadium that the element is elusive, and does not act consistently in similar cases. This statement is, of course, not true, but the foundation for it lies in the fact that conditions which would in no wise affect other and more common elements cause the vanadium to change its state of oxidation, to become either acid or basic, or both, or cause it to eliminate itself from the field of action by precipitation. Such are the slight changes in oxidation which will cause a solution containing iron, vanadium, sodium and sulphuric acid to precipitate when exposed to the air. This reaction is of extreme importance, as it has a direct bearing upon the extraction of vanadium from the ores, and will be discussed later.

Ionization of Vanadium Compounds.

Dilute solutions of vanadic salts are yellowish in acid when dilute and blood red when concentrated. Upon dilution the red color disappears, and the yellowish returns. This may be explained by the fact of the great degree of ionization in dilute solution, and the solution of the compound itself when concentrated.

The action of hydrogen peroxide upon acid solutions of vanadium salts is interesting, the reaction being generally considered as one of oxidation only. Rutter[22] suggests that the action of H_2O_2 causes the formation of pervanadic acid, which is not really a compound of vanadium in a higher state of oxidation, but simply an addition compound.

The phenomenon may be explained by considering the action of H_2O_2 to cause hydrolysis as indicated by the formula:

$$\begin{matrix} & & &

tuyeres and preheating the blast by molten slag. In place of a bustle pipe and tuyeres are a crucible chamber and a complete open space of 360° circumference and 2½ in. in height between the top of the crucible and the bottom of the bosh where the water jacket terminates.

In Fig. 1 the crucible is shown inside the crucible chamber on the jack plate, by means of which it is raised into position below the bosh. The slag overflows (it is never tapped) through

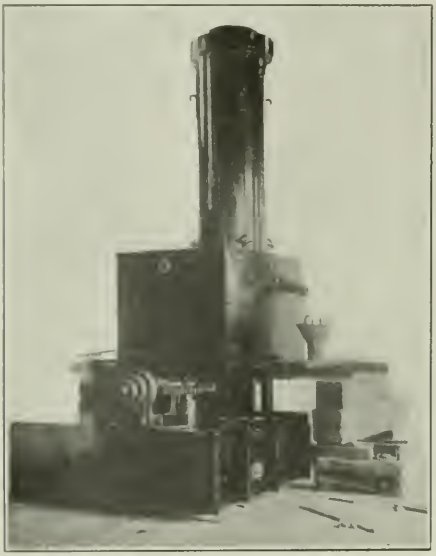


FIG. 2.—SMALL COPPER AND LEAD BLAST FURNACE.

the spout on the right, while the matte tap is on the outside of the crucible chamber (Fig. 2) and is made airtight by a clay packing on the spout block. The slag is carried in small, shallow cars that travel slowly beneath the slag tap and thence around the circumference of the slag box, finally dumping automatically into a boot in a water seal below. Here a perforated bucket elevator raises it to a car to be run to the dump.

The blast enters the slag box above the point where the cars discharge, passes over the cars of molten, inert slag (i. e., inert as far as the oxygen of the air blast is concerned) and up into the crucible chamber, where it then receives the radiation from the crucible itself. It thus enters the smelting zone at a high temperature. Consequently the molten material is not robbed of any heat to raise the temperature of the air to its level, so that the blast performs its duty of oxidation without the constant fluctuation of temperature inevitable at the tuyeres of the standard smelter due to the chilling effect of cold blast on the slag. It is perfectly evident then that a certain number of heat units are constantly being put back into the furnace and as a result less fuel is required to operate it. This method of heating the blast seems certainly preferable to others now in vogue; as, for example, an external stove, which requires extraneous fuel and has a short life, due to continual contraction and expansion, or to the system of utilizing the radiation from flue gases, which does not heat the blast sufficiently.

This furnace has demonstrated the fact that the introduction of the blast through tuyeres at a high pressure is erroneous. This applies to a reducing smelt as well as an oxidizing one, and is especially evident in "pyritic smelting." In order to smelt correctly it is essential that the blast enter the zone of fusion at a rate of speed consistent with maintaining an active combustion, but the excessive blast-forge effect is wrong. It causes heavy flue dust, an increase in the height of the fusion zone, and losses of metals by volatilization.

All of these disadvantages are overcome in this furnace and a pressure of from 4 oz. in small furnaces to 12 oz. in large ones is maintained. In other words the actual *air inlet* is increased and an enormous volume of air enters the furnace at a low pressure and velocity, enabling the oxygen to react thoroughly on the sulphides, thereby serving the dual purpose of oxidizing their iron content for fluxing the silica and utilizing the heat of the sulphur. Thus no preliminary roasting is necessary and high-grade mattes, or even black and blister copper, are obtained.

On a run made at the Denver experimental plant, under the supervision of the writer, on ore from Cen. al City, Col., the following results were obtained:

	Ore.	Slag.	Matte.
SiO_2	71.2 per cent.	46.5 per cent.	
FeO	0.8 " "	29.0 " "	
CaO	0.8 " "	15.0 " "	
Al_2O_3	4.5 " "		
Cu	2.73 " "	Tr	14.4 per cent.
Au	0.29 oz.	ni.	1.51 oz.
Ag	2.6 "	Tr	11.98 "

Coke used was 8 per cent of gross charge.

The ore was fluxed with metallic iron, which accounts for the low rate of concentration.

Iron matte smelting can be satisfactorily practiced when the

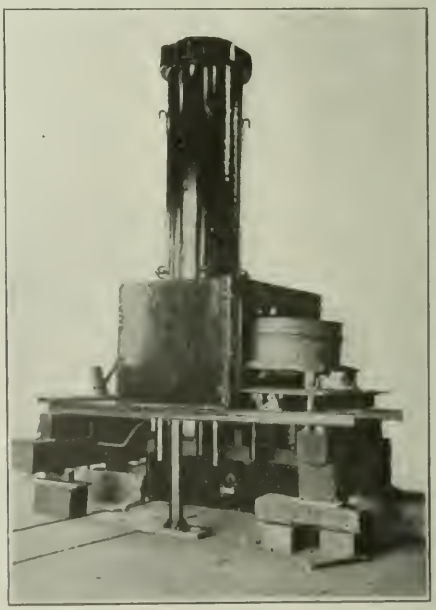


FIG. 3.—SMALL COPPER AND LEAD BLAST FURNACE.

copper content of the ore is very low or nil, but at least 15 per cent FeS_2 should be present and 0.25 per cent Cu to give a clean slag. The smelter is equally adaptable to lead ores, the only change in construction being a lead well connected to the crucible, and for furnaces of a greater capacity than 60 tons a forehearth is used.

The power required for the smaller sizes is as follows:

5 tons per 24 hours	3 hp.
15 " " " "	6 "
30 " " " "	12 "
50 " " " "	23 "

These sizes can be built in small sections, if necessary, for mule-back transportation, and riveted at the mine.

The ore shaft is water-jacketed with hand holes at the bottom to clean out sediment and scale and thus prevent the bottom plate from burning out. The water jacket discharges into the slag pit and the water seal thus formed prevents the blast escaping as the cars dump. Mica peep holes in the top of the crucible chamber allow the furnace man to watch the operation of his furnace. The only brick work necessary is that for the crucible lining.

The operations of "blowing in" and "blowing out" are very simple and rapid. By turning a crank, which operates a set of miter and spur gears, the four jack screws raise or lower the jack plate on which the crucible rests on a 27-in. gage track. The crucible chamber door has on its inside face a track that is in juxtaposition to that of the jack plate when the door is open and horizontal. This facilitates change of crucibles, as an old crucible may thus be run out on a T-rail track extending to a turn sheet outside of the building and an auxiliary crucible, previously heated, be run in, the door closed, and the crucible raised into position below the bosh. At the "blow in" the slag will be running within 10 minutes of turning on the blast.

The furnace shown in the cuts has an ore shaft 22 in. in diameter and is boshed at the bottom to 18 in. It has a rated capacity of 15 tons per 24 hours and is intended to meet the wants of the small operator. The standard type of smelter would surely "freeze" if made as small as this, for the introduction of cold air under a high pressure through small tuyeres has a tendency to chill a small furnace, and "muscular metallurgy" is often necessary. In fact, the smallest circular furnace carried in stock by the supply companies has a diameter at the tuyeres of 30 in. for copper smelting and 36 in. for lead smelting and these would have a capacity of only 20 tons to 30 tons. The open fusion zone and hot blast account for the free running and rapid driving of the Partridge furnace and it has a minimum smelting capacity of 8½ tons per square foot of furnace area for copper as against 6½ tons, the best result obtained by the standard smelter.

It is believed that this smelter will prove of great value to base metal mining in general and to small, inaccessible copper and lead properties in particular, such as those located in Central and South America, Mexico, Alaska and the western part of the United States.

Some Alloys for Permanent Magnets.

BY C. F. BURGESS AND JAMES ASTON.

In the investigation of electrolytic iron and its alloys, carried out at the University of Wisconsin under grant of the Carnegie Institution of Washington, tests were made of the magnetic quality of all samples. Among these many showed low permeabilities and high coercive forces, suggesting their utility for permanent magnets rather than as electromagnetic material. Upward of 100 samples were tested for this feature; all of the bars contained chromium, manganese, molybdenum, nickel and tungsten as binary alloys, or in various combinations together, or with other elements.

Much information regarding the best steels and the best treatment for permanent magnets has undoubtedly been accumulated by the manufacturers and users of such material, but is held as a trade secret, and published data along these lines is very meager. Again, such publications deal largely with the study of the critical points of the steels, with the view of determining the proper heat treatment to give them for the best service.

The tests here recorded are for the most part on carbon-free alloys. The results are mainly of value, therefore, as indicating the influence of the various elements in inducing the conditions necessary for a good permanent magnet. They were made upon an Esterline permeameter, an apparatus giving directly the flux density in the bar under test, for any impressed magnetizing force. It consists of a small dynamo, driven at predetermined constant speed. The test bar forms part of the magnetic circuit of the generator and can be subjected to magnetizing forces of varying degree. The flux density in the bar, being directly proportional to the voltage set up, can be read off on a properly calibrated millivoltmeter.

It is not necessary to make a complete magnetization curve in testing material for permanent magnets. The desirable feature is to have as high a residual magnetism as possible, or better, to have a high retentivity after some disturbing factor has tended to destroy the residual magnetism, together with a high coercive force to resist the tendency toward this loss of the magnetic flux. The method of procedure was as follows: The bar was inserted in the permeameter, the magnetizing force H was raised to the maximum of 200 and the density B (MAX) was recorded. H was decreased to zero, and the retentivity B (RET) noted; next the coercive force Hc necessary to reduce the retentivity to zero was determined. Similar readings were taken for the reverse magnetizing forces, and the average of these positive and negative readings of B (MAX), B (RET) and Hc for each bar was recorded for comparison. In fact, the whole operation is identical with the taking of a hysteresis loop in which only the few readings noted above were recorded.

The bars were tested as forged, and after water quenching at 1000° C. To prevent oxidation of the samples while heating them for quenching a barium chloride bath was used. The quenching at 1000° C. may not have brought out the best qualities of some of the bars, but it was deemed the best general treatment, since with so many bars under test it was impossible in this research to subject each sample to tests for its critical points and special conditions of heating.

In addition to the general test noted above all bars which showed a fair retentivity and a coercive force above $Hc = 30$ were subjected to a test for their stability, or their ability to hold residual magnetism after shock. For this test the bar was subjected to the maximum magnetizing force of $Hc = 200$, and the retentivity noted when this magnetizing force was reduced to zero. The bar was then removed from the machine, subjected to the disturbing factor, and the retentivity again recorded after reinserting the sample in the permeameter. The two means commonly employed in practice for seasoning permanent magnets were used, namely, jarring or rapping the bar and boiling. For the latter treatment each bar was inserted in a piece of wrought-iron pipe, which served as a shield to prevent the mutual influences of the magnetic fields and boiled in water for three hours.

A large number of comparative trials of the rapping and boiling treatments showed them to be practically equal in their effect, and because of the simplicity of the former it has been resorted to in the later tests. However, should the material be very brittle it would be safer in practice to use the boiling method.

As indicating the efficiency of the rapping treatment there are noted below the results of successive jarring upon the retentivity of a bar of composition 7 Cr 3W 2Mo.

H $(MAX) = 200$.
B $(MAX) = 11{,}100$.
B $(RET) = 7500$.
B $(RET) = 4900$ (removed and rapped 1).
B $(RET) = 4700$ " " " 2).
B $(RET) = 4700$ " " " 3).

As will be seen, but one vigorous rapping is necessary, since this reduces the residual magnetism to a value which is but little changed by further shock.

Only the general summary of the tests is given, since it was not deemed advisable in a paper of this character to include the complete tabulation of the data. In Table I are noted the data of the stability tests which, as mentioned previously, include only those materials best suited for permanent mag-

nets. The alloys are divided into five groups according to their content of chromium, manganese, molybdenum, nickel and tungsten, either alone with iron, or together with iron and any other elements except the above five. in addition, a sixth miscellaneous group includes materials of various compositions in which more than one of the above five metals are present.

The results as a whole showed a wide variation in the physical and magnetic hardness of the various bars, not only before quenching, but afterward as well. Also, it appeared that the effect of quenching was varied in its results, with either increase or decrease of coercive force, or with no material alteration. However, it appeared that where marked changes were the result it was generally with an increase of magnetic hardness.

Chromium.—In the chromium group, it appears that this element alone, even in percentages as high as 16.65, does not result in bars of high coercive force suitable for permanent magnets, even after quenching at 900° C. And the coercive force was no higher with large than with small chromium additions.

A large number of chrome-silicon alloys were tested, and in practically all cases the bars were relatively soft and of low coercive force in the forged condition. The coercive forces in all but two instances were well below 30, and the alloys were thus unsuited in this state for permanent magnets. In these two instances but little change resulted on quenching; but for the other bars quenching resulted in a marked increase of coercive force, and with the accompanying relatively high retentivities these materials were well adapted for permanent magnets. The best alloys seemed to have about a medium chromium content (5 per cent or 6 per cent) since then the retentivity was high and the coercive force great enough to insure stability.

The addition of vanadium also seemed to be very beneficial, although in this case the raw forged bars were especially hard, and the coercive forces were high initially, and suffered but little change in quenching. Here, again, the best results seem to be gained with medium chromium content because of the higher retentivities.

In a few instances carbon was present, together with vanadium and silicon. The materials showed hardening upon quenching, but the magnetic results were no better than where there was no carbon.

Summarizing, it may be said that the addition of chromium alone to carbonless iron will not result in a material suitable for permanent magnets, but that the further addition of 0.75 per cent to 1 per cent of silicon or 0.3 per cent to 0.5 per cent of carbon to 5 per cent or 6 per cent of chromium is highly beneficial and gives an alloy very suitable for this class of service.

Manganese.—According to the classification adopted, the only manganese alloys falling in this group were of a series in which manganese alone in varying amounts had been added to the iron. In the raw forged condition the coercive force increased, with a corresponding decrease of retentivity, for each increase of manganese content until at 10.41 per cent Mn the alloy was non-magnetic. A bar with 10 per cent of manganese had the high coercive force of 90.1, but the rententivity was too low for permanent magnets. The most satisfactory bars were two with 4.51 Mn and 6.0 Mn, where the retentivities were fairly high and the coercive forces great enough to insure stability.

But three bars were tested after quenching, and the result was a decrease of coercive force; a peculiarity corroborated by other tests upon this series.

Molybdenum.—Of the entire set of results for this series of alloys only one was magnetically hard in the forged condition, this was a bar of composition 8Mo, 0.3V, 0.6C, which had the high retentivity of 9,200 and the fair coercive force of 39.0. In the quenched condition there was practically no change in any of the carbon-free alloys. But in four cases where carbon was present the quenching resulted in physical hardening of the bars and they became very fine materials for permanent magnets, with exceptionally high retentivities and very good coercive forces. In one case a bar with 4Mo and 0.6V attained a good coercive force; whether this was due to absorbed carbon is problematical. If the magnetic hardening took place without the aid of carbon it would indicate that the proper conditions of temperature, etc., might have the same effect on the other molybdenum alloys. This is merely conjecture, however, and the tabulated results showed no hardening effect of vanadium, such as was noted for the chromium alloys.

Nickel.—In the binary alloys of nickel and iron, increase of nickel caused a decrease of retentivity and increase of coercive force until at about 26 per cent of nickel the material was non-magnetic. Beyond this percentage the magnetic properties were restored, the alloys being of rather high permeability and low coercive force. Quenching at 900° C. invariably greatly decreased the coercive forces, so that none of the binary nickel alloys can be considered suitable for permanent magnets in the quenched condition, either because of low coercive force, or because of low retentivity where the coercive force is ample.

The addition of vanadium resulted in increased magnetic hardness, which was again increased by quenching. The most satisfactory bar was one with 10 Ni, 0.6 V, which in the forged condition had a retentivity of 9900 and a coercive force of 32.6; both values were slightly lowered by quenching. One bar with 10 Ni, 2 Si, 1 C had a fair retentivity of 6800 with a high coercive force of 62.0 in the forged condition, but after quenching it became almost non-magnetic.

The nickel alloys as a whole, therefore, cannot be regarded as satisfactory materials for permanent magnets. In this case, as noted for chromium, the vanadium seems to act as a hardening agent.

Tungsten.—Seven binary alloys of tungsten and iron were recorded, with a tungsten content varying from 0.93 per cent to 23.87 per cent. In no instance was the coercive force high enough to make them suitable for permanent magnets; and quenching had but little effect, usually resulting in a decrease of coercive force. For the two alloys where there was an addition of vanadium, one bar with 4 W 0.4 V became, after quenching, an exceptionally good permanent magnet, with a retentivity of 11,300 and coercive force of 43.8; on the other hand, another sample with 7 W, 0.3 V experienced but slight increase of hardness. A bar with 5 W, 0.5 C became noticeably harder after quenching, with increase of retentivity and coercive force to the high values of 11,700 and 59.0. Four other bars were very good for permanent magnets as forged, and there was a marked hardening due to quenching.

Summarizing, the binary alloys of tungsten and iron are not satisfactory for permanent magnets. The addition of vanadium may be beneficial, and the addition of carbon certainly is. There were in this series some highly satisfactory alloys; of those worthy of particular mention, however, all but one contained carbon.

Miscellaneous.—This miscellaneous group included such a large number of combinations of these several self-hardening alloys that a systematic discussion of the results is difficult. Of the 51 bars here grouped, almost two-thirds were good materials for permanent magnets. The effect of quenching was somewhat varied; in 13 instances there was a considerable increase of coercive force, in 30 cases but little change and in six instances a marked decrease. In four cases bars which were originally poor became of fair quality after quenching, and in numerous instances the quality of fair materials was greatly improved, in no case, however, has quenching resulted in a bar of fair quality becoming poor material for permanent magnets, although at times the quality was better before quenching than after.

One of the most striking features noted was the detrimental effect of the additions of nickel with its tendency toward non-magnetic alloys. In all cases where the material was classed

as non-magnetic, or where it was worthless for permanent magnets because of very low retentivity, there was a considerable amount of nickel present. Of the 17 bars of poor quality for permanent magnets, 12 had nickel, and of the five bars without nickel the quality was poor not because of a tendency toward being non-magnetic, but because the coercive force was too low. Several bars of nickel with chromium, with molybdenum, or with tungsten, had fair retentivities together with high coercive forces.

The various combinations of chromium with molybdenum or tungsten, either alone or with additions of silicon, vanadium and carbon, gave generally satisfactory results. In fact, it is difficult to point out any preferable combinations of these various self-hardening elements beyond the caution to keep nickel out of the alloy.

Of particular quality may be mentioned the following:

Bar.	Composition.	B (Ret.)	He.	
136M	5 Cr, 3 Mn, 4 Mo, 1 Si	11,350	55.8	As forged.
136V	10 Ni, 5 Mo	10,150	36.7	As forged.
150C	2 Cr, 10 Mo, 0.5 C, 0.3 V	10,500	78.0	Quenched.
155Y	10 Ni, 5 W	10,300	35.2	Quenched.
156A	1 Mn, 2 Mo	10,500	39.6	Quenched.
155P	1 Mn, 10 Mo, 0.5 C	10,750	58.4	Quenched.

H(Max) = 200 PERMANENT MAGNETS—STABILITY TABLE 1.

Bar.	As forged after rapping.			Quenched 1000° C after rapping.		
Chromium. Composition.	B(Ret).	B(Ret).	H(c).	B(Ret).	B(Ret).	H(c).
133E 4 Cr, 0.84 Si	15050		14.9	9000	3500	41.5
133D 5 Cr, 0.84 Si	7500	3300	41.0	9500	4800	41.8
133B 5 Cr, 1.4 Si	15700		16.2	8800	3900	46.0
160J 5 Cr, 0.3 V	7400	3400	53.0	9000	2700	42.2
160H 5 Cr, 0.5 V	9000	3600	50.5	9100	4200	45.0
160K 5 Cr, 1 V	8400	4100	48.1	9200	4900	49.0
133L 6 Cr, 0.4 Si	15650		14.9	9600	3700	49.0
133K 6 Cr, 0.84 Si	15300		13.6	9400	5200	52.3
147Z 6 Cr, 1.3 Si	9200		16.2	8800	3400	50.0
133F 6 Cr, 1.68 Si	14300		17.9	8300	3800	50.0
133M 8 Cr, 1 Si	12550		16.2	8700	2400	48.5
111S 10 Cr, 2 Si	10000		19.2	7100	3600	54.0
155K 10 Cr, 2 Si, 0.3 V, 0.6 C	11200	2300	36.5	3000	1600	69.0
137L 10 Cr, 2 Si, 2 Ti	10850		16.4	7500	4200	62.0
122E 10 Cr, 3 Si	10800		12.3	7400	3350	47.8
160I 10 Cr, 0.3 V	7500	4000	72.5	7800	3500	58.0
155L 10 Cr, 0.3 V, 0.65 C	4500	3100	84.6	7200		91.9
160G 10 Cr, 0.5 V	7000	3600	65.0			
163O 10 Cr, 1 V	7600	3500	62.8	8400	4500	54.5
141C 10.3 Cr, 1 V, 2 Si, 0.3 C	9500		20.9	8000	2600	38.0
133N 13 Cr, 1.8 Si	12750		16.7	7100	4000	62.8
136H 17 Cr, 1.5 Si	2100	1100	89.3	4900	2700	85.0
Manganese						
170D 6 Mn	8500	4000	50.0			
Molybdenum.						
160E 4 Mo, 0.6 V	12950		15.3	10400	2600	33.3
156E 8 Mo, 0.3 V, 0.6 C	9900	3500	37.8	12000	5000	52.2
160C 10 Mo, 0.3 V, 0.4 C	13700		23.7	10400	4900	55.8
163H 10 Mo, 1 V, 0.4 C	10850		25.1	9400	5000	53.5
141F 10 Mo, 1.2 C				10100	4000	37.0
Nickel.						
160M 10 Ni, 0.6 V	9700	3400	30.8	9700	4000	27.2
146P 10 Ni, 2 Si, 1 C	6700	3200	58.7	1000	900	58.0
Tungsten.						
160F 4 W, 0.4 V	8250		12.4	11300	3300	37.5
163J 4 W, 0.4 V, 0.5 C	11200	3000	34.8	8800	4100	67.5
163D 4 W, 0.4 V, 0.6 C	10400	2900	33.0	9200	4600	68.2
163K 4 W, 0.4 V, 0.8 C	8900	2800	32.5	6600	4400	70.6
155X 5 W, 0.5 C	10350		24.5	12000	5200	52.5
WIC (6 W + C)(?)	12500	2700	28.0			
163E 7 W, 0.3 V, 0.6 C	10900	3800	38.0	9600	5600	81.0
Miscellaneous.						
150C 2 Cr, 10 Mo, 0.5 C, 0.3 V	6900	3600	58.0	10700	6300	77.2
155S 10 Cr, 5 Mo	8700	4000	59.0	7300	3600	70.5
155U 2 Cr, 10 W	10100		23.4	9500	3900	49.0
156F 3 Cr, 8 W	9400		24.1	9900	4700	47.5
163A 5 Cr, 2 W, 0.6 C	6900	3800	61.0	7300	2500	55.0
163G 5 Cr, 5 W, 0.6 C	3400	1700	68.5	6400	4200	85.0
163C 10 Cr, 2 W	7400	4200	71.7	7500	4200	62.0
156K 10 Cr, 2 W, 2 Si	5800	3200	76.6			
156J 10 Cr, 2 W, 0.6 C	4900	3000	89.9	7100	4800	95.0
156H 10 Cr, 2 W, 0.3 V	6900	3700	66.6	6900	3800	61.4
156I 10 Cr, 2 W, 0.3 V, 0.6 C	3300	2500	82.7	5900	3900	65.8
155V 10 Cr, 10 W	5800	3300	80.5	6500	3600	72.7
150B 2 Cr, 10 W, 10 Mo, 0.3 V, 0.4 C	6100	3500	66.5	8400	3700	59.0
150D 2 Cr, 16 W, 5 Mo, 0.3 V, 0.5 C	5700	3600	64.0	8600	5000	63.5
163B 7 Cr, 3 W, 2 Mo	7300	4100	82.5	7400	2500	47.5
156G 10 Cr, 2 Mo, 2 W	7100	4100	78.5	6600	3800	65.8
156M 10 Cr, 2 W, 2 Mo, 0.6 C	4300	2500	82.0			
156P 10 Cr, 2 W, 2 Mo, 2 Si, 0.3 V	8500	1900	31.0	6800	2700	60.0
136M 5 Cr, 3 Mn, 4 Mo, 1 Si	10800	5300	64.7	5800	2900	72.2
140Q 5 Cr, 6 Mo, 10 Mn, 10 Ni, 1 C	10100	4000	38.2	7000	4400	85.3
142Z 2 Cr, 10 Ni	8100	3200	36.9	8200	3300	29.0
154E 5 Cr, 10 Ni	9100	4000	57.7	5400	2900	47.0
154F 7 Cr, 10 Ni	4700	2500	64.0	2000	1100	55.7
154A 10 Cr, 8 Ni	1600		80.8	800	300	73.5
136V 10 Ni, 5 Mo	9600	2300	36.0	8600	3600	32.0
146Q 10 Ni, 10 Mo, 1 C	5500	3000	69.7	5500	2800	70.4
146S 10 Ni, 10 Mo, 0.5 C	8900	4900	61.0	8400	4000	52.0
152Z 10 Ni, 5 W	8400	3100	36.0	8600	2800	29.7
155Y 10 Ni, 5 W	9050		17.0	10900	4600	38.0
155AB 15 Ni, 5 W	8300	3000	38.5	8000	2300	37.8
156A 1 Mn, 2 Mo	9450		27.3	10600	4600	38.5
155P 1 Mn, 10 Mo, 0.5 C	6900	3600	61.0	11200	5400	57.5
156C 2 Mn, 5 Mo, 0.6 C	9500	5500	57.2	8400	4700	68.0

Strange to say, two of these alloys have a high nickel content, but they may be put down as exceptions to the general rule. The values given above indicate a very high standard of material for permanent magnets.

Summarizing, it may be deduced from the above tests that chromium, manganese, molybdenum and tungsten are the important additions in the manufacture of steels for permanent magnets, but the presence of some third element is necessary. While carbon is beneficial, it is possible to obtain highly satisfactory materials either by various combinations of the above four elements or by suitable additions of silicon or vanadium, the latter, in particular, being very desirable. Also, suitable materials may be made which will require quenching, although in this case carbon is most beneficial, or, if desired, equally satisfactory results may be obtained without.

Stability.—As mentioned previously in the general discussion of the tests for permanent magnets, a better indication of the value of material for this service is obtained by a stability test which gives the residual magnetism after shock. In our tests the rapping method was used, the details being given above. Those bars were tested which showed a coercive force above 30, either before or after quenching, this being considered the minimum value at which permanence might be expected.

The details are tabulated in Table I in the same order as used in the general discussion. The figures of interest are those for the retentivity, B (Ret), after rapping; they show the marked decrease of the residual magnetism due to shock, a decrease averaging about 50 per cent, but varying according to the coercive force of the material under test.

For purposes of comparison, and for illustrating the above points, the following tables have been prepared. For Table II

Table II.

Bar.	Composition.	B(Ret).	H(c).	
WIC	(6 W + C) (?)	13,450	30.0	As forged.
156E	8 Mo, 0.3 V, 0.6 C	11,750	53.7	Quenched.
155X	5 W, 0.5 C	11,700	59.0	Quenched.
136M	5 Cr, 3 Mn, 4 Mo, 1 Si	11,350	55.8	As forged.
160P	4 W, 0.4 V	11,300	43.8	Quenched.
163J	4 W, 0.4 V, 0.5 C	11,050	38.0	Forged.
155P	1 Mn, 10 Mo, 0.5 C	10,750	58.4	Quenched.
155K	10 Cr, 2 Si, 0.3 V, 0.6 C	10,700	32.9	As forged.
163D	4 W, 0.4 V, 0.6 C	10,650	35.5	As forged.
150C	2 Cr, 10 Mo, 0.5 C, 0.3 V	10,500	78.0	Quenched.
156A	1 Mn, 2 Mo	10,500	39.6	Quenched.
163E	7 W, 0.3 V, 0.6 C	10,450	39.4	As forged.
160E	4 Mo, 0.6 V	10,400	35.0	Quenched.
160C	10 Mo, 0.3 V, 0.4 C	10,350	59.8	Quenched.
155Y	10 Ni, 5 W	10,300	35.2	Quenched.
136V	10 Ni, 5 Mo	10,150	36.7	As forged.
141F	10 Mo, 1.2 C	10,000	37.4	Quenched.

Table III.

Bar.	Composition.	B (Ret) After Rapping.	Hc.	
150C	2 Cr, 10 Mo, 0.3 V, 0.5 C	6300	77.2	Quenched.
163E	7 W, 0.3 V, 0.6 C	5600	81.0	Quenched.
156C	2 Mn, 5 Mo, 0.6 C	5500	57.2	As forged.
155P	1 Mn, 10 Mo, 0.5 C	5400	57.5	Quenched.
136M	5 Cr, 3 Mn, 4 Mo, 1 Si	5300	64.7	As forged.
155X	5 W, 0.5 C	5200	52.5	Quenched.
133K	6 Cr, 0.84 Si	5200	52.3	Quenched.
150D	2 Cr, 16 W, 5 Mo, 0.3 V, 0.5 C	5000	63.5	Quenched.
163H	10 Mo, 1 V, 0.4 C	5000	53.5	Quenched.
156E	8 Mo, 0.3 V, 0.6 C	5000	52.2	Quenched.

the data have been taken from general tests and there are listed those bars having a retentivity above B(Ret) = 10,000 and a coercive force exceeding H(c) = 30.0, the order being accord-

ing to the values of B(Ret). In Table III the selections have been from stability Table I, and are ranked on the basis of the values of B (Ret) after rapping.

As will be seen by comparison of the two tables there are bars listed in Table III which do not appear in Table II. Also the order of rank is not the same for those materials which are noted in both tables. It will be noted, likewise, that some of the alloys which have high rank in Table II are low down in Table III, or do not appear at all, because the coercive forces are not great enough to retain a high residual magnetism after shock. All materials listed in Table III have high coercive forces, above $H_c = 50$, and it would appear that for permanent magnets to give good service it should reach this value.

As a result of these tests it appears that for the investigation of materials for permanent magnets this stability test gives a very satisfactory indication of the quality. In addition, of course, it would be necessary to test the ageing qualities, or the ability to retain this residual magnetism for long periods of time, and under working conditions.

Northern Chemical Engineering Laboratories,
Madison, Wis.

Temperature Drop, a True Measure of Thermal Resistance.

By Carl Hering.

In the calculations of the heat losses through the walls of furnaces, and in fact in most calculations of flows of heat, it has been the custom, dictated perhaps more by usage than by reason, to use the thermal conductivity of the material as the basis of the calculations, instead of the resistivity, although the latter is the more general practice in electrical calculations. There seems to be no good reason, however, for making this difference; on the contrary, it would probably be much better to follow electrical practice. The writer has repeatedly recommended making the change, and has often used thermal resistivities to advantage in place of conductivities. In an article in this journal, January, 1909, page 11, he showed that for the very cases occurring in electrical furnaces in which the walls are made of layers of different materials, it is much simpler to use resistances than conductances.

Besides the difficulty in overcoming what might be called mental inertia—that is, changing our accustomed methods—the chief objection to using thermal resistivities seems to be the lack of a named and easily understood unit for representing those quantities numerically. There is a natural and excusable hesitancy to use, in important calculations, units or quantities with which we are not familiar and about which there is a mistiness or an apparent mystery. Just as we do not like to purchase the proverbial cat in the bag, so we do not like to deal with units when we do not know what they mean or cannot refer to them by name.

Some time ago the writer showed that thermal conductivities may be correctly and rationally expressed and measured in terms of watts for either an inch or a centimeter cube and for centigrade degrees. Thus the thermal conductivity of graphite is about 3 watts for an inch cube; that of copper is about 10 watts, etc. Moreover, for electric furnace calculations in which the production of heat energy is naturally expressed in watts that unit is by far the simplest and most convenient one to use.

The physical significance of this unit is that, if the difference of temperature of 1 deg. C. be maintained between the two opposite faces of an inch (or centimeter) cube of graphite, there would be produced a flow of heat through it equal to 3 watts (assuming that there is no lateral loss), that is, the amount of heat flowing out of the cooler end would be equal to that set free by a coil of wire consuming 3 watts and setting it free as heat. Hence for thermal conductivities we now have a very convenient, simple, rational and easily understood unit with which we are already familiar, and one which has the additional advantage that it was born with a familiar name; hence, we need not even learn a new name. It was not necessary to create it, it required only to discover it.

The purpose of the present article is to show that there also exists an equally simple, convenient, rational and easily understood unit of thermal resistivity, which like its mate also required merely to be discovered and not created, and already had a name, being a well-known quantity with which we are all familiar, namely, a drop in temperature.

The drop of temperature in degrees can be shown to be a true measure of thermal resistivities, just as watts are a true measure of thermal conductivities. Moreover, these two units are the correct mates of each other, because the reciprocal of the numerical value of a conductivity expressed in watts is equal to the corresponding numerical value of the resistivity of that material expressed in degrees, and vice versa.

There is, therefore, no longer any good reason why we should not hereafter use resistivities instead of conductivities in thermal calculations, and there are often good reasons why we should do so. Tables of constants may also now be given quite as well in resistivity units as in conductivity units.

The physical meaning of this new unit of thermal resistivity is that if its value is, say, n°C., it means that it requires a difference or drop of temperature of n°C. between the two opposite faces of a cubic inch (or centimeter) to cause a flow of 1 watt of heat.

If, for instance, the thermal conductivity of graphite is 3 watts in centigrade-degree and cubic-inch units (or 1.18 in. cb. cm. units), its thermal resistivity will be 0.333° C. in cubic-inch units for 1 watt flow (or 0.848° C. in cb. cm. units), the two values being the numerical reciprocals of each other, as they should be in any rational system.

The electrical analogy would be to express electrical resistivities in terms of the difference of potential in volts required to cause 1 amp to flow through a cube of 1 in. This could be used quite as well as the more usual ohms for 1-in. cube. The higher the resistivity the higher this number of volts or degrees, and in exact proportion.

The watt and the degree are, therefore, convenient, simple and rational units for expressing and measuring thermal conductivities and resistivities respectively. But there are still wanting similarly convenient and simple units for expressing and measuring thermal conductances and resistances. The differences between these quantities are unfortunately not always clearly understood. Conductivities and resistivities are specific quantities; like specific gravity, specific heat, etc., they refer to some unit quantity of the material and are, therefore, physical constants, or properties of the materials, and are independent of the actual size or shape of a piece of the material. Conductances and resistances, on the other hand, refer to actual pieces of the materials of a given size and shape; they represent the amounts of these properties possessed by the pieces of the given sizes and shapes, and they are, therefore, dependent on the size and shape. Just as the total weight of a body depends on its size, while its specific gravity (the weight of a unit quantity in terms of water) is independent of its size, so the conductances and resistances depend on the actual size and shape while the conductivities and resistivities (or specific conductances and specific resistances) are independent of the size or shape.

The electrical resistance of a given piece of material is measured in ohms, while the electrical resistivity of that material is measured in the ohms which 1-in. cube has. In a precisely analogous way it is quite proper to measure a thermal resistance in degrees of temperature, when a thermal resistivity of the material can be measured in degrees for 1-in cube. Hence, the total thermal resistance of a given furnace wall or a whole furnace may also be correctly and rationally expressed and measured in temperature degrees, strange as it may at first seem.

The meaning of such a thermal resistance expressed in degrees is simply that that is the number of degrees drop of temperature which would cause a flow of heat of 1 watt through that resistance, say, of a particular furnace wall or whole furnace. Hence, to determine the actual heat current or flow in watts through this resistance, we must divide the actual temperature drop for the furnace by this resistance (also in degrees), just as with electricity, to determine the current C, we divide the actual drop of voltage E by the resistance R, according to Ohm's law.

It is, therefore, quite correct and proper to express a thermal resistance in degrees and to use this quantity in subsequent calculations just as we use ohms in electrical calculations. The results will be quite correct numerically.

Quite similarly, thermal conductances can be correctly and properly measured and expressed in watts, and this number of watts can be used in calculations just as electrical conductances (in mhos) are used. The physical meaning of a thermal conductance in watts, say, of a complete furnace or any part of it, is that this is the number of watts which would flow when there is 1° difference of temperature between the two end surfaces. Hence, to get the actual flow in any given case, multiply this conductance in watts by the actual existing drop in temperature, just as, in accordance with Ohm's law, to get the flow of current C in any given case we multiply the conductance K by the drop of voltage E.

The thermal resistance and conductance of a furnace or any part of it are fixed quantities like its dimensions, and are independent of any flows of heat; just as the electrical resistance and conductance of a wire are fixed quantities independent of the current which may or may not flow. The currents of heat or electricity will be dependent upon the drop of temperature or voltage, and are determined by Ohm's law.

Of course, in so far as the temperature coefficients of the thermal resistivities or conductivities affect the values of the latter, the proper corrections must be made if they are sufficiently great to become important. But here the electrical analogy still holds good, because when a current flows it raises the temperature and this in turn changes the resistivity; whenever this is of importance (as in incandescent lamps, for instance, where this change amounts to about 50 per cent) it must be allowed for. But this is of the nature of a correction factor and should not be permitted to destroy or to cloud the very interesting and valuable result given us by this very close analogy, or the fundamental laws involved.

The advantage of using the thermal resistance and conductances in furnace calculations, instead of the heat losses through them, is that it simplifies the calculations, and in some cases even makes a direct calculation possible where troublesome trial calculations would otherwise have to be used. Furnace walls, made as usual of layers of different materials, are an illustration. The thermal resistance of each layer is calculated by itself; the total is then equal to their sum, and from this the heat losses can finally be calculated; the drop of potential in each layer is then determined from its resistance and the total flow, quite like the drop of voltage in various electric conductors connected in series. To make such a determination with conductances alone would be very tedious, if indeed possible at all.

It seems unnecessary to accompany the above with formulas and examples, as the calculations are in every respect quite analogous to the corresponding electrical ones, and are all extremely simple. One need only remember that thermal conductivities are expressed in watts for 1 in. (or centimeter) cube and 1° drop; that conductances are expressed in watts for 1° drop; that thermal resistivities are expressed in degrees drop for 1-in. cube and 1-watt flow; that resistances are expressed in degrees drop for a flow of 1 watt; and that the numerical values of the resistivities and corresponding conductivities are then the reciprocals of each other, as are also the values of the resistances and conductances.

The latter, being the important connecting link in this new system of units may be further illustrated by the following example. Inch units and centigrade degrees are implied.

Let the thermal conductivity of graphite be 3 watts. Assume a piece 6 in. long and 4 sq. in. section. The conductance of this piece will then be $3 \times 4 \div 6 = 2$ watts, which means that a heat current of 2 watts would flow for 1° drop between the ends. If this system of units is correct the reciprocal of this conductance must be the resistance in degrees, hence equal to 0.5° C. This means that the drop over the whole length would have to be 0.5° for a flow of 1 watt. In the above calculations it was found to be 2 watts for 1°; hence, it will be 0.5° for 1 watt; the resistance is, therefore, the reciprocal of the conductance. If the resistance is 0.5° the resistivity must be: resistance $0.5° \times$ section $4 \div$ length $6 = 0.333°$, which, it will be seen, is the reciprocal of the conductivity, 3 watts.

It would, of course, be more rational to follow electrical practice and create two new thermal units, one for resistance (and named the thermal ohm*) and the other for conductance (and named the thermal mho*) based on the two quotients of the two variables, watts and degrees, than to use those described above, which are based on making one of them unity so that the other one is numerically equal to their quotient. But until such a rational system has been devised and adopted the units here deduced will at least serve our purpose, even though it may at first require some mental effort to overcome one's natural dislike to measure a resistance in thermometer degrees and a conductance in power units.

There are still other resistance and conductance units which could be similarly deduced as, for instance, a length under unit conditions of flow, temperature and section; or a section under unit conditions of flow, temperature and length. They might even have some advantages over those recommended in this article. But the valuable analogy with Ohm's law would then appear to be destroyed, and this would require new laws to be formulated and learned.

It may be of interest to recall here the statement that Dr. Ohm based his famous electrical law on the corresponding and older one which had been found to be true for thermal flows.

Philadelphia, Pa.

Zinc Ore Dressing in Colorado—III.

By H. C. PARMELEE.

The Creede District.

Among the mining camps of Colorado none has had a more rapid and spectacular rise and fall than that of Creede in Mineral County. Practically unknown prior to 1890, the district had a railroad in 1891, experienced a boom in 1892, with an output of over $3,000,000 for the year, and collapsed in 1893 with the sudden fall in silver. Since that time the district has had varied fortunes and is now among those in which the greatest activity is confined to the dressing of base sulphide ore containing zinc and lead. In this district the mineral usually occurs in compact quartz veins, carrying galena and blende, the latter in the varieties known as rosin and black jack. Lead carbonate also occurs with the galena in some of the ores. Silver minerals and gold in small quantities are also common constituents.

The Amethyst Mill.

Of the three mills now in operation near Creede the Amethyst is probably the most interesting. The crude ore treated there averages about 5 per cent zinc and 10 per cent lead, with some silver and free gold. The silver minerals are closely associated with the silicious gangue, and there is a consequent heavy loss of silver in the tailing. The gold occurs free and may be seen on some of the tables treating the finer sizes of pulp. The lead and zinc minerals occur free enough to warrant coarse concentration with jigs, followed by intermediate treatment on

*These two very appropriate names were first suggested by Dr. A. E. Kennelly.

tables, finishing with slime. Quite an extensive plant is installed to save the latter. The average saving of the total value is nearly 85 per cent, the highest recovery being made on lead, next on zinc, gold and silver, the last suffering the greatest loss.

Ore is delivered at the head of the mill by an aerial gravity tramway, 1700 ft. long, connecting mine and mill. The line has 18 buckets carrying about 500 lb. of ore each, moving at about 200 ft. per minute, so that a bucket is dumped a little oftener than once a minute. This gives a capacity of about 125 tons in eight hours, which is the capacity of the mill for 24 hours. The tramming and crushing department is, therefore, operated only one shift.

At the tram loading station at the mine the ore is roughly sized by being passed over a grizzly with 1½-in. openings. These two classes are separately trammed and dumped into separate bins at the head of the mill, the coarse passing over a second grizzly before going to the crusher, and the fine going

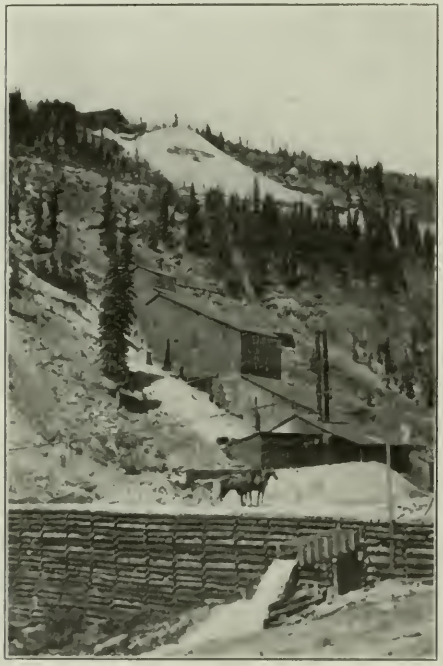


FIG. 1.—THE AMETHYST MILL DISTRICT, CREEDE, COLO.

directly to the crushed ore bins. The bin capacity for crushed ore is 75-80 tons, or sufficient to run the mill two shifts.

Coarse ore is crushed through a jaw crusher reducing to 1½-in. size. From the crushed ore bin the ore is fed by automatic shovel feeder to the coarse rolls (14 in. x 27 in.), water being added as the ore leaves the feeder. The coarse rolls reduce the ore to about ½ in., and the pulp is then elevated by a 10-in. B. & B. elevator to the trommel screen line of four 3 ft. x 6-ft. trommels fitted with woven wire screen of 4, 5, 8 and 12 mesh. The first half of each trommel, except the last, is fitted with screen of larger wire than the second half, as shown in the table on top of next column.

The oversize of No. 1 trommel is returned to finishing rolls for regrinding, thence to the elevator and screen line. The undersize is passed along through the screen line, the oversize of the three succeeding screens being jigged, and the undersize of the last screen being classified for fine jig and table treatment, the coarsest class going to a jig and the others to tables.

Mesh and Wire of Trommel Screens.

Trommel No.	Mesh.	Wire Gage. 1st half.	2d half.
1	4	11	13
2	5	13	15
3	8	16	18
4	12	20	20

Jigging thus commences on —4+5-mesh material and continues to the coarsest size in the —12-mesh pulp. It is probable that jigging could be continued on smaller sizes, and less coarse work done on tables, as the minerals are quite free, especially the lead, and would make good jig products. There are four four-compartment jigs of the Harz type, with hutch discharge only. The first three are fed with screen oversizes as described before, and the fourth with the coarsest of the classified pulp. Lead concentrate is made in the first two compartments of all jigs, and middling in the third and fourth. Only jigs 1 and 2 (coarse) have tailing discharge which goes to waste, while jigs 3 and 4 (fine) are run without screens in the third and fourth compartments, so that the tailing water is almost clear and is sent to the slime plant.

Jig Data.

Jig. No.	Speed.	Mesh and Wire Gage in Compartments. 1.	2.	3.	4.
1	188	5-15	5-13	5-15	5-15
2	240	5-13	5-13	5-15	5-13
3	270	8-16	8-16	no	no
4	270	8-16	8-16	screen	screen

The middling from the third and fourth compartments of all jigs is reground in Chilian mills, of which there are two, 5 ft. in diameter, running at 30 r.p.m., fitted with 16-mesh woven wire screens. The roll rings are renewed every 80 days and the dies every 160 days, one die serving for two sets of rings. The Chilian discharge is elevated and combined with the undersize of the screen for classification.

The classifier used at the Amethyst mill is of the type known as the trough or shallow pocket classifier. The trough is rectangular in cross-section and has a pitch of about 1 in. per foot. Shallow pockets are placed at intervals in the trough, according to the number of classes which is designed to treat and the location of the machines on which they are to be treated. The pockets have the effect of momentarily retarding the flow of pulp in the trough, so that a classification can take place either with or without the use of hydraulic water. At the Amethyst mill no hydraulic water is used, although it is available for use in opening plugged spigots.

The pulps from the undersize of the screen line (12 mesh and finer) and from the Chilian mills (16 mesh and finer) are combined in the trough classifier just described. There are nine pockets in the trough, the discharges from which feed No. 4 (fine) jig and eight tables.

Card tables are used exclusively in this mill, there being 12 on classified pulp and three on slime. Lead concentrate is a finished product on all tables, being cut clean and free from the zinc middling, which is retreated on other tables where both lead and zinc concentrates are made. It will be observed that no zinc concentrate is made in this plant on the jigs or until the retreatment tables are reached. All concentrates are laundered directly to dewatering bins at the lower end of the mill, so that no handling is required except in loading for shipment.

The precautions taken at this plant for the recovery of mineral in the slime are worthy of special mention, as the slime recovery is high. The slime from Nos. 3 and 4 jigs and from 9 of the 12 tables treating classified pulp is laundered to three 8 ft. Callow thickening tanks, the underflow of which is treated on three Card slime tables. The overflow of the Callow tanks, the slime from all the tables which do not discharge into the tanks and the overflow from the dewatering bins are combined

and treated in the canvas plant. This is a recent addition to the mill and comprises six 50-ft. canvas tables 3 ft. wide, built on a gentle slope and covered with 10-oz. duck. The feed is distributed uniformly over all, but can be diverted from any one during the operation of cleaning, which is accomplished by a spray of pressure water directed by the operator. The accumulated slime is thus washed into settling boxes, of which there

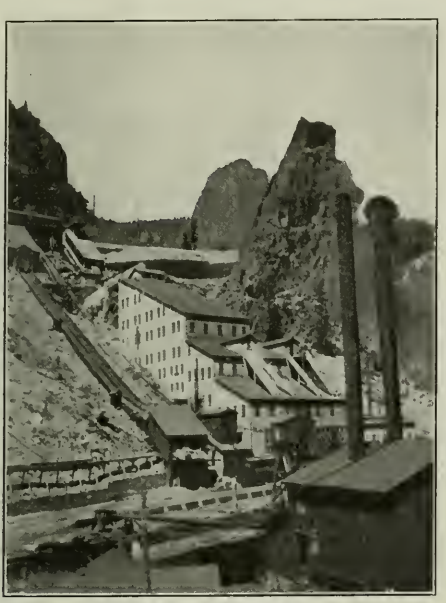


FIG. 2.—THE HUMPHREYS MILL DISTRICT, CREEDE, COLO.

are two sets arranged in compartments so that a compartment in one set can be cleaned out while the corresponding compartment in the other set is receiving the flow of slime from the tables. The overflow of these settling boxes is laundered to final deep settling tanks, which are cleaned but once in five months. One man is regularly employed on one shift to clean up the settling boxes by siphoning as much water as possible and then shoveling out the slime. It is dried in bins having steam pipes on the bottom and sides. As the slime is principally lead it is mixed with the lead concentrate for shipment.

The lead concentrate contains 70 per cent to 74 per cent lead; the zinc concentrate, 40 per cent to 45 per cent zinc, and the slime, about 35 per cent lead. The waste from the slime plant carries from 1 per cent to 3 per cent lead. No regular sampling of mine or mill products is done at this plant, the management satisfying itself with an occasional sampling of the entire mill.

Steam power with rope drive is used and has been found superior to the irregular electric power formerly available in the district. The power consumption is about 85 hp. From five to seven men are required per shift, in addition to the mining force and those engaged in hauling concentrates to the cars, a distance of about two miles.

The Humphreys Mill.

This mill is larger than the one just described, and if run at capacity would probably treat 250-300 tons per 24 hours; but being run at reduced capacity and for only one 10-hour shift per day, it is treating about 50 tons. The ore is practically the same as that treated in the Amethyst and the general process is the same. The equipment comprises crusher, coarse and fine rolls, automatic sampler, a double screen line of four trommels each, 12 four-compartment Harz jigs, three Chilian mills, trough hydraulic classifier and 12 Wilfley tables.

The jigs make lead concentrate in the first two compartments and zinc-lead middling in the last two. The latter are run with continuous hutch discharge, and with coarse screens and beds instead of without screens as at the Amethyst. The undersize of the screen line is not combined with the Chilian mill product, but each is treated on tables separately. Experiments have been made with various screens for the Chilian mills, trying different weaves of screen cloth and round punched sheet steel. The results indicate that while there is some preference as to weave of cloth, the round punched screen is superior to them, being more free in its action and having longer life.

The slime plant at this mill is not in commission. It contains reciprocating tables and several 80-ft. canvas tables similar to those at the Amethyst mill. The power consumed in milling is about 100 hp, which is generated by steam and transmitted by rope drive. The illustration shows the power house in the foreground, with rope drive extending across to the mill.

The Ridge Mill.

From the point of view of the nature of ore treated, and the high quality of zinc concentrate produced, this mill is one of the most interesting in the district. The same minerals occur in the Ridge ore as mentioned before, viz., galena, cerusite and rosin and black jack zinc. The ore is easily mined, and as practically all of the values are in a stiff clay the hard rock is sorted as waste and thrown over the dump. Large boulders of galena covered with zinc are frequently found in the clay.

The nature of the ore eliminates crushing, and the first step in the milling process consists in disintegrating the stiff clay in a revolving disintegrating screen of the American Concentrator Company type. As shown in the illustration (Fig. 3), which is a sectional view looking through the screen toward the discharge end, the machine consists practically of a revolving grizzly, jacketed with a screen and revolving in a trough of water. Heavy cast-iron hoods reduce the openings at each end, through one of which the ore is fed, and from the other the barren rock discharged. One-inch iron bars extend from end to end of the trommel and into the faces of the hoods. The screen jacket surrounds these bars and has 1¼-in.

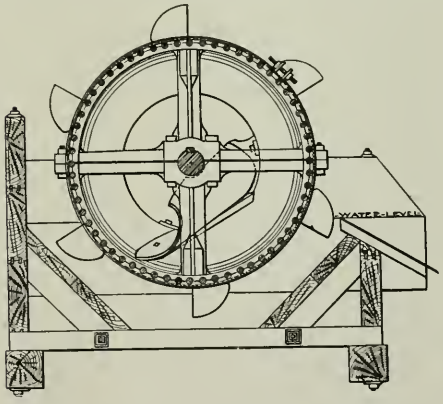


FIG. 3.—SECTIONAL VIEW OF DISINTEGRATING SCREEN.

round openings. On the outside of the screen jacket are bolted at intervals elevator cups, as shown in the illustration. The number of cups determines the quantity of feed and may be increased or decreased as needed. At the discharge end of the screen is a plow-shaped casting which, in its revolutions, picks up and discharges any pieces of rock which may have been fed with the clay. The rock is taken to the dump.

The ore, disintegrated by the revolution of the screen, falls through the openings into the trough, where it is picked up by the buckets and discharged onto an apron. It is then sized for jig and table work in the usual trommel screens to which it is elevated in the usual manner. The first trommel has 4-mesh screen cloth, the oversize of which is sent to rolls and again elevated. The undersize passes to a second trommel, which delivers two undersizes, one from each half of the trommel, and an oversize The upper half of the trommel has 8-mesh screen, and the lower half, 6 mesh. The 8-mesh undersize is hydraulically classified, the underflow going to the fine jig and the overflow to the tables. The 6-mesh undersize goes to the medium jig and the oversize to the coarse jig.

There are three four-compartment Harz jigs having 5-mesh, 6-mesh and 8-mesh screen on the coarse, medium and fine jigs respectively Hutch products only are made; lead concentrate in the first compartment, middling in the second, zinc concentrate in the third and middling in the fourth. Zinc concentrate is occasionally skimmed from the last compartment. The middling is returned to the elevator and rescreened without regrinding. The coarse, medium and fine jigs are run at 230 r.p.m., 240 r.p.m. and 260 r.p.m. respectively.

Three Wilfley tables of an early date are used and do good work. The first two make lead concentrate and middling, which latter is retreated on the third table where zinc and lead are separated. Most of the concentrating at this mill is done in the jigs, and the tables are not relied upon as much as in the other mills where no jig zinc concentrate is made.

Crude Ridge ore assays about 1.5 oz. silver, 0.01 oz. gold, 4 per cent to 6 per cent lead and 8 per cent to 12 per cent zinc. The proportion of the two base metals to each other is the reverse of that in the ore treated in the other mills.

The quality of the lead concentrate is indicated in the following assays of recent shipments:

Gold oz. per ton.	Silver oz. per ton.	Lead per cent.	Zinc per cent.
0.42	4.95	68.6	8.8
0.25	5.00	65.0	7.0
0.16	6.20	68.0	7.5
0.16	5.45	72.5	6.4

The zinc concentrate is of excellent quality, carrying from 50 per cent to 56 per cent zinc, 3 per cent lead, 0.05 oz. gold and 2.0 oz. silver.

The Ridge mill is provided with low-pressure water-power. The 20-in pipe line has a drop of about 100 ft., and water is delivered through a 3-in. nozzle to a 4-ft Pelton wheel at a pressure of 35 lb. per square inch. The mill has a capacity of 50 tons per 24 hours, but is run only one 10-hour shift and treats about 20 tons per day

The three mills of the Creede district are now treating about 200 tons of ore per day in plants having a capacity of nearly 450 tons. The ratio of concentration is about 8 or 10 into one. It is probable that the dressing of base ore will continue to be an important feature here, and with steady development underground will probably increase in magnitude One of the needs of the district is cheap and reliable electric power, such as is supplied to other districts in Colorado At present coal is expensive, especially when hauled from the cars to distant plants, the latter haul often adding 50 per cent to 100 per cent to the original cost of the coal.

Settling dust from furnace gases is accomplished principally by reducing the velocity of the gases and allowing sufficient time for the dust to settle. Reduction of the temperature is not a necessary element, in fact, a higher temperature might be of benefit as the specific gravity of the gases would be reduced more by the temperature than would that of the dust particles. It has been demonstrated that if the velocity of gases is reduced to about 3 ft per second there will be practically complete settling in less than 500 ft in about 8 minutes.

Corrosion of Metallic Food Containers.*

By Edward Gudeman, Ph.D.

With a product for analysis one will be satisfied to call it a c. p. product, if the amount of total impurities is 0.001 per cent or less. Atomic weights generally do not go beyond the second decimal place, and most chemists in figuring analytical results use only the first decimal figure. In food chemistry, however, a quantity of 0.001 per cent is often considered a tremendous quantity, especially if it is a metallic impurity or an antiseptic or coloring ingredient, and I cite some cases to give an idea how far some food chemists and food officials carry this "foolish accuracy," not to use a stronger term.

Within the last few weeks there was submitted to me from outside of this State (Illinois) some carbonated water, condemned as unwholesome and poisonous, due to its lead contents, and the distributers were given five days to take same out of the market. Analysis showed the amount of lead to be from 0.00003 per cent to 0.00005 per cent. Another similar product was found to be free of lead as it was put into the container, but contained about 0.0003 per cent lead after three days. The lead in this case came from the metallic button used to seal this special kind of bottle. In another case the trace of lead was found to be due to solvent action on the pipes used in the apparatus and the manufacturer was given 10 days to take out the whole machinery and replace it with such machinery as would give a lead-free product.

The stronger the corrosion of the metal, the quicker the solvent action. In the case where the metallic stopper was the cause of the lead entering the beverage it was found that the button used to hold the rubber cap averaged in new buttons about 18 per cent of lead. The old, corroded button was found to contain about 8 per cent of lead. The total amount of lead and tin that actually went into solution could not be exactly determined, as such buttons vary in weight, but it was strongly indicated that approximately twice as much lead went into solution as of the tin. It was also found that the solvent action of carbonated waters on lead alloys was about the same as on the lead itself.

As another case showing to what foolish extremes of accuracy some food chemists will go I cite the following: In one of our neighboring States the distributers of a food product were given 10 days' notice to clear the market of a food product said to contain a quantity of sulphur dioxide or its compounds. On request I investigated this matter, and the chemist who made the analysis stated as follows: 200 grams of the product were taken for analysis, using the official methods, the distillation was continued until the distillate amounted to 2500 cc. To obtain complete oxidation, with bromine, this was concentrated down to 50 cc, placed into a test tube and the sulphates precipitated with barium chloride. As no visible precipitate appeared, the solution was allowed to stand about 18 hours, the clear liquid carefully decanted and the bottom of the test tube examined with a magnifying glass, showing a thin film. The film was considered as barium sulphate due to the sulphurous oxide in the original product, and the product was condemned and the distributers and manufacturers given a short time to dispose of the stocks on hand.

Duplicating the work quantitively it was found that the film would approximate 0.000001 per cent of the amount taken for analysis. Under the official standard for sulphurous oxide, pertaining to imported beverages, a maximum of 50 milligrams free sulphurous oxide per kilogram is permissible

Parties have been cited into the courts to defend the presence of a faint trace of nitrogen oxide in starch. The testimony showed that the amount present was less than found in the atmosphere under normal conditions and the defendants were discharged

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*A paper read at the Chicago meeting of the American Electrochemical Society. An abstract of this paper was already given on page 635 of our November issue. But in view of the interest that attaches to the subject the paper is herewith printed in full.—Editor.

Within the last month Federal Inspection Decision No. 126 places the maximum amount of tin allowed in food products at 300 milligrams per kilogram being 0.03 per cent. Tin has been one of the substances most extensively used to protect iron from corrosion and until the issuing of the last Federal Inspection Decision was not legislated against; in fact, standards were accepted for the amount of tin required on tin plate to make it acceptable to be used for the manufacture of food containers.

Tin, copper, lead, silver and iron are very soluble metals (especially when traces are taken into consideration) in the liquids used with canned food products and in the peculiar liquids produced in the decomposition of foods. It is immaterial how the food itself is protected; it will undergo some decomposition, not necessarily putrefaction nor producing an unwholesome product. This decomposition is often a very slow one, not noticeable until months or years have passed since the product was put into the hermetically sealed container.

Among the two most commonly found products of decomposition are sulphuretted hydrogen and carbonic acid. Carbonic acid is one of the strongest corroding acids known to me, especially when acting under pressure on metals in the presence of water. Some 18 years ago I was interested in the process of directly corroding metal lead with carbonic acid. Treating finely divided lead with carbonic acid under pressure would change the lead into carbonate in a very short time, the process being now in use for the manufacture of carbonate of lead, white lead, direct from the metal. The finely divided lead or lead sheets are subjected to water or steam under a pressure of carbonic acid. Carbonated beverages are always under carbonic acid pressure varying from 5 lb. to 125 lb. per square inch, and the decomposition of food products giving rise to carbonic acid produces a pressure, and thereby solvent action is increased. This no doubt explains why food products placed in vacuum containers are less liable to deterioraton.

Tin is a good protecting medium for iron under two conditions: The coating must be perfect and the contact between the tin and the iron must be perfect. I have never seen a piece of tin plate where both of these conditions were met with, and very few where either of these two conditions was fulfilled.

There are four main reactions that take place with food products, due to the character of the foods themselves or to their decomposition under normal conditions of ageing. There is either oxidation or reduction, or both, and the action of acids or alkalis, or both. A perfect container will be one that will withstand all of these reactions.

One cannot tell with certainty which reactions will take place nor how they will take place nor when. Iron is porous and in tinplating the pores are not filled or coated. Tin also is porous, and this porosity must be taken into consideration. A perfect contact between the tin and the iron is of the greatest importance, using sufficient tin to coat all the superficial area of the iron. With such a tin plate, the corroding action on the tin need only be considered. After a large series of trials this object was attained, but the process is not commercially successful, as the cost of the container is a vital consideration, being about twice the value of the present tin can with its contents.

Iron and steel can be impregnated so as to have the resisting properties of gold. Steel springs, knives, surgical instruments, etc., look like gold and have the insolubility of gold, with all the characteristic properties of tempered steel. The process is simple. I made an amalgam of gold and impregnated the iron article with such liquid amalgam, then by heat vaporized the mercury and obtained an iron-gold article that withstood the corroding action of acids, exactly like gold. The amount of gold required was very small, about 2 oz. of gold to a ton of iron being sufficient, figuring this from the small experimental samples made.

The serious objection to this process was that the amalgam of gold had to be liquid and it was necessary to vaporize about 99½ lb. mercury for every ½ lb. of gold used, an amalgam of over ½ per cent not working satisfactorily. Experimentally the process is a success, but commercially the cost of operation is such that the cost of such a plate will be from 12 to 15 times the cost of a tin plate, making the cost of the gold-iron container exceed, often, the cost of the present container with contents.

Other liquid alloys were tried and some very peculiar results obtained. Copper impregnated with sodium amalgam was found to resist the action of acids and water to a greater extent than the metals themselves. I have no explanation to offer why a copper-sodium plate should resist acid more than the copper itself. The resistance of such a plate to water being greater than the resistance of sodium itself can be explained on the theory of the dilution of the sodium with the copper, thereby protecting the sodium from coming into contact with the water.

Very peculiar results were obtained when impregnating copper with antimony amalgam. It was found that such antimony-impregnated copper greatly resisted the action of dilute nitric acid, the amount of copper that went into solution from the surface contact being not over 10 per cent of the amount that went into solution from the edges. The acid penetrated from the edges toward the center, and the rapidity of the solvent action of the acid from the edges toward the center was visibly noticeable. Strength of acid, temperature and time are factors that materially influence this action. Protecting the edges by dipping into paraffin showed conclusively that the corroding action of nitric acid on copper was greater than on the same amount of copper impregnated with antimony.

Very extensively used for food containers are the so-called enameled cans. They are not real enameled cans, but simply metal plates on which an enamel has been baked. Real enamel should become part of the whole and not simply a superficial protecting coating that can be separated by mechanical means. Experiments were made to impregnate enamels in iron plates, but the high pressure required makes the process too expensive to be commercially developed now. The enamel was forced into the iron plate at a pressure of about 50,000 lb. per square inch. The superficial excess coating of enamel was scraped off the plate and the impregnated plate was subjected to a heat sufficient to melt the enamel. Enamel-impregnated iron plates withstood the action of acids and alkali fully as well as glass.

There were many difficulties found in using enamels and experiments were then made to impregnate iron plates with glass. The results were most satisfactory, and until the process and products are fully protected no details can be given.

The best results were so far obtained when using a very fusible lead-glass. Such impregnated glass-iron plates were found to resist the corroding action of food juices fully as well as the glass itself. The amount of lead that will go into solution has not yet been determined. Whether such a container will meet the requirements of food officials and food chemists, who condemn food products containing 0.00003 per cent of lead and who consider about 0.000001 per cent of sulphur dioxide as unwholesome, remains to be seen.

CHICAGO, ILL.

Fine Grinding.—Apropos the discussion recently engaging so much attention from South African metallurgists in regard to the efficiency of the stamp-tube mill combination, it is interesting to note a recent statement in the *South African Mining Journal*. It has been suggested before that classification and regrinding in stamps would effect the same result better than the stamp-tube mill combination, but the suggestion has not been received favorably. Now, however, according to the authority mentioned, "the New Kleinfontein mill is being prepared for crushing operations, in which the work of the tube mill, now almost an essential of the metallurgical scheme of the Rand, will be done much more efficiently, cheaply and conveniently by stamps alone."

The Equilibrium of the System Consisting of Calcium Carbide, Calcium Cyanamide, Carbon and Nitrogen.

By M. de Kay Thompson and Robert H. Lombard.

(*Concluded from page 620.*)

The experiments described in the first part of this article, published in the November issue, show the method of working, and for the sake of brevity the detailed record of experiments 9 to 17 will be omitted. The final results are given in table 3 on top of the next column on this page.

EXPERIMENT 18.

The disadvantage of the Arsem vacuum furnace for experiments of this kind is that the volume is so great that it takes quite a long time for the solid substance to absorb or evolve enough gas to reach equilibrium pressure. It was therefore decided to try an experiment in the Heraeus furnace that had been used as an auxiliary furnace for calibrating the Wanner pyrometer. Of course, this could be used only at a comparatively low temperature. It was glazed only on the inside, and it was anticipated that it would leak.

The charge was contained in a graphite boat 1.5 in. long. It consisted of 6.14 grams of a mixture of calcium carbide and calcium chloride containing 10 per cent of the latter. The ends of the porcelain tube of the furnace were closed with rubber stoppers; the stoppers and ends of the tube were shellaced. A small glazed porcelain tube for holding the thermo-electric couple extended through the larger tube and the two stoppers. The junction was pushed in until it was opposite the graphite boat in the middle of the furnace.

Aug. 10 at 5 p. m. the furnace was pumped out to a pressure of 1 mm and left till Aug. 11, 16 hours later, when it had leaped to 13.1 mm. The furnace was then heated to about 1070° and pumped out a number of times, after which the gases remaining were swept out with nitrogen. While still hot nitrogen was admitted to 110 mm. Absorption began to take place rapidly and the run was begun.

Time.	Pressure.	Temperature.
2:08 p. m.	110	....
2:10 " "	32	1074
2:13 " "	23.2	1074
2:20 " "	21.5	1074
2:25 " "	21.6	1074
2:32 " "	22.1	1074
2:40 " "	22.8	1077
2:45 " "	23.2	1077

It seems that here the absorption of nitrogen at first overbalanced the leakage. The following day minima of 26.8 mm, 30 mm and 31.5 mm were obtained, starting with a pressure of 145,165, and between 100 mm and 200 mm respectively. The heat value, of course, would be obtained by starting very little above the equilibrium pressure, so that there would not be so much time for leakage to take place.

Although these determinations were somewhat in error from the leakage, this error was evidently not great, as the pressures agreed well with those obtained in the vacuum furnace. The temperature measurements were more reliable than the others, as they were made directly with the thermo-electric junction. The lowest pressure, 21.5 mm, is presumably the most reliable. The result of this run is therefore 21.5 mm at 1074° C.

The results of the above experiments are collected in Table 3 and are plotted in Fig. 5. The ordinates are pressures in centimeters of mercury and the abscissæ are the temperatures in centigrade degrees. The pressures are not reduced to zero degrees, as this correction is much less than the experimental error.

It is remarkable that the relation between the temperature and pressure is linear through such a great range in temperature. It would seem that at a lower temperature the line representing this relation must become concave upward, and it is hoped that in the near future it will be possible to carry out some experiments on this equilibrium below 1000° to test this point.

TABLE 3.

No. of experiment.	Pressure in cm. of Hg.	Temperature centigrade.
4	27.0	1263
6	22.4	1223
7	30.8	1297
8	37.8	1378
10	12.5	1160
10	16.1	1196
10	27.3	1278
11	28.3	1308
12	2.3	1053
12	6.3	1117
13	6.8	1114
15	44.4	1448
16	38.8	1372
17	37.5	1385
18	2.2	1074

In order to compute the heat Q, evolved by the reaction

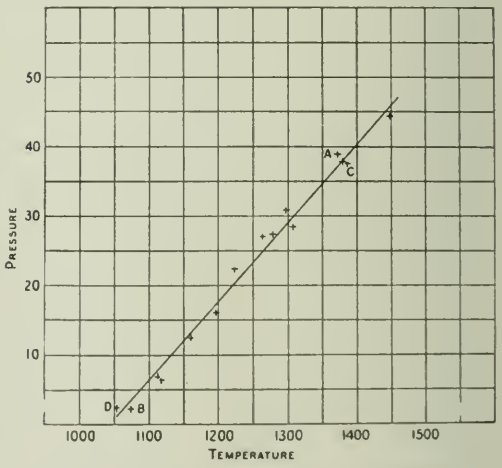


FIG. 5.—CHANGE OF HEAT OF REACTION WITH TEMPERATURE.

$CaC_2 + N_2 \rightleftarrows CaCN_2 + C$, taken from left to right, it is not allowable to use the integrated van't Hoff equation

$$4.57 \log \frac{p_2}{p_1} = Q\left(\frac{1}{T_1} - \frac{1}{T_2}\right).$$

because this equation is derived on the assumption that Q is constant, which is not the case in this reaction. If Q is computed from the differential equation

$$\frac{dp}{dt} = \frac{Q}{T(v - v_1)}$$

in which T is the absolute temperature, v is the volume of the carbide and nitrogen and v_1 is that of calcium cyanamide and carbon on the other side of the equation, and $\frac{dp}{dt}$ is the pressure gradient at the temperature T, Q is found to have the values given in Table 4. The volume v_1 is negligible in comparison with v. Omitting v_1, the equation may be put in the following form:

$$Q = \frac{RT^2}{p}\frac{dp}{dt}$$

The value of R is 1.99 calories and the unit in which p is expressed cancels out

TABLE 4.

Showing the Change in the Heat of the Reaction with the Temperature.

Temperature Centigrade.	Q in Gram Calories.
1100	63,900
1200	27,000
1300	18,800
1400	15,200

The value of $\frac{dp}{dt}$ is, of course, constant for every temperature

The line as drawn in Fig. 5 seems to represent the observations better than any other. The maximum variation in the slope would be represented by the lines *AB* and *CD*. From the value of $\frac{dp}{dt}$ obtained from these lines, the heat was calculated[4] for 1200° and found to be 29,800 and 25,800 calories, respectively. This shows that the value calculated from the best representative line, 27,500, cannot be in error by more than 2000 calories, or 7 per cent.

It seems evident from the fact that the heat effect is greater the lower the temperature that it would be advisable to work at as low a temperature as possible in the fixation of nitrogen by this method, because less heat would be required from external sources. This would be true only if the velocity of the reaction did not decrease with decreasing temperature, for this might more than balance the benefit due to the heat of the reaction.

In view of the enormous increase in the heat of the reaction with decreasing temperature, as computed from these measurements, it is desirable that the heat should be measured in some other way. This might be done, as suggested by Mr. S. Peacock, by combustion experiments in a bomb. This would give indirectly the heat of the reaction at room temperature, which would be expected to be much larger than the values calculated above.

The free energy increase of the reaction

$$CaC_2 + N_2 \rightleftarrows CaCN_2 + C$$

at 1450° C. taken from left to right is

$$\Delta F = -RT \log \frac{76}{46} = -1714 \text{ gram calories.}$$

At 1100 C., $\Delta F = -6700$ calories.

The errors in the temperature measurements are probably the main cause of the deviations of the single determinations from the best representative line. If all the error is assumed due to the temperature measurements the average deviation of a single point from the line is only 12°, and the largest deviation is 19°. These are not greater than the experimental errors in the temperature measurements.

The thermo-electric couple had been cut off several times since the original calibration. It was subsequently calibrated as follows:

TABLE 5.

Date.	Temperature.	Millivolts at date given.	Millivolts at original calibration.
July 2.	Sulphur point.	3.60	3.60
Aug. 13.	Copper point.	10.08	10.19

The original calibration had therefore remained practically constant throughout the experiments.

After each experiment for determining the equilibrium pressure, the furnace walls were covered inside with a white powder. A similar result was obtained in determining the equilibrium of the reaction $CaO + 3C \rightleftarrows CaC_2 + CO$. It was noticeable, however, that there was more powder formed with calcium cyanamide for a given temperature and given duration of heating than with calcium carbide. It therefore seemed probable that calcium cyanamide is volatile, especially after it was found that the white powder gave a yellow precipitate of silver cyanamide on adding silver nitrate to an ammoniacal solution of the powder. It was therefore thought desirable to make an analysis of this material.

Some of the pure white powder was analyzed for nitrogen by the Kjeldahl method, taking about one gram for analysis. The digestion with chemically pure sulphuric acid was continued for five hours near the boiling point; immediately after, potassium permanganate was added. The two determinations gave 12.38 per cent and 11.67 per cent nitrogen, corresponding to 34.3 per cent calcium cyanamide.

To get a check on this value the calcium cyanamide was determined by the method of Perotti.[5] This method consists in allowing the substance to stand over night in water, filtering and washing, adding ammonia, and then an excess of silver nitrate. A yellow precipitate of silver cyanamide is produced which is filtered off and the excess of silver nitrate is titrated with ammonia sulphocyanate. The sample taken was not perfectly white, but from its gray color evidently contained some carbon which had also distilled to the wall of the furnace.

The residue obtained on dissolving the white powder in water was of a dirty gray color. It was evidently carbon and calcium oxide, for (1) it dissolved slightly in a fresh portion of water and gave a distinct precipitate when ammonia and ammonium oxalate were added; (2) on treating some that had been exposed to the air with hydrochloric acid it dissolved partly with effervescence, leaving a black residue behind; and (3) when the hydrochloric acid solution was neutralized with ammonia and ammonium oxalate was added an abundant precipitate was formed. The analysis of the water solution for calcium cyanamide gave 38.59 per cent, 39.52 per cent and 38.84 per cent, corresponding to 13.51 per cent, 13.84 per cent and 13.60 per cent nitrogen.

An analysis of a sample of the same material used in the Kjeldahl determinations gave only 25.96 per cent calcium cyanamide, corresponding to 9.09 per cent nitrogen. Since this did not agree with the determinations by the Kjeldahl method, or with the values obtained by the same method for other samples, analyses were made by the Perotti method of two more samples. The perfectly white material used in the Kjeldahl method could not be checked again as it had all been used up. The results of the analyses of the powder obtained in Experiment 16 were 15.96 per cent and 15.35 per cent calcium cyanamide, corresponding to 5.59 per cent and 5.38 per cent nitrogen. Analyses of the material obtained in Experiment 17 gave 29.10 per cent and 28.95 per cent calcium cyanamide, corresponding to 10.19 per cent and 10.13 per cent nitrogen. Experiments 16 and 17 were made at about the same temperature and still the amount of calcium cyanamide in the powder distilled to the cold part of the furnace was quite different.

Two determinations of calcium in the white powder used in the Kjeldahl determinations were made by first destroying the organic matter with fuming nitric acid and then decomposing the calcium nitrate by ignition in a platinum crucible and weighing as the oxide. The fuming nitric acid had to be added very slowly, as it attacked the white powder with great violence, sometimes causing it to flame. The results were 39.93 per cent and 39.27 per cent calcium, the average of which is 39.60 per cent. From this and the fact that the content of nitrogen was 12.1 per cent the amounts of calcium oxide and of calcium cyanamide in the first sample of white powder analyzed were as follows:

Calcium cyanamide	34.5	per cent.
Calcium oxide	30.5	" "
Unaccounted for	35.0	" "

Time did not permit of making a combustion analysis to see how the carbon checked up. This sample apparently contained no free carbon, for it was of a cream-white color.

Since calcium cyanamide was present in the powder distilled

[4] *Trans. Am. Electrach. Soc.* 15, 197 (1909).

[5] *Chemisches Centralblatt* 2, 1058 (1905).

to the walls of the furnace, when heated with carbon it should produce calcium carbide with the evolution of nitrogen. In order to see whether this actually takes place 0.4 gram of the powder was mixed with 0.9 gram Acheson graphite and heated in the vacuum furnace. The results are the following:

EXPERIMENT 14.

		Kilowatts.	Pressure.
July 11	11:00 a. m.	...	2.8
	11:12 " "	0.33	10.0
	11:20 " "	0.78	...
	11:35 " "	...	19.0
	11:35 " "	1.27	...
	11:45 " "	...	23.8
	11:45 " "	1.88	...
	12:00 m	...	28.8
	12:00 m	2.7	...
	12:20 p. m.	...	67.0
	12:20 " "	5.3	...
	12:35 " "	8.0	...
	12:45 " "	...	183
	2:15 " "	...	195

The temperature was measured at 12:35 p. m.; with the current in the Wanner 6.1 units the temperature measured 1287° C.

On opening the furnace after this run no white deposit was found on the walls, and the material in the crucible seemed to be hardly changed in appearance. It gave no test for calcium carbide with an ammoniacal cuprous chloride solution, nor for cyanamide with silver nitrate. It is to be noticed, however, that a great deal more gas was evolved from this mixture on heating than on heating a sample of carbide, from which occluded gases always escape. For example, in Experiment 15, on heating carbide to 1450° C., these gases produced a pressure of 127 mm, which is considerably less than the 195 mm produced from the white powder and carbon at 1287°.

It therefore seems certain that the large amount of gas evolved in Experiment 14 was partially due to a chemical reaction and not to occluded gases alone. A possible explanation of the absence of carbide in the crucible is that some oxygen got into the furnace. This would be changed to carbon monoxide, which would decompose any carbide formed. The result of the experiment is, however, not conclusive.

Since calcium oxide was found in the white powder, an experiment was carried out to see to what extent lime would distil when heated alone in a vacuum. A graphite crucible was filled with pieces of Mercks lime about the size of a pea, and was heated in a vacuum for three hours at 1531°. On opening the furnace it was found covered on the inside with a thin grayish white powder. On collecting this it filled about ¼ in. of an ordinary test tube, and was placed in a desiccator several days before testing. It was then dissolved in hydrochloric acid and tested for calcium with ammonia and ammonium oxalate, a large amount of precipitate of calcium oxalate was formed. This experiment, therefore, shows that lime distils at about 1500° in a vacuum. This doubtless is the cause of the thin layers always found in the experiments on the equilibrium of the reaction $CaO + 3C \rightleftarrows CaC + CO$. It was not known at the time of the determination of this equilibrium that lime is distilled in the neighborhood of 1500°.*

3. Summary.

The equilibrium pressure of nitrogen in the reaction

$$CaC_2 + N_2 = CaCN_2 + C$$

was determined from both sides for a number of temperatures between 1050° C. and 1450° C. The results are plotted in Fig. 5. It was found that the heat of the reaction varies enormously with the temperature. The values calculated from the plot in Fig. 5 are given in Table 4.

The free energy increase of the reaction taken from left to right is —1714 gram calories at 1450° C. and —6700 calories at 1100° C.

Calcium cyanamide distills to the cold parts of the furnace at temperatures as low as 1050° C. Pure lime distills appreciably in the neighborhood of 1500° C.

Electrochemical Laboratory, Rogers Laboratory of Physics.
Massachusetts Institute of Technology,
Boston, Mass.

*This Journal, 8, 1, 27 (1910).

A Method for Increasing the Calorific Power of Washington Lignites.*

By H. K. Benson

Many of the coal deposits of Western Washington are found in localities where the rocks have undergone only a moderate amount of folding and without any shearing movement. This is especially true of the Puget Sound basin proper and in these places the coal is lignitic in character.

The most important lignite fields are the Renton-Cedar Mountain and the New Castle-Issaquah fields in King County and these are the only ones that are being operated on a large scale at the present time. Extensive lignite deposits also occur in Thurston, western Lewis and Cowlitz Counties, but are not mined to any extent on account of the low heating value of the coal.

In order to ascertain the effect of drying and partial distillation on the heating value of some of the low grade lignites of Washington, the following experiments were undertaken in the laboratory of industrial chemistry, at the University of Washington, under the direction of the writer.

Four samples of run of mine coal were obtained for the experiments—one from Tenino in Thurston County; one from the Seattle Electric Company's mine at Renton in King County, taken from veins Nos. 2 and 3; one from the Denny-Renton Clay & Coal Company's mine at Renton, representing vein No. 1, and one from Centralia in Lewis County, obtained from the Chlopeck Fish Company, of Seattle, who use it in their gas producer. The above coals are what are commonly known as black lignites in Washington. The United States Geological Survey classifies them as sub-bituminous coals.

The coal was crushed to pea size and carefully sampled. Lots varying from 16 to 30 pounds were weighed out and placed in a large, horizontal, cylindrical iron retort, which was then sealed and heated with Bunsen burners placed underneath it until the bottom became a low red heat. The heat was applied until the gas ceased to come off in appreciable quantities. This took from 12 to 18 hours on account of the poor facilities for heating the retort.

The gas was passed through two large bottles, then through a condenser, then through water, and finally through a meter. It was sampled and analyzed. Most of the by-products condensed in the first two bottles. They consisted mainly of the water present in the coal, with some tar and a little ammonia.

Results of Experiment with Tenio Lignite.

Weight of coal placed in retort	= 30 pounds.
Weight of resulting produce	= 21 pounds 5¾ ounces.
Weight of by-products	= 6 pounds 5½ ounces.
Volume of gas	= 28.3 cubic feet.

Proximate Analysis of Tenino Lignite Before and After Drying.

	Before drying.	After drying.
Moisture	13.4 %	1.52%
Volatile matter	31.7 %	18.08%
Fixed carbon	34.17%	54.3 %
Ash	20.73%	26.1 %
Sulphur	1.34%	1.84%
Heating value per pound of coal	8525 B.t.u.	10,320 B.t.u.

The by-products contained 303 grams of ammonia and 147 grams of tar separated out. The tar had a specific gravity of .910.

*Abstract of paper submitted for the Kerl prize in Industrial Chemistry by Charles R. Fettke.

Analysis of Gas Given Off by Tenino Lignite.

Carbon dioxide	31.6%
Illuminants	2.3%
Oxygen	2.6%
Carbon monoxide	6.0%
Methane	37.1%
Hydrogen	7.6%
Nitrogen	12.8%
Heating value per cubic foot	488 b.t.u.

There was a decrease in weight of 28.8% in drying the coal and its heating value was increased 21.1%.

Results of Experiment with Seattle Electric Company's Lignite, Representing Veins 2 and 3, at Renton.

Weight of coal placed in retort = 16 pounds.
Weight of resulting product = 10 pounds 12¼ ounces.
Weight of by-product = 3 pounds 9¼ ounces.
Volume of gas = 21 cubic feet.

Proximate Analysis Before and After Drying.

	Before drying.	After drying.
Moisture	12.2%	1.4 %
Volatile matter	34.0%	25.53%
Fixed carbon	42.9%	56.97%
Ash	10.9%	16.1 %
Sulphur	.73%	.94%
Heating value per pound of coal	11,798 b.t.u.	11,735 b.t.u.

The by-product contained 3.49 grams of ammonia and 372.5 grams of tar separated out. The tar had a specific gravity of 9455.

Analysis of Gas.

Carbon dioxide	19.4%
Illuminants	2.4%
Oxygen	2.0%
Carbon monoxide	6.4%
Methane	26.4%
Hydrogen	23.9%
Nitrogen	18.5%
Heating value per cubic foot	433 b.t.u.

There was a decrease in weight of 33% in drying the coal and no increase in heating value. The heating value remained practically the same.

Results of Experiment with Denny-Renton Clay & Coal Company's Lignite, Representing Vein No. 1, at Renton.

Weight of coal placed in retort = 20 pounds.
Weight of resulting product = 13 pounds 15¼ ounces.
Weight of by-products = 4 pounds 6½ ounces.
Volume of gas = 21.3 cubic feet.

Proximate Analysis Before and After Drying.

	Before drying.	After drying.
Moisture	11.3 %	1.62%
Volatile matter	27.7 %	15.77%
Fixed carbon	37.6 %	51.55%
Ash	23.4 %	31.06%
Sulphur	.59%	
Heating value per pound of coal	8930 b.t.u.	9819 b.t.u.

The by-product contained 6.05 grams of ammonia and 275 grams of tar separated out. The tar had a specific gravity of .935.

Analysis of Gas.

Carbon dioxide	20.0%
Illuminants	2.1%
Oxygen	4.9%
Carbon monoxide	6.8%
Methane	24.7%
Hydrogen	22.9%
Nitrogen	18.6%
Heating value per cubic foot	407 b.t.u.

There was a decrease in weight of 30.20% in drying the coal and an increase of 10% in the heating value.

Results of Experiment with Centralia Lignite.

Weight of coal placed in retort = 20 pounds.
Weight of resulting product = 12 pounds 3¾ ounces.
Weight of by-products = 5 pounds 10¾ ounces.
Volume of gas = 24.5 cubic feet.

Proximate Analysis Before and After Drying.

	Before Drying.	After Drying.
Moisture	18.4%	1.4%
Volatile matter	29.3%	16.71%
Fixed carbon	36.7%	56.87%
Ash	15.6%	25.02%
Sulphur	1.76%	
Heating value per pound of coal	9086 b.t.u.	10,240 b.t.u.

The by-products contained 3.68 grams of ammonia and 312 grams of tar separated out. The tar has a specific gravity of 0.940.

Analysis of Gas.

Carbon dioxide	26.6%
Illuminants	1.6%
Oxygen	3.2%
Carbon monoxide	5.6%
Methane	29.2%
Hydrogen	19.1%
Nitrogen	14.2%
Heating value per cubic foot	428 b.t.u.

There was a decrease in weight of 28.8 per cent in drying the coal and an increase of 12.7 per cent in the heating value.

Results Calculated for 100 lb. of Coal.

	Tenino Lignite.	Seattle Electric's Lignite.	Denny-Renton Clay & Coal Co.'s Lignite.	Centralia Lignite.
Weight of coal taken in pounds	100	100	100	100
Weight of dried product in pounds	71.13	67.29	69.75	61.15
Increase in calorific power	21.1%	0.00%	10.0%	12.70%
Weight of ammonia in pounds	0.0223	0.0481	0.0667	0.0406
Weight of tar in pounds	1.08	5.13	3.03	3.44
Volume of gas in cubic feet	94.2	131.3	106.5	122.5

The foregoing results show a considerable increase in calorific power of low-grade lignites through partial destructive distillation with the exception of the Seattle Electric Company's lignite, in which case there was no change in heating value. However, this coal already had a high heating value for a lignite before it was dried. Partial distillation in this case resulted in increasing the ratio of volatile matter from 5 to 9, while in the other cases this ratio was increased from 5 to 16. The best results were obtained with the lignites from Tenino and Centralia, which run very high in moisture and are low in heating value.

Of the by-products the tar is probably the only one that would be of any economic value. From 3 per cent to 5 per cent of the weight of the coal taken was received as tar. The ammonia is produced in such small quantities that it would probably not pay to save it. From 1 cu. ft. to 1¾ cu. ft. of gas is also obtained per pound of coal used. This gas would be worth nearly as much as ordinary coal-gas and this would make the gas recovered from one ton of coal worth about $2. It could be used as fuel to dry more coal, or it could be used in a gas engine, or it could be sold.

Summary.

1. Analyses of lignites show high moisture and the volatile matter nearly equal to fixed carbon.

2. After partial distillation the moisture content is greatly reduced and the relative amount of fixed carbon increased.

3. A heating value of lignites is increased from 0.00 per cent to 21.1 per cent.

4. By-products recovered per ton are as follows:

Coal gas (b.t.u. = 430-488 per cu. ft.)..2000-2600 cu. ft.
Ammonia (NH_3) .....0.8-1.2 lb.
Coal tar .....100-20 lb.

5. The extended range of utility for these lignites and the commercial value of the by-product suggest this method as an improvement over the means heretofore used for utilizing low-grade lignites.

Laboratory of Industrial Chemistry
University of Washington,
Seattle, Wash.

The Smelting of Lead-Antimony Ores.

By A. W. Hahn.

As far as my knowledge goes the small smeltery located in the State of Queretáro, Mexico, and two days ride east of the city of the same name, is the only one smelting lead-antimony ores and making a saving at one operation of both the lead and antimony.

The property is owned by O. y T. Braniff, of Mexico City, and the mines are among those first worked by the early Spaniards in Mexico. The surface ores were clean lead carbonates, rich in both lead and silver, and easily treated. At depth the character of the ores changed, sulphides appeared and the lead was replaced to a greater or less extent by antimony. It is these ores which are being treated to-day, and in 1907, when I was connected with the company, at a profit.

The smeltery is located on a hillside, at the mouth of the main working tunnel of the mine called Socavón. The ore is trammed from the hoist station, by contract, to the ore-picking level, where it is dumped over a 1-in. grizzly. Here the ore is carefully picked and sorted into "gabarro," 4-in. size; "granzon," 1-in. to 4-in. size; "arrenillas," fines and waste. The 4 in. and 1 in to 4 in. go to separate bins; the fines go to the briquetting yard, and waste to the dump. Sulphides are separated into coarse and fines; the coarse going to the roast heaps direct, and the fines are first briquetted and then roasted. Ore is also received from a mine two hours' packing distance from the smelter; this ore has already been sorted, and is of two classes, raw carbonates and roasted galena, both classes containing very little antimony.

On the sorting level are the briquetting and roasting yards, briquetting being done by hand and roasting in heaps, both by contract. Below this level come the ore bins, and below these the ore beds, iron flux, charcoal and coke bins. This is the charge floor level. Limestone of a superior grade is quarried at one side of the plant, broken to size by women, and delivered to the charge floor through an inclined 10-in. pipe

The smelting equipment consists of two 36-in. x 60-in. water-jacketed furnaces, of the usual type, one number four rotary blower, belt-connected to motor drive. The power for the plant was generated at a small hydroelectric installation in the valley, about seven miles distant from the plant The native furnace men were very proficient in handling these ores, all of them having worked at the plant for years

TABLE "A."

Class of material	SiO_2 %	FeO %	CaO %	Al_2O_3 %	S %	Sb %	ZnO %	Pb %	Ag oz.
Socavón Ore, Crse.	46.1	14.0	6.5	6.5	1.8	7.8	9.8	2.0	32
" " Fns.	52.8	14.1	6.8	7.0	0.0	4.6	1.0	1.4	21
Sto. Entierro Ore	24.0	10.0	3.8	3.0	2.4	0.4	3.3	13.5	42
Calcines "A"	42.2	11.7	3.5	6.0	5.0	2.0	6.0	7.5	26
" "B"	20.8	28.0	3.7	3.5	8.0	0.3	9.5	12.8	31
Calcined Matte		64.0	2.8		5.1		2.0	8.5	50

As will be seen from the table the material available for smelting is not of the best. For the small furnaces the percentages of zinc, alumina and antimony are high, and the lead content low. The average analysis of the bed mixture was as follows: SiO_2, 30.5 per cent, FeO, 24.5 per cent, CaO, 7.0 per cent, Al_2O_3, 4.8 per cent, S, 3.7 per cent, ZnO, 4.6 per cent; Pb, 8.3 per cent; Sb, 3.6 per cent and Ag, 40 oz. The iron flux was very variable, from 10 per cent to 18 per cent SiO_2, 38 per cent to 48 per cent FeO, 0 per cent to 4 per cent Pb and 2 oz. to 10 oz. silver. The limestone was excellent, never falling below 52 per cent CaO, and carried about 2 oz. of silver. Welsh coke was used, but owing to the long transportation on pack animals the fines loss was excessive. The charcoal was exceptionally good, burned well and with practically no fines.

The typical daily furnace record, for 12 hours, slags produced, together with bullion and matte, and the extraction, are given in Table "B." Fourteen ounces air pressure at furnace was used, the tuyeres were never bright and the lead in lead well always a dark red. The furnaces had to be very carefully watched, as the charge had a tendency to stick and bridge over above the tuyères. Accretions formed very quickly in the shaft and usually a campaign lasted at the most 32 days.

TABLE "B."

Furnace Mix.	Tons.	Production.	SiO_2 %	FeO %	CaO %	Al_2O_3 %	ZnO %	Sb %	Pb %	Ag oz.
Bedded Ore	16.6	Slag	34	32	18	9	3	1	1	3
Iron Flux	6.0	Matte							13	53
Limestone	3.8	Bullion						26	70	240
Slag	6.2									
Calc. Matte	1.0	Weight of Matte Produced, 1.634 tons.								
Charcoal	4.2	" " Bullion " 1.779 "								
Coke	1.3	Percentage Extraction, Lead, 83.7 per cent.								
Poor Lead	0.1	" " Silver, 82.3 per cent.								

Taking into consideration the refractory nature of the ore, the extreme volatility of antimony, the fuel used and the size of the furnaces, the slags are exceptionally low in both silver and lead. With suitable appliances for the recovery of dust and fume, and with an adequate forehearth, an excellent extraction of all values could have been obtained.

As in all isolated sections, the question of freights was a serious one. Coke laid down at the smeltery cost $21.75 per ton and charcoal $7. Ore, iron flux and limestone were laid down at a reasonable figure. Wherever possible work was done by contract, as mining, roasting, briquetting, tramming and bedding materials. The wages paid, both in the mine and smeltery, were very low, ranging from 12½ cents for boys and women per day to $1 for furnace foremen.

The cost of smelting per ton of ore was in the neighborhood of $12; this, however, did not include cost of marketing product, which was very expensive, as freight to the railroad amounted to $7.25 per ton.

The success of this plant during 1906 and 1907 was primarily due to the high prices of silver and lead (the selling contract called for antimony being paid for at the same price as lead). Aside from this fact, the extreme care taken when bedding the ores to have no fines and as little sulphides present as possible; to mix the ore mixture thoroughly before charging, and the screening of both charcoal and coke to insure the absence of breeze; finally, at no time would the furnaces be crowded. All of the foregoing facts contributed to the successful operation of the furnaces.

Metallurgically, the plant was interesting in many ways. The ore is an unusual combination; the bullion produced shows that antimony can be included as a collector of precious metals and presents but few difficulties from a smelting standpoint. It is rare that such a clean matte is produced, and contrary to expectations, this matte, except on very rare occasions, carried more than traces of antimony. It must also be borne in mind that smelting was carried on, comparatively speaking, with a cold running furnace, the appearance of which, at its best, would have made any lead metallurgist, accustomed to bright tuyères and red-hot lead, very uneasy

Campo Morado,
Gro., Mexico.

Zinc box precipitate may be refined with a solution of sulphurous acid as well as by sulphuric. The action is a little slower, but it has the advantage of being inexpensive, and is adapted to isolated places.

Recent Metallurgical and Chemical Patents

Iron and Steel.

Sintering Pulverous Iron Ore.—An interesting method of sintering pulverous iron ore, flue dust, magnetic concentrates, etc., and bringing them into suitable form for smelting in blast furnaces and for use in open-hearth furnaces in a single continuous operation has been patented by Mr. SIDNEY CORNELL, of Duquesne, Pa. When pulverous iron ore is heated to a temperature slightly above 2500° Fahr. part of the material is melted and the fused portion acts as a binder. This is probably due to the formation of ferrous silicates. For subsequent use of the clinker in the blast furnace or open hearth it is advantageous to return the ferrous compounds back to the ferric condition by subjecting the clinker while heated to oxidation. The process is carried out in a long heating chamber, the cross-section of which is shown in Fig. 1. From a hopper the material is charged into the pit 3 and carried along the whole chamber by means of a screw conveyor 12. The material is continuously forced into the pit until the pit is filled from end to end and the material is forced above the upper edges of the pit and passes out and over upon the upper faces of the curved walls 2, 2^1. The gas passing through the ducts 4, 4 and burners 6, 6, being ignited at the inner ends of the burners, heats the inside of the arched wall 1 to a high degree, and the mass of material being treated is heated directly and by the heat reflected from the arched wall. This induces a sintering of the outer portion of the material, which becomes incrusted with clinker. As more of the material is forced by the screw into and up through and out of the pit 3 it rises with a wave-like motion and breaks the crust or clinker formed upon the upper face of the pile of material, turning or folding over the broken pieces of the crust and exposing to the direct action of the heat the raw or unsintered sides, so that it becomes sintered on all sides. As the feeding process continues the broken clinker piles up upon the curved walls 2, 2^1, beyond the angle of repose (indicated by the dotted line 16), and then rolls down upon the tables 10, 11. When it reaches these tables the material consists of conglomerated iron oxides, but a large portion of which is in ferrous condition. On these tables the broken clinker remains sufficiently long for the air passing from below, through and over it into the chamber 5, to oxidize the ferrous oxide of the clinker into ferric condition, whereby the material becomes properly prepared and workable in the smelting furnace. When the clinker has laid long enough upon the tables to become sufficiently cool it is dumped into the car 17 below. The air passing over the clinker on the inclined tables cools the clinker, and is at the same time highly heated thereby, and while aiding the combustion of the gas passing from the burners also aids in the production of the higher temperature within the furnace. Any gases produced escape upward into a smokebox 8 and leave through the smokestack 9. In a single continuous operation the pulverous material is thus brought into masses of size and weight available for smelting in blast furnaces and for use in open-hearth furnaces, the ferrous compounds being simultaneously changed to the ferric condition. (962,006, June 21, 1910.)

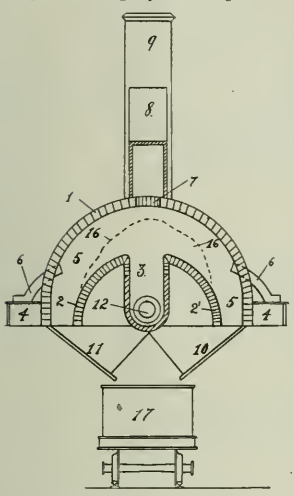


FIG. 1.—FURNACE FOR SINTERING PULVEROUS IRON ORE.

Steel Directly from Iron Ore.—Mr. FREDERICK T. SNYDER, of Oak Park, Ill., proposes the direct production of steel from iron ore in a single operation in an electric furnace by observing the following conditions: The minimum of carbon is used in the furnace charge required to secure reduction of the metal from the ore and produce steel. The steel collects in a crucible of basic refractory, such as dolomite, which is intended to take up small amounts of phosphorus and sulphur from the steel, but which cannot introduce carbon into the steel. The slag is confined by a fluid-cooled wall in contact with it so that the outlying portions of the slag bath are frozen, forming a lining of solid slag. The construction of the furnace is shown in Fig. 2, where 3 is the dolomite crucible, surmounted by the water jacket 5, which forms the furnace chamber; 6 are tap holes for the slag 9, while 7 is the tap hole for the steel 8, which is not allowed to accumulate in the crucible to such an extent that it would come into contact with the waterjackets. Carbon elecrodes 10 (of which only one is shown) are suspended from the top into the molten slag, but are kept high enough so as not to come into contact with the molten steel in the crucible. The charge to be smelted is fed through the open top of the furnace onto the slag bath. The carbon electrodes are protected by a coating of lime, which prevents waste due to the absorption of oxygen from the oxide ore. (974,608, Nov. 1, 1910.)

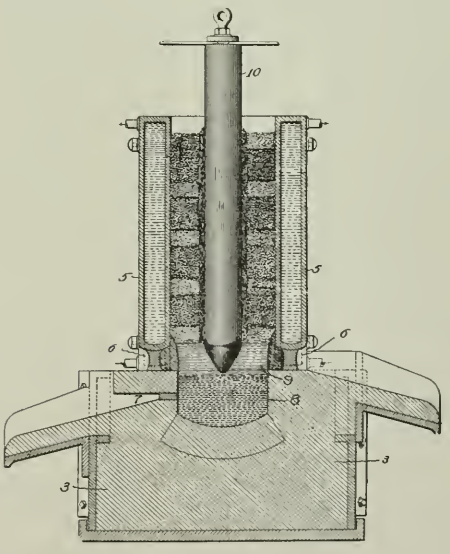


FIG. 2.—ELECTRIC STEEL FURNACE.

Steel from Ore in Electric Furnace.—An electric furnace of Mr. JOHN W. EVANS, of Belleville, Ont., Canada, is a combined arc and resistance furnace. Two sets of electrodes are provided; one set of opposite horizontal electrodes, the other of vertical electrodes suspended from the top. When operation is started the latter two electrodes are raised and not in

operation. An arc is started between the two horizontal electrodes and the upper portion of the charge is melted, producing a slag. The two horizontal electrodes are then withdrawn and the two vertical electrodes are lowered into the slag. The furnace now acts as a resistance furnace. If the material on one side cools off, an arc is produced between the vertical electrode and the horizontal electrode on this side. (960,862, June 7, 1910.)

To produce steel directly from iron ore in this furnace Mr. Evans mixes 10 per cent by weight of charcoal or tar with 90 per cent of iron ore and fluxing material and proceeds as described above. For the production of malleable iron a smaller quantity of carbon is required. The furnace is said to be "particularly adapted to treating ore containing titanium and vanadium, and by suitably regulating the quantity of limestone, sulphur and titanium can be slagged off and fine steel can be obtained, or if a hard tool-steel is desired when treating titaniferous iron ores less limestone may be used, and by regulating the quantity thereof any desired proportion of titanium can be retained in the steel." (960,863, June 7, 1910.)

Pure Metals from Impure Oxides and Sulphides.—Mr. J. W. Beckmann, of Niagara Falls, Ont., Canada, has devised an electrolytic furnace method for the production of pure metals, of which are mentioned iron, aluminium, chromium, titanium, tungsten, molybdenum, manganese and silicon, in general any metal whose oxide will form a salt with calcium. As applied to iron the process is as follows: Iron oxide is mixed with calcium oxide and heated in an electric furnace to produce a calcium salt of iron:

$$CaO + Fe_2O_3 = CaO\,Fe_2O_3.$$

Dibasic, tribasic or n — basic salts may also be produced. This salt is now used as a solvent for iron oxide, which is electrolyzed and yields iron according to the equation, $Fe_2O_3 = 2\,Fe + 3\,O$. Either direct current or alternating current may be used to obtain the bath in a state of fusion, while direct current is, of course, required for electrolysis. The fused $CaOFe_2O_3$ seems to act chiefly as a solvent for the iron oxide. Instead of the oxide, sulphide may be used (973,336, Oct. 18, 1910.)

Pyrites Containing Zinc and Sulphur.—Concerning the treatment of waste pyrites containing zinc and sulphur remaining from the manufacture of sulphuric acid so as to recover zinc and ferrosilicon, see the patent of A. Wiens under Alloys.

Induction Furnace for Steel Refining.—See the patents of Ferranti and Gin under Electric Furnaces on pages 691 and 692.

Copper.

Recovery of Copper from Wet Process Liquors.—An improved method of obtaining copper from the liquors of wet processes treating roasted or unroasted ores, pyrites cinder, residues, ashes or scrap, or from mine water, has been patented by Messrs. J. H. Tilwaites and S. J. Ralph, of Peterborough, England. The object of the invention is to obtain the copper practically free from iron and other metals contained in the liquor. Sulphite of magnesium is the means used by the inventors, although they find that the sulphites of calcium, barium and strontium give the same results, and with greater latitude in the operation. In the use of magnesium sulphite it must be restricted in quantity to a deficiency of the proportion necessary to precipitate the whole of the copper. It is preferred to heat the liquor with the sulphite and afterwards cool it, as in many cases the quantity of copper precipitated thus is greater per unit weight of sulphite. The precipitate can be treated in any manner desired, but as it is practically free from iron the authors suggest its use for the manufacture of copper sulphate, as this process would be cheaper and more rapid than the dissolution of copper in sulphuric acid. Cobalt, nickel and other metals contained in the liquor may be obtained later by precipitation as sulphides. (974,584, Oct. 11, 1910.)

A New Form of Copper Converter designed to avoid the usual destruction of the lining during the reduction of the matte to metal has been patented by Messrs. W. J. Murphy and A. E. Culley, of Butte, Mont. As shown in the upper diagram of Fig. 3, the converter is of the usual form except that it has the throat 19 at one end of the cylinder instead of in the center and is provided with a tap hole 20 and a spout so that the charge may be tapped. It is further provided with hoppers 22 and passages 21, so that fluxing material can be introduced and automatically spread on the bottom and sides

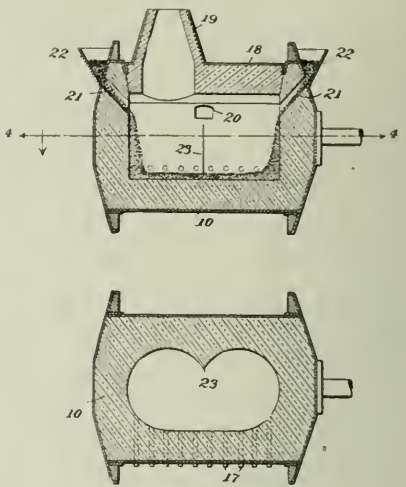


FIG. 3.—COPPER CONVERTER.

of the converter as shown. In the lower diagram, which is a sectional view on the line 4-4 of Fig. 1, is shown the form of the interior lining of the shell, having a rib 23 and the walls preferably rounded. This provides for the swirl of the charge in two directions as it is agitated by the blast from the tuyères 17, causing it to become thoroughly mixed with the charge of fluxing material. The converter could be used with either acid or basic lining and the nature of the flux suited to the matte. The inventors claim that such a converter may be used for a large number of charges without the necessity of relining. (972,564, Oct. 11, 1910.)

Handling Matte.—The usual manner of handling matte when it is not immediately converted to copper is to allow it to cool in molds, after which it is broken in pieces suitable for further smelting or transportation. In order to expedite this process, William D. Kilbourn, of Murray, Utah, has invented a process whereby the matte button is allowed to cool partly in a pan into which it is drawn from the furnace, then broken at a point on its edge and "bled" of its inner molten contents. This leaves a solidified shell which is easily broken by means of a stream of water directed upon it, and handling is thus reduced to a minimum. The smaller buttons formed by the bleeding of the larger are also easily broken by water as described. For the easier handling of the buttons loose anchor pins are placed in the pans when the molten matte is poured in them, so that the entire button can be picked up by the pin when the matte has solidified. (974,074, Oct. 25, 1910.)

Zinc.

Electric Zinc Smelting.—Mr. Woolsey McA. Johnson's work in electric zinc smelting has been known for a series of years. A recent patent granted to him forms apparently an important link in his inventions. In present practice of smelting zinc ores the ore is usually mixed with an excess of carbon above the amount necessary for reducton. There are two rea-

sons for this. Firstly, such a highly reductive charge insures the substantial reduction of carbon dioxide, aqueous vapor and other reactive products which would be capable of oxidizing zinc. Secondly, carbon is insured to be present in this way in the residue, whereby the latter is rendered pulverulent and infusible, any slag fusible at the temperature of reduction being absorbed or retained by the carbon; this is necessary with externally fired furnaces, because a fluid slag or matte in contact with externally fired fireclay retorts would quickly destroy

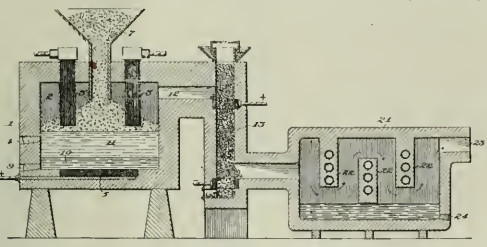


FIG. 4.—ELECTRIC ZINC FURNACE AND CONDENSER.

them. Mr. Johnson's method of operation is different in principle. He uses only such proportion of carbon as is necessary for reduction, producing thereby an easily fusible slag which may be tapped off, while the reduced lead, copper or iron accumulate beneath the slag and are withdrawn. He adds fresh portions of the charge continuously or at intervals, thus securing the advantages of continuous operation. Such operation logically requires the use of an electric furnace, because the production of fluid slags would be destructive to externally fired retorts, and any method other than retorting would require the introduction of such quantities of air in the form of a blast that it would be impossible to secure efficient condensation. The mineral components of the charge which enter into the slag should preferably be so proportioned as to yield a slag fusible at 1100° C. to 1175° C., such slag being superheated at the working temperature of the furnace, 1250°-1300° C., and therefore readily tapped. The gases derived from a charge of the character described are necessarily oxidizing in character, and this would, of course, interfere with condensation. For this reason the vapors are passed through a body of porous material which is artificially heated. This is best done by passing an electric current through the column of coke through which the vapors are passed whereby a more perfect reduction of the volatile products is secured than can be obtained by operating at a temperature of 1020° C. to 1150° C. upon a charge containing a large excess of carbon. The combination of apparatus used in Mr. Johnson's process is shown in Fig. 4, where 1 is an electric furnace with two upper carbon electrodes 3 of equal polarity and a carbon plate 5 in the bottom of opposite polarity. The charge is introduced through the hopper 7 into the furnace. 4 is the slag tap and 9 a tap for the removal of non-volatile metals (lead, copper, etc.) or matte. The zinc vapor passes through 12 over to the electrically heated column 13 of coke or coal, where the vapors are freed from carbon dioxide, aqueous vapor, etc. The purified zinc vapor then passes to the condenser 21, which is provided with water-cooled partitions 22 so as to direct the vapors in a tortuous path. The zinc is condensed in the hearth and tapped at 24. The non-condensing gases pass out at 23. As to the quintessence of his invention Mr. Johnson believes himself to be "the first to recognize that a continuous operation may be combined with a high degree of working efficiency by electrically smelting a charge adapted to yield a fusible slag and passing the vapors therefrom through a body of carbon at a reducing temperature and preferably at the temperature of maximum reductivity; that is to say, at a temperature at which carbon vapors are present." (964,268, July 12, 1910.)

Electric Zinc Smelting.—While well known by his former work in other metallurgical fields, Mr. HIRAM W. HIXON is a newcomer in electric zinc smelting. He employs a shaft furnace with a carbon bottom plate as one electrode and another carbon plate near the top as the other. The furnace is filled with zinc ore and carbon up to a point above the upper electrode. The charge acts as resistor. The top of the furnace is provided with a double hopper to prevent the zinc vapors from getting out and the air from coming in; an upper door may be opened for feeding in new charge, while a lower door is shut; the upper door is then closed and when the charge is sufficiently preheated the lower door is opened and the charge is fed into the furnace shaft. The zinc vapors are drawn off near the bottom of the shaft and passed into the condenser, while in the side of the condenser adjacent to the top of the tap hole a number of arcs are provided (probably in order to prevent clogging of the tap hole by blue powder). After the furnace is once started there should be no let-up in the vapor density, as the furnace is fed continuously at the top and the hot zinc vapors are forced to discharge at the bottom by keeping the furnace tightly closed. It is thus "possible to maintain the density of the zinc vapor at an approximately fixed point. This prevents the formation of the blue powder and results in the continuous production of the metallic zinc." (957,337, May 10, 1910.)

A modification of this process is described in another patent of Mr. Hixon. The chief new features are an exhaust pipe at the top of the furnace for pumping out the diluting gases, carbon dioxide and moisture, and of a tube inside of the furnace for drawing off the zinc vapor. This tube is made of a refractory material (magnesite or carborundum). It is placed in the center of the furnace and reaches from the bottom upward to a point below the upper electrode. It is closed at the top, but in its side walls inclined passages are provided through which the zinc vapors are intended to pass from the inside of the furnace into the tube and then to the condenser. (No zinc vapors can escape through the exit tube for carbon dioxide, etc., at the top, because at this point the temperature is too low for zinc to exist as vapor.) Otherwise the construction of the furnace is essentially the same as in the preceding patent. (957,058, May 3, 1910.)

Blast Furnace for Zinc-Iron Ores.—A patent of Mr. FREDERICK W. GORDON, of Philadelphia, is of considerable interest in view of Mr. Gordon's prominent connection with the large-scale experiments with the Lungwitz zinc process. Fig. 5 shows the blast furnace intended for the smelting of zinc-iron ores. 1 is the hearth, 2 the lower hottest zone, 3 the hopper, 4 the bell, 5 the downcomer, 6 the hot-blast tuyères and 7 the tap hole for the iron. As far as the reduction of iron from iron ore is concerned the furnace acts exactly like an ordinary blast furnace. Now, if the ore contains zinc, let us assume that line 8 indicates the region in the furnace column in which the temperature is high enough to vaporize reduced zinc, while line 9 may represent the colder region in which the temperature is low enough to condense the zinc. Then the zinc would vaporize at line 8, rise up and condense at 9, descend again to 8, vaporize and rise again and thus dance up and down between 8 and 9. This would be the precedure in an ordinary blast furnace. But Mr. Gordon provides a cooler 10 around this intercepting zone 20 between the lines 8 and 9 of the furnace. The details of construction are clearly shown in the diagrams. If the ports 11 and 15 are open, communication is established between the zone 20 in the furnace and the outer cooler 10. Gas with zinc vapor passes from the inside of the furnace through the ports 11 into the outside cooler, where it is forced to rise through 12, descend through 13 and rise again through 14. Condensed liquid zinc is withdrawn through the tap hole 17 (peep holes being provided at 18 and 19). The non-condensed vapors pass through the ports 15 back into the furnace. The vapors "enter the cooler at a temperature approximating that of vaporization under the pressure within the furnace, assumed at 17 lb. absolute or 2.3 lb. gage per square inch, and

leave the cooler at a temperature about that of the solidification of zinc." In continuous operation of the furnace the withdrawal of molten zinc from the cooler should correspond with practical precision with the quantity of zinc contained in the

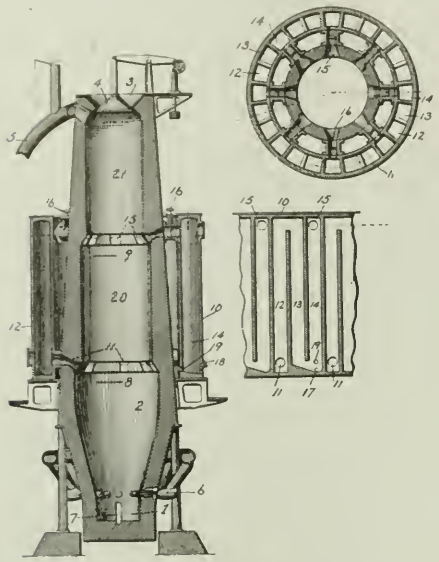


FIG. 5.—BLAST FURNACES FOR ZINC-IRON ORE.

new additions of ore made to the charge at regular intervals. The upper zone 21 of the furnace acts simply as a preliminary heating chamber to utilize the heat of the waste gases. (974,257, Nov. 1, 1910.)

Direct Treatment of Blende.—A new improvement in the treatment of zinc sulphide is suggested by ADOLPHE DESGRAZ, of Hanover, Germany, in a patent recently granted to him and assigned to Imbert Process Company, of New York. The author recites the former methods of reducing zinc ores, comprising roasting and reduction with carbon, or direct reduction of the sulphides by fluidifying them and adding a precipitating substance, such as iron, to precipitate the zinc. He states that various vehicles have been suggested for fluidifying the blende, such as a mixture of lime, oxide of iron and manganese; peroxide of iron and monosulphide of iron; or, peroxide of iron alone. It is pointed out, however, that all of these substances have a value in themselves and would cost something for transportation, etc. The inventor shows the need of a fluidifying material that is plentiful, cheap and otherwise suitable. Such a substance he finds in ordinary slag from lead furnaces. Lead slag, he states, is an excellent dissolvent for blende, i. e., allows of its fluidification at low temperatures, is cheap, contains some zinc, and is generally suited to the purpose indicated. He suggests the following treatment: Melting the slag and blende in a suitable closed furnace, from which the mixture can be tapped to a precipitating vessel where cast iron is used to precipitate the zinc from the mass. (975,217, Nov. 8, 1910.)

Improved Chloridizing Process.—For the treatment of mixed oxidized and sulphide ores of zinc, copper, etc., CHARLES E. BAKER, of Chicago, Ill., finds that a mixture of hydrochloric acid gas and chlorine gas gives better results than either reagent separately. He prefers that the chlorine should be proportioned to the quantity of sulphides in the ore, while the hydrochloric acid gas is used in proportion to the oxides and carbonates present. A typical reaction is given as follows:

$$2\ ZnO + ZnS + 4\ HCl + Cl_2 + O_2 = 3\ ZnCl_2 + 2\ H_2O + SO_2$$

It is merely necessary that the ore be properly mixed with the reagents in a revolving drum or other suitable vessel. The author claims an efficient chlorination with absence of agglomeration of the ore. (972,149, Oct. 11, 1910.)

Zinc Oxide.—A patent of Messrs. WILLIAM SIMM and HENRY SIMM, of Portico, near Prescot, England, describes details of mechanical construction of apparatus for rapidly producing electrically zinc oxide from zinc ores such as calamins and metallic substances containing zinc. The features of the apparatus are sufficiently indicated by the second claim which relates to "the combination of a condensing chamber having suitable baffles, a hood connected therewith, and a suction fan for drawing the fumes into the hood and discharging them into the condenser, with an electric furnace removably located beneath the hood, a space being provided between the bottom of the hood and the top of the furnace whereby air for the supplying of oxygen is freely admitted on a level with the heating material carried by the furnace." (946,688, Jan. 18, 1910.)

Alloys.

Antifriction Alloy.—An inexpensive alloy, affording slight friction and sufficiently hard to reduce abrasion to a minimum, has been patented by JOSEPH R. STRATTON, of New Brunswick, Canada. His discovery lies in the fact that the addition of spelter to a combination of other metals produces an excellent antifriction metal suited for railway and heavy service. The formula preferred by the inventor is as follows:

	Pound.	Ounce.	Per Cent.
Copper	0	8	1.7
Tin	1	12	5.8
Lead	3	0	9.8
Spelter	25	0	82.0
Antimony	0	3	0.7
Total	30	7	100.0

The metals are melted together and thoroughly mixed in any preferred manner. (972,786, Oct. 11, 1910.)

Copper Alloy.—A new chromium alloy of copper compounded with nickel and other metals has been invented and patented by Messrs. JAMES NAULTY and JOHN SCANLIN, of Philadelphia, Pa. The alloy is of great tensile strength and approximates the hardness of ordinary steel, so that edged tools can be made from it. The alloy is made by melting in a crucible 5 lb. of chromium, with 5 lb. of aluminium of commerce used as flux. When thoroughly melted, after about an hour, 110 lb. of copper are added to the mass and melted. This requires about 30 minutes, after which 25 lb. of nickel are added and the melting continued for another hour. Finally 20 lb. of zinc are added. The ratio of chromium to copper and nickel to copper may be varied, but the order in which the metals are admixed and the use of aluminium as a flux are essential. The alloy has a slightly reddish tinge and gives a tensile strength of 78,950 lb. (974,410, Nov. 1, 1910.)

Ferrosilicon from Sulphur-Containing Iron Ores.—When using sulphur ores for the production of ferrosilicon a product containing a large amount of sulphur is obtained even in electric furnaces when the content of silicon is relatively low. But Mr. ARNOLD WIENS, of the Elektrochemische Werke, of Berlin-Bitterfeld, in Bitterfeld, Germany, has found that by increasing the silicon content by the addition of sufficient silica to the furnace charge the sulphur content can be greatly reduced. The reduction must, of course, be carried out in an electric furnace. For example, when obtaining a ferrosilicon

with 6.4% silicon, 3.1% sulphur is present;
" 8.5% " 2.8% " " "
" 12.0% " 2.4% " " "
" 22.0% " 0.12% " " "
" 31.4% " no sulphur is present

Therefore, if the ferrosilicon contains at least 20 per cent Si an entirely marketable alloy is obtained. "If the ore contains other metals besides iron, such, for example, as zinc, which is very often the case in common iron pyrites, it is essential that the material should be first roasted; substances are then added thereto, with which the remaining sulphur that is set free combines without being additionally driven off either by carbon or by silicic acid. By these means the zinc is recovered in receivers in the metallic form or as oxide, but without sulphur. Lime is generally used as such a sudphur-absorbing substance." The waste pyrites containing zinc and sulphur which remain from the manufacture of sulphuric acid can be treated in this way to recover zinc and ferrosilicon. "Zinc ores that are only reduced with difficulty can be added to the iron ores ,as in principle it is immaterial whether oxidized or sulphurized ore predominates. Further, it is of the greatest importance in the usefulness of the process that since the electric furnace is entirely closed a very pure carbonic acid gas is obtained under the circumstances." (973,732, Oct. 25, 1910.)

Furnace Slag.

Utilizing the Heat of Slag.—The generation of low-pressure steam from hot furnace slag is the basis of a patent recently granted to CLAUD VAUTIN, of London, England. An extended explanation of this system was given in our issue of May, 1910, page 297. It will be observed from Fig. 6, however, that the system involves the use of a slag hopper *C*, with a trap *C'*, from which molten slag flows into water contained in a tank *D*. As it enters the tank it is struck with a stream of water as shown at M^2, the water being drawn from the tank

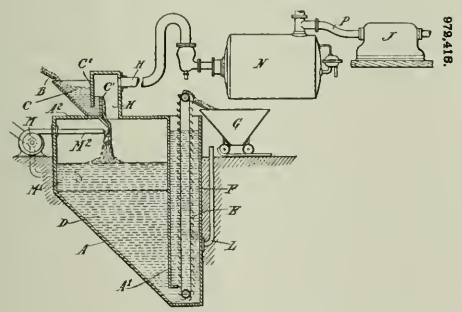


FIG. 6.—APPARATUS FOR UTILIZING THE HEAT OF SLAG.

through the pump *M*. The granulated slag which has given up its heat to the water falls to the bottom of the tank and is elevated through the elevator *A'* into a pot *G*. The steam generated is led past the inflowing stream of hot slag, and thence through an accumulator *N* to a low-pressure turbine *J*. (972,418, Oct. 11, 1910.)

Electric Furnaces.

Mixing by Induction.—Mr. S. Z. DE FERRANTI, of Grindleford Bridge, near Sheffield, England, who is one of the original pioneers of the electric induction furnace, has recently devised several modifications of the induction furnace and especially worked out methods of stirring and agitation by induction. One of his designs is an induction furnace with an oval annular melting chamber, in which the metal is heated by induction currents produced in the same way as in the ordinary induction furnace. The new feature is that at a certain part of this annular melting chamber special windings are provided with operative portions normal to the free surface of the metal, with iron masses, the arrangement being exactly analogous to the stator of an induction motor. These produce a rotary magnetic field which, acting on the metal in the melting chamber in the same way as on the rotor in an induction motor, produces a movement or rotation of the metal and thereby thorough stirring and mixing. Of course, the ordinary induction furnace may be left unchanged and the molten metal, after having been sufficiently heated, run off into a special mixing chamber, which is surrounded with windings to produce a rotary magnetic field. (962,013, June 21, 1910.)

Combined Open-Hearth and Electric Furnace.—A further development of the same idea by Mr. Ferranti is shown in Fig. 7. which represents a horizontal and vertical cross-section of a Siemens regenerative furnace with an electrical arrangement for mixing and circulating. The furnace *a* proper is of circular form, with three auxiliary channels *b* of comparatively small section in communication with the central portion and surrounding the three vertical limbs of the magnet core. On

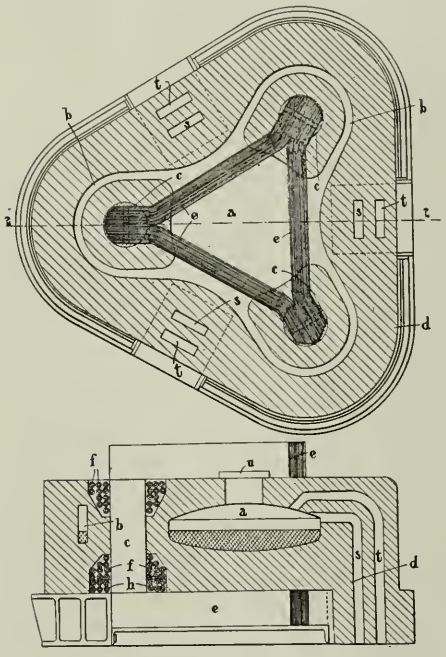


FIG. 7.—COMBINED OPEN HEARTH AND ELECTRIC FURNACE.

the limbs *c* of the three-phase arrangement coils *f* are wound above and below the furnace bath in the corners of the encircling magnets so that the turns of the coils are roughly in the form of a double cone with the furnace bath disposed in part in the angular spaces between the cones. Additional coils *h* are provided in the lower parts of the magnet limbs, so that the unsymmetrical field produced may rotate or mix the material acted upon. The central body of the bath is violently rotated by the leakage field, thus inducing a flow around the subsidiary channels *b*, which are intensely heated electrically. The entire mass is thus caused to pass continually through the channels *b*. When the metal is rotated in the central compartment a hollow forms in the center by centrifugal force and this may be made use of for drawing off the spent slag by means of a duct. The usual regenerators are indicated at *s* and *t*.

Among the various applications described is an intermittent acid process with separate complete heats for working iron with very low phosphorus and sulphur and high in carbon, manganese and silicon. The bath of the electric furnace is first filled with a charge of molten pit iron taken from the mixer or from the blast furnace direct. The bath is then electrically heated and mixed. Iron ore is added and the blast is started from the nozzles above the bath so as to "drive in oxide which is churned up with the metal by the blast and electric mixing.

The oxygen in the blast at the same time acts on the metal. The heat produced by the chemical reaction and the electric current is very great, and as the mixing is most complete the impurities are rapidly oxidized. The metal being spun around by the leakage field piles up on the sides and its surface assumes a dish form. In this dish so formed the reaction takes place, thus saving the walls of the furnace from the action of the slag. As soon as CO is given off, the blast may consist of air only so as to oxidize it to CO_2. The metal may have carbon added, for example, in the form of spiegel. Instead of pouring immediately, as in the Bessemer process, the metal may be kept in the furnace to be further heated and mixed and only poured after sufficient treatment, such as the addition of an alloy. The effect of this operation of heating and rotating is to free the metal from occluded gas, burned particles and slag. Slag may be used in this process to any desired extent." The treatment of iron high in phosphorus is also described. (969,261, Sept. 6, 1910.)

Induction Furnace Design.—Mr. Gustave Gin, of Paris, France, has designed an electric steel furnace which, in the form of his long-extended narrow channel, resembles much his older furnace with steel blocks as electrodes which attracted some interest when electric steel refining first came up for general discussion (our Vol. II, page 20). Mr. Gin has now given up his steel-block electrodes and heats the channels by induction. He still employs elongated, endless and parallel channels of uniform cross-section, with extended portions folded back into parallelism with each other. "The energizing magnet is so disposed relatively to said parallel portions that parts of latter lie relatively outside of the magnetic influence and so close to each other as to largely reduce self-induction and reactance, the distance of said parts from said magnet being much greater than the diameter of the horizontal cross-section of said magnet taken in the plane of said crucible." (970,700, Sept. 20, 1910.)

A patent of Mr. Paul Gredt refers to some details of construction of an oscillatory induction furnace with annular hearth and especially to methods of charging it by means of conveying worms which revolve in tubes surrounded by heating jackets, so as to preheat the fresh charge. Tapping holes are placed at different levels for drawing off either the slag or the molten metal. (950,911, March 1, 1910.)

Aluminium Furnace.—Mr. Heinrich F. D. Schwahn, of Belleville, Ill., has devised an electric furnace for the smelting of metals, and especially aluminium (his aluminium process having been noted in our September issue, page 547). The smelting chamber is a ring-formed retort and in the center of the ring the step-down transformer which supplies the electrical power is placed. Two sets of electrodes are provided; one set in the bottom of the retort, consisting of two plates made of powdered carbon and magnesite and connected to the secondary of the transformer. The other two electrodes are water-cooled copper or aluminium plates inserted in the sides of the retort, near its top. They are connected either to a source of direct current or also to the secondary of the transformer. Graphite pipes are inserted in the walls for the introduction of reducing or fluxing gases into the furnace. Above the annular furnace chamber a gas-hood is provided formed of two concentrical cones; the space between these two cones is used for carrying off the spent gases and for introducing and preheating fresh charge. The outside air is excluded by this means from the furnace. (962,532, June 28, 1910.)

Electrolytic Furnace.—Mr. H. A. Danne, of Clarksburg, W. Va., employs a cylindrical crucible which is closed on the top by an annular cover with an open hole in the center. Upon this annular cover, so as to fit into the central hole, a shaft is placed which receives the charge from above, revolving blades being provided for stirring. At the bottom of this shaft (hence at the top of the crucible) the positive electrode is provided, formed of powdered carbon cemented together and suitably reinforced. This electrode increases in diameter downwardly in step-by-step fashion (or in a frusto-cone shape). This positive electrode is surrounded by groups of negative electrodes suspended from the annular part of the crucible cover; each group is disposed in a different horizontal plane corresponding to the step arrangement of the opposite electrode. The furnace is intended for continuous operation (for instance, in the manufacture of aluminium). When the charge is decomposed between the electrodes it passes away and new charge is fed between the electrodes. The fresh charge descending from above, is thoroughly preheated. (958,855, May 24, 1910.)

Calcium Carbide.—A patent of Mr. James E. Hewes, of Plattsburg, N. Y., refers to an electric arc furnace for the production of calcium carbide. Below two oppositely directed horizontal water-jacketed electrodes and at a slightly lower level, a traveling sectional hearth is employed for supporting the carbon and lime to be smelted. This hearth consists of a series of carriages with bottoms and sides, but no ends, and abutted end to end to form a long trough-like structure. As this hearth travels the carriages are added at one end and removed with the carbide at the other end. (969,622, Sept. 6, 1910.)

A patent of Mr. Herman L. Hartenstein, of Constantine, Mich., refers to details of construction of tap-hole for calcium carbide, the object being to prevent the tap-hole from being clogged. (971,445, Sept. 27, 1910.)

Cyanamide Furnace.—In our September issue, p. 548, we described the process of Messrs. E. H. Meyer and J. M. A. Stillesen, of Niagara Falls, Ont., Can., for the production of calcium carbide and cyanamide. Details of construction of their revolving inclined cyanamide furnace, which is charged from the bottom and in which the relative distance between the electrodes is automatically controlled according to the load, are given in their patent 955,655 of April 19, 1910.

Electric Glass Furnace.—Various forms of electric furnaces for the continuous manufacture of glass are described in two patents of Mr. Marius Sauvageon, of Columbes, France. All of them have in common the combination of a relatively small smelting and refining chamber with one or two larger working chambers. The smelting chamber contains two electrodes, either at the two ends or one above the other. The furnace acts as a simple resistance furnace, the molten glass being the resistor. Fresh raw material, frit, is continually fed onto the top of the molten glass in the melting chamber, so that the electrically produced heat in the molten glass is employed to melt down the frit. The refined glass passes through a passageway into the larger working chamber, where it is cooled to a suitable working temperature. (972,778 and 972,779, Oct 11, 1910.)

Manganates.—Mr. F. J. Machalske patents a process of producing alkali manganates by heating a mixture of the alkali haloid and manganese dioxide in presence of air in an electric furnace. For instance, by heating two parts of sodium chloride and one part of manganese dioxide he produces one part of disodium manganate and two parts of free chlorine. The chlorine may be collected separately or it may be used for converting the manganate into permanganate according to the equation $2Na_2MnO_4 + 2Cl = 2NaCl + Na_2MnO_4$. The sodium manganate may be treated with water, three parts of the manganate and two parts of water giving one part of manganese dioxide, one part of sodium permanganate and four parts of sodium hydroxide. The manganese dioxide may be returned to the electric furnace as part of a new charge. The sodium permanganate may be converted into potassium permanganate by causing it to react with potassium chloride. (956,120, April 26, 1910.)

Electric-Arc Furnace.—A kind of electric-arc reverberatory furnace is the subject of a patent of Mr. J. F. Shawhan, of Dayton, Ohio. A current of air is passed over an electric

arc formed between a set of electrodes, and the heated air is pressed by air pressure into a working chamber which may be the hearth chamber of a reduction furnace or of a reverberatory furnace. The electrodes are placed outside of the furnace and the arc is formed in a twyer sleeve of refractory material opening into the working chamber. The air is passed through this sleeve and is heated by the arc and by contact with the muffle walls. The air blast is forced at such a pressure that the arc is drawn out and deflected into the working chamber. This blast of air performs a threefold function, it carries the heat necessary for reduction or refining, it furnishes the oxygen and it blows up the slag. For drawing out the arc electromagnetic means may also be used. When the arc is drawn out it is maintained at the extremities of the electrodes so they keep their rounded form and burn off evenly. (951,458, March 8, 1910.)

Revolving Arc Furnace.—A patent of Mr. CHARLES A. WEEKS, of Philadelphia, Pa., refers to details of mechanical construction of an arc furnace with a ring-formed furnace body and a cylindrical furnace chamber in the center. One or several pairs of electrodes are provided in the center and the whole furnace revolves so that its lining is worn out uniformly. (949,511, Feb. 15, 1910.)

Arc Furnace.—A batch of patents, granted to Mr. J. H. REID, of Newark, N. J., relates of various constructions of arc furnaces. In one type a fixed bottom electrode is provided in form of a thick carbon rod. Above this and inclined to both sides are adjustable electrodes adopted to form a hopper to receive the ore. Arcs are produced between the inclined side electrodes and the bottom electrode and act on the ore. The position of the side electrodes is automatically regulated so as to permit "only ore in a more or less molten state to pass through." It then drops directly into the quenching tank (947,849, Feb. 1, 1910). In another type of furnace the charge, after having been subjected to the action of the electric arcs, is then subjected to the action of gases passed into the furnace through pipes inserted in the walls below the electrodes; these gases are sucked in by exhausting the upper part of the furnace. (950,116, Feb. 22, 1910.) The product of the furnace process, together with the slag, is then withdrawn into another cylindrical vessel which is heated by means of electrodes inserted through the walls and which is rotated, the object being to separate "the metals of different specific gravities by rotation of the containing vessel." The vessel is at the same time exhausted to remove the metals which volatilize. (950,028, Feb. 22, 1910.) Instead of passing the charge through a single furnace chamber as described above, several furnace chambers may be arranged on top of each other so that the charge, having passed through the action of one set of electrodes in one chamber, is subjected again to the action of another set of electrodes in the next lower chamber and so on. (968,079, Aug. 23, 1910.) Provision is made for independently and automatically regulating the movement of any electrode according to the current passing through it, while the movement of all the electrodes is effected from a single source of power. (968,603, Aug. 30, 1910.)

Electrode Support.—Mr. ISAAC R. EDMANDS, of the Electro-Metallurgical Company, of Niagara Falls, has patented details of adjustably supporting the depending electrodes used in electric furnaces, the electrode being counterbalanced by counterweights by means of flexible conductor ropes. The mechanical arrangement selected permits of a wide range of adjustment of the electrode, while requiring but little head room above the furnace. (959,486, May 31, 1910.)

Various Electric Furnace Processes are also described above under Iron and Steel, Zinc and Alloys.

Electrolytic Processes.

Treatment of Copper-Nickel-Iron Alloys.—An electrolytic process for the treatment of nickel-copper or nickel-copper-iron alloys for the production of copper and a nickel-iron alloy has been devised by Mr. P. F. COWING, of New York City, the process being shown diagrammatically in Fig. 8. The original nickel-copper-iron alloy is used as anode 1 in the diaphragm cell *A*, a warm concentrated neutral aqueous solution of nickel chloride being used as the initial electrolyte in the anode compartment. By anodic action the metals of the anode *A* pass gradually into solution, which then contains nickel chloride, ferrous chloride and cuprous chloride. This anodic solution is withdrawn into a second electrolytic cell *B* in which selective deposition is carried on, copper alone being deposited on the cathode 7 by maintaining a sufficiently low voltage. The anode 5 consists of nickel or iron or both. The solution is heated and agitated by means of the stirrers 7. Most of the copper is here deposited from the solution which now contains essentially nickel and ferrous chlorides and some cuprous chloride. This solution is passed into the tumbling barrel *C*, where the rest of the copper is removed by cementation on iron scrap 11. The solution, which now contains nickel and ferrous chlorides, is run into the settling tank *D* and then lifted by means of the pump 13 into the heating chamber *E*, from where it is passed

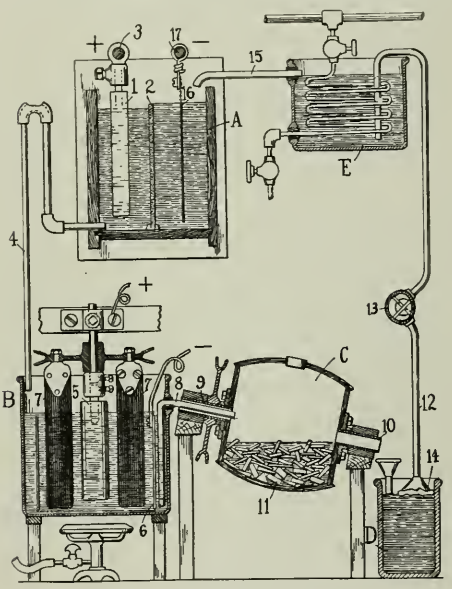


FIG. 8.—TREATMENT OF NICKEL-COPPER.

into the cathode compartment of cell *A*, where an iron-nickel alloy is deposited on the cathode 16. The solution in the cathode compartment is permitted to flow over the diaphragm 2 into the anode compartment, thereby completing the circuit of flow. Throughout the process the iron and copper salts are formed and should be maintained as far as possible in their reduced condition as ferrous and cuprous salts. As no insoluble anodes are used the electrolyte remains at all times electrochemically balanced. The anodes should contain as small a percentage of sulphur as possible to keep down the formation of sulphates, and these, if formed, may be removed by the occasional addition to the solution of barium chloride. (969,773, Sept. 13, 1910.)

Production of Ferric Salts.—A recent patent of Mr. ANSON G. BETTS, of Troy, N. Y., refers to the production of ferric salts by the electrolysis of ferrous salts of polybasic acids, for example, ferrous sulphate and ferrous fluosilicate. One of the difficulties encountered is polarization at the anodes and destruction of the carbon anodes. This may be overcome

by employing a large relative movement between anode and electrolyte. But without such movement it may also be overcome simply by electrolyzing at a high temperature. For instance, a solution containing 2.8 grams iron as ferric sulphate, 1.6 grams iron as ferrous sulphate, 1.75 grams copper as copper sulphate and about 4 grams free sulphuric acid per 100 c.c. can be electrolyzed at a temperature of 86° C., with a graphite anode, at an anode current density of 11 amp per square foot, without showing more than slight extra polarization beyond that necessary for the formation of ferric sulphate. The same solution at a temperature of 70° C., with a current density of 8.5 amp per square foot, shows an extra polarization of half a volt or over, which is still greater at lower temperatures. With a current density of 14.5 amp per square foot there is an extra polarization of over half a volt at 84° C., showing the necessity of using a still higher temperature at the higher current density. By circulating the electrolyte, or moving the anode, a lower temperature may be used In fact, especially good results are obtained by empoying considerable relative motion and some heat. (969,833, Sept 13, 1910.)

Hypochlorite Cell.—While in the production of caustic soda and chlorine by electrolysis of common salt solutions the object must be to keep the chlorine produced at the anode away from the caustic at the cathode, the opposite is required in the electrolytic production of hypochlorite (bleaching liquor) solutions from common salt. Here the anodic chlorine must be made to react with the cathodic caustic. Mr. Paul McDorman, of the National Laundry Machinery Company, of Dayton, Ohio, has devised a cell in form of a rectangular box through which the liquid (sodium chloride solution) is forced to pass in a circuitous path. An anode is provided at one end and a cathode at the other end. Between them there is a series of electrodes acting as bipolar electrodes; that is, they are not connected to the external circuit, but the electric current passes through them from the liquid and into the liquid so that one face of each of these plates acts as anode, the other as cathode. These intermediate bipolar electrodes do not fill the whole cross-section of the cell, space being left above and below each electrode These spaces are filled with glass plates, one being inserted above and below each electrode. In order to permit the solution to pass, recesses are formed in the glass plate above the first intermediate electrode, in the glass plate below the second electrode, in the glass plate above the third electrode, and so on. The solution is therefore forced to pass through the recess over the first electrode, then through the recess below the second electrode, and so on. (968,492, Aug. 23, 1910.)

Electrolytic Cell.—An electrolytic cell in which special provision is made for circulation of the electrolyte and for artificially cooling it is patented by Messrs. Gilbert C Lansis, of Wadsworth, Ohio, and Jacob H Smith, of Rittman, Ohio (assigned to Ohio Salt Company). The cell contains a series of electrodes and the electrolyte is passed from one end to the other, passing over the top of the first electrode, below the bottom of the second, over the top of the third, and so on Between each two electrodes a glass coil is provided through which cooling water is passed. (974,576, Nov 1, 1910.)

Chemistry.

Synthesis of Ammonia.—Mr H S Blackmore, of Mount Vernon, N Y., proposes to make ammonia as follows A vessel containing mercury is subjected to agitation and then nitrogen and hydrogen gas is introduced into the mercury and thereby caused to unite and form ammonia NH_3, some of which, together with a portion of the hydrogen, unites with the mercury producing ammonium amalgam, which may be finally recovered as ammonia by heating the amalgam so formed It is recommended to heat the hydrogen and nitrogen gases Instead of passing the nitrogen and hydrogen through agitated mercury the mercury may be atomized with a blast of compressed nitrogen and hydrogen. It is recommended to use mercury containing calcium, as the latter facilitates the union of nitrogen and hydrogen by absorbing the hydrogen at ordinary temperatures, transiently forming intermediately calcium hydride, which is broken up into ammonia by action of the nitrogen. If methane or natural gas is used instead of hydrogen, methylammonias are produced, such as methylamin, dimethylamin, trimethylamin, or methyl ammonium compounds. By employing such substances as methane or natural gas, nitrogen may be fixed in such a manner as to form compounds or salts with suitable acids, which are valuable as fertilizing agents. By passing a current of electricity through the mercury during agitation with the nitrogen and hydrogen "union and formation of ammonia is further facilitated by the numerous electric sparks produced by the rapid completing and breaking of the electric circuit throughout the mass during agitation." (974,633, Nov. 1, 1910; modifications are described in two other patents of Blackmore, 974,741 and 974,742, Nov. 1, 1910.)

Cleaning Furnace Gases.

Dust Arrester.—A device for use primarily in the manufacture of sulphuric acid has recently been patented by John Hickey, of Buffalo, N. Y. The invention relates to dust arresters of the class comprising a group of baffles or deflectors through which the dust laden gas is caused to flow and which intercepts the dust. In Fig. 9, which is a fragmentary elevation of the device, A represents a chamber in which the collector B is located. The gas enters at a, coming from the furnace, and has an exit at a', passing to the chambers. Spaced baffles or deflectors D are supported in the chamber. They are con-

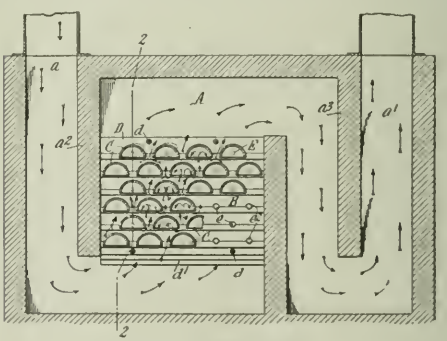


FIG. 9.—DUST ARRESTER.

siderably wider than the intervening spaces, so that the rising gases impinge on the concave surfaces of the baffles and are deflected in two directions against the convex upper surfaces of the two baffles below. The dust is thus allowed to settle on the convex surface, from which it occasionally falls onto the baffles below and thus to the bottom of the chamber. The baffles may be of an inverted V shape instead of semicircular as shown. Provision is made for the rocking of a tier of baffles should it become necessary to dislodge dust from them. (972,534, Oct 11, 1910.)

Gas Washing Machine.—For cleaning producer or blast furnace gas, removing dust, tar and other impurities a special machine has been patented by Willis H Carrier, of Buffalo, N Y. The patent is assigned to the Buffalo Forge Company, Buffalo. The machine comprises a casing enclosing three impellers or fans, each rotating within a chamber The impellers rotate together and may be regarded as parts of a single compound impeller. As the impellers rotate rapidly the impure gas is drawn through an intake where it is wetted by a spray Passing into the first impeller chamber the gas is set in rapid motion, and the heavier particles of tar, etc., are thrown out into an impact chamber at the periphery of the

impeller. From the impact chamber the gas passes into the spray chamber, where it is again set in motion by the second impeller. Here it is caused to meet sprays of water which further wash the gas, while the rapidly revolving impeller throws the condensed impurities to the periphery of the impeller, where they are removed. Passing through the second and third impellers the gas is dried and flows out of the machine. In its passage out, however, it automatically operates a pressure device so that the machine delivers gas at practically constant pressure. In order to prevent the condensation of light tars which have remained gaseous during the passage through the machine the gas is passed through a superheater before going to the gas engines. This prevents the condensation of these products in the gas engine. (973,626, Oct. 25, 1910.)

Synopsis of Metallurgical and Chemical Literature

Gold and Silver.

Refractory Low-Grade Ore.—Rather unusual means taken to recover gold from a refractory ore are described by Mr. R. H. KENDALL in the October *Bulletin* (No. 75) of the Inst. of Min. and Met. The ore of the Ouro Preto mine, in Brazil, is composed of quartz, tourmaline, arsenical and iron pyrites, some bismuth and free gold up to nearly 60 per cent of the fire assay. Of the latter, about 5 per cent will remain on a 40-mesh screen, 45 per cent on 100-mesh and 50 per cent will pass 100-mesh screen. Amalgamation was originally tried, but owing to the bismuth present the process was not successful, and the old-fashioned practice of recovering free gold on blankets was installed, supplemented by the cyanide process. The efficiency of this system is excellent, showing a total recovery of about 92 per cent on ore worth $7 to $8 per ton.

There are two mills of practically the same design and equipment, located one below the other on the hillside. The equipment of each comprises Blake crushers, 40 stamps of 750 lb. weight, blanket tables, Frue vanners, classifiers and cyanide tanks. Water-power is used and the stamps are driven by double vortex turbines at a speed of 96 drops per minute, with an 8-in. drop and discharge of about 4 in. The screen is 30-mesh and the average tonnage per stamp is 3.2 tons, but owing to the fact that the mill is usually ahead of the mine, only 2.8 tons are treated per stamp per day.

Some coarse gold, amounting to about 6 per cent of the assay value, is recovered from the mortar boxes. The pulp from the stamps is run over blanket tables set on a gradient of 1½ in. per foot. Each blanket is 18 in. wide and 4 ft. long. There are four blankets to each set of five stamps, or 32 in all. They are changed two or three times an hour, the work of changing all blankets requiring about 10 minutes. The blanket concentrate contains practically all the free gold in the ore, and is subjected to successive treatment on other blanket tables and in bateas whereby the final residue is enriched to about 800 fine. The ratio of concentration effected before gold is first recovered is 10,000 into 1.

The pulp flowing from the blankets in front of the stamps is treated on Frue vanners, but before flowing onto the belts it is treated on shaking blankets fixed to the distributors of the tables. A rich concentrate is obtained from these blankets, amounting to 6-7 tons per month and assaying about 16 oz. gold per ton. The vanners produce about 300 tons per month of concentrate assaying about 1.2 oz. gold per ton.

The vanner concentrate, together with the residue from the reconcentration of the first blanket concentrate, is treated in the concentrate cyanide plant. The vanner tailing, assaying about $1.20 per ton, is classified into sand and slime, and each product is treated by cyanide. There are thus four distinct products treated by cyanide, viz., the *rich concentrate* from the shaking blankets on the vanner distributors, to which is added the residue from the last regular blanket concentration; the *ordinary concentrate*, which is made up of the vanner concentrate and the residue from the first regular blanket concentration; the *sand* from the vanner tailing, and the *slime* from the same.

The *rich concentrate* contains about 16 oz. gold per ton, and is treated only once a month. It is mainly arsenical pyrites. A lime-water wash is given, after which the concentrate is dried by vacuum filtration and dumps onto a floor where it is allowed to air for 24 hours preceding cyanidation. This operation is conducted in agitator tanks; working strength of solution, 0.3 per cent KCN; proportion of liquor to concentrate, 1:1; time of first treatment, 12 hours. They are then filtered, washed and retreated with cyanide for 15 hours. The first treatment reduces the value to about 1 oz. gold, and the second to about 0.5 oz.; KCN consumption, 11 lb. per ton; zinc, 2 lb. per ton. The final residue is stored for six months and again run through the mill mixed with ore.

The *ordinary concentrate* is cyanided in practically the same manner as the rich concentrate. Its value is about 1.12 oz. gold per ton and gives an extraction of about 90 per cent. Proportion of solution to concentrate, 60:100; time of agitation, 22 hours; solution, 0.22 per cent KCN; consumption of KCN, 2.75 lb. per ton, and of zinc ¾ lb.

The *sand* has a value of about 80 cents to 85 cents and is treated in 24-ft. percolators served by one distributor. The time of the cycle of treatment is as follows:

Filling	16 to 18 hours.	
Drying and airing	4	4 "
Circulation of solution	36	36 "
Removing liquor and washing	6 "	8 "
Sluicing out	2 "	3 "
	64 to 69 hours.	

When a percolator is charged with sand, sump solution of 0.08 per cent KCN is run on, followed by slightly turbid gold liquor from the slime plant. After the latter is all on, circulation of solution is started and continued for 36 hours. Charge, 115 tons to 120 tons of sand and about 40 tons of solution. The value of the residue is about 20 cents. Consumption of cyanide, 0.4 lb. per ton, and of zinc, 0.08 lb.

The *slime* is thickened and run to dewatering vats, where it is dewatered by a set of vacuum filter frames lowered into the pulp. The dewatered cake is lifted and discharged into the extraction vat previously filled with KCN solution. Three journeys of the filter frame are required to fill one extraction vat. After agitation in this vat, the pulp is run into another tank, where the same vacuum procedure is carried out to remove the gold solution from the slime, which is washed and discharged as usual. The average value of the slime is about $2 per ton, and an extraction of 79 per cent is obtained.

Excellent precipitation is obtained in the zinc boxes, the sump solutions rarely running more than a trace of gold. Ninety per cent of the recovery is made in the first compartment. The precipitate is treated with H_2SO_4, filtered, roasted, fluxed and melted, the resulting bullion running 940 to 960 fine. The free gold recovered on the blankets and in the mortar boxes is refined and melted, making bullion over 900 fine.

Crushing and Grinding Devices.—In an endeavor to find a suitable combination of coarse and fine crushing machines, both working under the best conditions, Mr. G. A. DENNY presents in the November, 1910, issue of the *Mexican Mining Journal* some observations on the subject. After reviewing the history of the stamp, the author finds it remarkable that this device has retained its position of eminence as a grinder. He shows that the only practical development in stamp machinery is in the increase in weight, without sufficient evidence adduced to show that the increase is necessary or efficient. He criticises the manner of calculating the mechanical efficiency of a stamp, stating that no allowance is made for the work done by previous machines, whereas in calculating the mechanical

efficiency of the tube mill the custom is to deduct the mechanical energy already expended by the stamps on the tube-mill feed.

The efficiency of the stamp falls off rapidly when crushing through fine screens. The author shows this by comparing an 1150-lb. stamp crushing 9 tons per day through an 8-mesh screen, with a 900-lb. stamp crushing 4 tons per day through a 20-mesh screen. The mechanical efficiency of the former is calculated by Stadler at 70.4 per cent and of the latter (by the author) at 45 per cent, ignoring the work of previous machines in each case. The author assails the economic usefulness of the stamp at both limits of its functions, coarse and fine crushing, showing that breakers are more efficient coarse crushers and other machines more efficient fine grinders, so that there must be a critical point where the stamp performs its function most efficiently. The author's opinion is that the economic weight of stamps is from 1000 lb. to 1250 lb., and the best average mesh is 20.

Concerning Rand practice of heavy stamps and tube mills, the author finds room for criticism in that the material delivered by the stamp at 70 per cent efficiency is fed to a machine that completes the grinding at only 17 per cent efficiency. Since the heavy stamp reduces only 25 per cent of its product to the required degree of fineness, it leaves 75 per cent of the material to be retreated in a machine of lower efficiency. He believes that lighter stamps would reduce 75 per cent of the product to the proper fineness, leaving only 25 per cent to be retreated in the mill of lower efficiency.

Iron and Steel.

Composition of Some Tool Steels.—According to L. Klusmeyer, in the composition of tool steels the following elements influence the quality of the steel in a marked degree: Phosphorus lowers the resisting power of steel against impacts and makes it cold-short brittle. It gives a coarse crystalline structure to the steel. Sulphur lowers the strength of steel at a dark-red heat and makes it red-short brittle. Silicon lowers the coefficient of expansion. Manganese in small quantities eliminates the bad influence of sulphur and silicon. In too great quantities it makes the steel brittle. Chromium and tungsten increase hardness without influencing the expansion in the same measure as carbon does. Chromium and tungsten steels at same hardness are stronger than carbon steels. The percentage of phosphorus in first-class steel is at times not more than 0.008 per cent, and moves within the limits of 0.01 per cent to 0.03 per cent; sulphur percentage is 0.03 at the most and copper is 0.25. As manganese raises the malleability of steel, from 0.2 per cent-0.3 per cent is added. Silicon, in amounts of 0.2 per cent, facilitates the production of good castings. The percentage of carbon wavers between 0.75 per cent and 1.50 per cent, and depends on the amount of hardness desired. Following is a comparative table of several steels:

	Carbon.	Silicon.	Manganese.	Phosphorus.	Sulphur
Stirian tool steel (Steiermark)	1.12	not deter.	0.23	0.023	0.016
Tool steel—Etienne (Ledebur)	1.00	0.06	0.08	0.02	0.015
Tool steel—Kappenberg (Ledebur)	0.92	0.09	0.12	0.02	0.005
Sheffield steel (Bischoff)	0.75	...	0.23	0.04	0.022
German crucible steel (Bischoff)	1.24	not deter	0.15	0.016	0.016

Manganese, chromium and tungsten increase the natural hardness of steel in a marked degree; silicon and phosphorus increase it slightly. High-speed steels have a large percentage of chromium and tungsten with a small percentage of manganese and silicon.

Following is a table of the composition of chromium tungsten high speed steels:

	Chromium.	Tungsten.	Carbon.	Manganese.	Silicon.
Bedel & Co., Paris	8.10	22.80	0.90	0.47	0.20
Austrian Phönix steel	3.70	20.70	0.67	0.14	0.15
English Novo steel	2.95	18.85	0.76	0.42	0.33
Austrian high-speed steel	7.19	24.50	0.03	0.23	0.24
Universal steel	6.50	23.50	0.60	0.14	0.42
Kysar steel	1.22	18.77	0.87	0.27	0.11
Armstrong steel	3.40	12.41	0.78	0.40	0.44
German steel	3.70	30.20	0.60	0.24	0.23

(*Werkstatts Technik*, October, 1910.)

Electrical Steel Furnace at Dommelsdingen.—Mr. Herman Thieme gives a description of the new electric steel furnaces of the Gallais-Metz Company, in Luxembourg, in *Elektrotechnische Zeitschrift*, Sept. 8 and 15. The electric-steel plant has two single-phase induction furnaces of three or four tons capacity and one three-phase induction furnace of 1.5 ton capacity. The tilting devices of the furnaces are controlled by a 10-hp motor. To safeguard against tilting the furnace at too great an angle an electrically operated differential band brake stops the tilting device at the proper angle of about 45 degrees, and when tilting back to draw slag at an angle of 17 degrees from the horizontal. The furnace chamber is built up of iron plate 18 mm thick, surrounding the transformer cores and lined inside with firebrick. After the firebrick lining has been dried, magnesite or dolomite, with tar as binding material, is stamped in. To dry and preheat the lining, wrought-iron rings are placed in the furnace and are heated by induced currents to a bright-red heat. About 180 kw are employed for this preliminary drying and heating operation, which lasts about eight hours; this is for the single-phase furnaces which consume 380 kw in normal operation. The furnaces are started by means of oil switches from the charging platform. All instruments and regulators and controlling devices are installed on the furnace platform. A special advantage of the three-phase furnace over the single-phase furnace is that the rotary magnetic field produced in the former provides an excellent mixing and stirring of the charge.

Zinc.

Condensation of Zinc Vapors.—In *Metallurgie* of October 8 Mr. A. Roitzheim discusses the condensation of zinc vapors to liquid zinc and the reasons for the formation of zinc dust. He finds that to obtain zinc in liquid form it is all important to cool the zinc very slowly and gradually from the higher temperature at the entrance into the condenser to the temperature of liquid zinc. If the vapor is cooled too suddenly zinc dust is obtained. The dilution of the zinc vapors by carbon monoxide in the retort does not seem to offer difficulties with respect to condensation. But the carbon monoxide seems to have a certain effect on the structure of the zinc. As to the temperature variations from muffle to condenser the following data is given: The reduction of the ore in the muffle takes place according to the equation $2\,ZnO + C_2 = 2\,Zn + 2\,CO$. The two terms on the right-hand represent the gas mixture which enters into the condenser. Under normal atmospheric pressure the partial pressure of the zinc vapor is, therefore, 388 mm mercury. At this pressure a temperature of 867° C. is required to have equilibrium between liquid zinc and zinc vapor (this temperature being obtained by graphical interpolation from figures given in R. Schenck's "Phys. Chemie der Metalle," page 5). Since the temperature of reduction of zinc oxide in the muffle is in practice between 900° C. and 1400° C. there is no danger of the formation of liquid zinc in the muffle, and the entrance to the condenser is also heated above 867°. The other end of the condenser has a temperature of 500° C. At this temperature the tension of the zinc vapor is only about 5 mm mercury, so that by far the largest part of the zinc has been condensed. The balance is still in the form of vapor and is caught outside as zinc dust. The volume of this "lost" zinc vapor is $V_{zn} = 5/755\ V_{co} = 1/151\ V_{co}$. It is thus seen that theoretically 100/151, or 0.662 per cent of the zinc content of the charge, would be obtained as zinc dust. Under correct operation in practice 3 per cent are obtained in form of dust, due to the difficulty of maintaining proper conditions of operation.

Influence of Steam and Hydrocarbons on the Roasting of Zinc Blende.—Mr. Felix Thomas in *Metallurgie* of October 8 and 22 reviews the results of experiments conducted by him on the influence of steam and hydrocarbons on the roasting of zinc blende. The favorable results obtained by briquetting zinc blende with some binding material, rich in hydrocarbons,

and by the use of air saturated with steam for reducing the ore, caused investigations on the influence of steam on ZnS and its products of oxidation. The investigations led to the following results: Decomposition of $ZnSO_4$ by steam begins at 600° with moderate speed and ends at 820° quickly and can, therefore, be carried out in practice at a temperature lower by 100° than when dry air is used. Steam begins to act on Zns at 650°, and under the conditions in which the decomposition of the sulphate in a steam-saturated atmosphere is completed the decomposition of ZnS is 30 per cent when the air is shut off. This observation would make a simplification of the present roasting process possible under the following conditions: The ore briquetted with a fuel containing hydrogen, for instance, tar. The ore briquets are roasted by the oxidizing flame of a fuel rich in hydrogen (gas, by-products of oil distillation, producer-gas, water-gas, etc.) at a temperature from 700° to 750°; near the end of process, the temperature is raised to 820°.

A marked acceleration of the roasting process is gained if a good furnace is chosen, for instance, with movable grates, so that the roasted surface layers of the briquets are scraped off from the unroasted core in the center and are brought into the hottest part of furnace, where the last traces of sulphates and sulphides will be removed. The practical advantages which can be obtained are a lower roasting temperature, and, therefore, less loss of ZnO, less reduction of ZnO to Zn and less danger of sintering and smelting through agency or metallic sulphides, Cu_2S, PbS, Sb_2, S_3, etc.; greater economy of time and fuel, especially if the roasted material is removed immediately, as suggested above, the possibility of using to good advantage mechanical roasting furnaces. Manual labor, as used at present, can be eliminated entirely. At the low temperatures used in the process iron parts could be used for the furnace without danger. More satisfactory results are obtained, since the roasted material is separated from the unroasted material, and the possibility of retaining sulphur is not so great.

Bisulphide Process.—Reference has been made previously in these columns (Vol. VIII, No. 10, October, 1910, pages 591 and 595) to a process for the treatment of zinc-lead ores by roasting and leaching with sulphurous acid. In this connection the *Australian Min. and Eng. Rev.* chronicles in its issue for Sept. 5, 1910, the plans for construction of a plant at Rosebery to treat ores by this process. The details are practically the same as those indicated in our former synopsis and patent review.

Aluminium.

Use of Aluminium in the Gas Industry.—The *Revue Des Mataux Et Alliages*, of October, 1910, brings an article on the use of aluminium in the gas industry. Aluminium seems to be unaffected by sulphuric gases, sulphuric acid, hydrogen sulphides and gases generated in the gas retorts, which are more or less compounds of the cyanogen group of gases. It is not attacked by ammonia compounds either, and, therefore, well adapted to the replacing of copper in many ways. Compared to brass aluminium offers an even greater resistance to the sulphur elements in gases, and can be substituted for brass and bronze with advantage. Aluminium tubes of convenient diameter could easily be substituted for metal now used in gas fixtures and installations. Where aluminium installations have been tried they have given excellent results and have proved more economical than brass fixtures. The metal is stronger, of good appearance, makes further decorations unnecessary, and is not affected by the effects of the combustion of gas. Aluminium has been used in French gas plants for manometers and measuring instruments and has proved a great success.

Chemical Engineering.

Oil Shale Industry in Australia.—Early references to oil shale deposits in New South Wales date from 1807. In the Sept. 5, 1910, issue of the *Aust. Min. and Eng. Rev.*, occurs a description of the most recent developments of the oil shale deposits there, and the plant in which they are treated. The Commonwealth Oil Corporation has crude oil and refining works at Torbane and Hartley, respectively, and has in course of erection extensive plants at Newnes.

The richer portion of the shale is exported for gas enrichment, and the remainder is crushed and charged into retorts of which there are two benches of 32 and 40 retorts, respectively. Producer gas is used as fuel. Blowers exhaust the oil vapors from the retorts through mains and a series of condensers, where the gases are cooled and the crude oil drained off. The ammonia is removed in scrubbers and the uncondensable portion of the gas is returned to the retorts as fuel. The crude oil is shipped to Hartley from Torbane, where it is treated with sulphuric acid to remove tar, and then with caustic soda to neutralize the acid. The oil is then distilled with steam, and by fractional distillation the various products are obtained. The paraffin base is filter-pressed to separate it from contained oil, after which it is solidified into cakes, which are treated in the sweating-room until the wax is of a whitish color. It is subsequently treated with floridine prior to filtration, after which it is ready for candle making. A residue from the stills is utilized as carbonaceous fuel.

At the Newnes works the first bench of retorts is completed. It comprises 64 retorts, 56 ft. high, the lower half being of firebrick and the upper of iron. Producer gas for firing is consumed around the brick column, and the uncondensable gases from the distillation of the shale are also returned to the retorts for fuel; in fact, once the retorts are started the shale itself produces almost sufficient fuel gas. The operation of the retorts is continuous, the shale being fed at the top and discharged continuously at the bottom by means of automatic drawing gear. The paraffin extraction department is equipped with an ammonia refrigerating plant. The tar removed by treatment with acid and alkali is used as fuel in the distillation department.

Radioactivity.

A New Radioactive Mineral.—A new radioactive mineral named Pilbarite, frim its discovery and occurrence in the Pilbarra gold field, Australia, is described in the Sept. 7, 1910, issue of the *Aust. Min. Standard* by Mr. EDWARD S. SIMPSON. The mineral occurs in a tantalite lode in nodules varying in size from that of a pea to a walnut, and has a bright canary-yellow color. Other properties are: Specific gravity, 4.4 to 4.7; hardness, 2.5 to 3; composition, hydrated silicate of lead, uranium and thorium; uranium trioxide, 27.09; thoria, 31.34; lead oxide, 17.26; silica, 12.72, and water, 7.56 per cent. According to Rutherford and Boltwood's value for the ratio of radium to uranium in such minerals, Pilbarite would contain about 7.5 centigrams of radium per ton. Under treatment by acids the mineral gives off gas which is neither oxygen nor carbon dioxide, and is assumed to be helium, although the author was unable to prepare spectrum tubes of it. Pilbarite is strongly radioactive, and when placed in a Glews scintilloscope yields a rapid succession of flashes due to the impact of emanations on the zinc sulphide screen.

The Amethyst Coloration of Crystal Glass.—In some notes on radium research Mr. J. H. NIEMANN advances the idea that radium is the main agent in the amethyst coloration of glass. The article appears in full in the *Aust. Min. Standard* for Sept. 21, 1910. The coloration is usually ascribed to the slow peroxidation of manganese or to the effect of violet solar rays. The author finds, however, that this coloration will occur under conditions which preclude the acceptance of these explanations. Glass becomes colored in some sections much more rapidly than in others; the colored glass acts radioactively, stimulating the growth of plants, and darkening photographic plates; the colored glass decomposes water, and commonly explodes after having been highly colored. The author regards these phenomena as tending toward his idea that radium emanations are responsible for the coloration of the glass.

Notes on Chemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

The Iron and Steel Institute.

At the autumn meeting of this institute at Buxton the reading and discussion of papers extended over three days, and the chair was occupied by the president, the Duke of Devonshire.

In opening the discussion on Mr. C. de Schwarz's paper, entitled *"The Briquetting of Iron Ores,"* Prof. Henry Louis dissented from the author's view that the use of more effective explosives increased the proportion of small and dust ore, and said that the author appeared to be under a misapprehension as to the nature of the Grondal invention, which was in use in Great Britain, Spain, Norway, the United States and, he believed, also in Germany, as well as in Sweden. When nodules were properly made in the rotating furnace they equaled the best briquets. With reference to the process in which the ores are mixed with spathic ore and lime for briquetting, his investigations did not confirm the author's statement as to the reaction which took place.

Mr. S. G. Robinson thought that the process described had been copied from the Portland cement rotary kilns, the great merit of which was cheapness of operation.

Mr. J. E. Stead said briquets were not made firm enough to stand transport without the outsides breaking up and producing 15 per cent to 20 per cent of dust. Organic compounds were suitable with magnetic oxides, but not with peroxides, the oxygen from which consumed the binding material very soon after the briquets were put into the furnace.

Mr. G. E. Moore having read his paper on *"The Manufacture of Rolled H-Beams,"* Mr. Andrew Lamberton said rolling-mill owners in Britain had not ignored the subject, but rolled H-beams were not produced here in large quantities, and there was not any mill of the type which the author had described, because it was questionable whether there was a market in this country for beams much larger than those which were commercially produced at the present. The built-up girder was coming into favor with engineers who considered the commercial as well as the professional aspect of the matter.

Mr. Moore, in reply, said H-beams up to 15 in. × 15 in. were imported into England, and he thought that they might be manufactured here.

Mr. E. Colvert-Glauert read a paper by Dr. S. Hilpert, of Berlin, and himself, on *"Sulphurous Acid as a Metallographic Etching Medium."*

Prof. A. McWilliam remarked that Mr. Colvert-Glauert was a Sheffield student, and he was interested in the fact that one of his photo-micrographs corroborated some of the views expressed in a paper by Prof. Arnold and himself, treating on the thermal transformations of carbon steels, which had not been accorded general acceptance by other metallurgists.

Mr E. H Saniter said his experiments with sulphurous acid had not been successful in consequence of the great staining effect it had on the specimens.

Mr W H Hatfield asked for a definition of austenite and hardenite, which the authors mentioned as occurring in the same sections.

Mr Colvert-Glauert replied that sulphurous acid used as described (a 3 per cent to 4 per cent solution of a saturated solution of SO_2 in water) did not produce staining unless the specimen were unduly exposed to the air before being rinsed. As to austenite and hardenite in the same section, he did not regard austenite as a true constituent.

Dr S Hilpert's paper on *"The Preparation of Magnetic Oxides of Iron from Aqueous Solutions"* evoked no discussion. It was of a purely chemical nature, and the author's main conclusions were that the production of magnetic oxide of iron Fe_3O_4 from aqueous solutions is only possible through the precipitated FeO dissolving in the ammonium residue, that the magnetic properties of Fe_3O_4 are derived from the acid property of Fe_2O_3, and that the true magnetic oxide of iron is Fe_3O_4 in the form of feric ferrate.

On the second day the proceedings opened with the reading of a paper by Mr. Arthur Hague and Prof. Turner on *"The Influence of Silicon on Pure Cast Iron,"* which was followed by Mr. H. I. Coe's paper on *"Manganese in Cast Iron and the Volume Changes During Casting,"* and the two papers were discussed together.

Mr. E. H. Saniter said that it was an interesting fact, which was brought out in both papers, that manganese increased the brittleness of cast iron. It was his experience that if ingot molds contained more than 0.5 of manganese they cracked much sooner than those with a lower percentage of manganese.

Mr. J. E. Stead was of opinion that the research conducted by the authors had not gone far enough to fully elucidate the facts. It was noteworthy that small variations in the amount of manganese caused such differences in the expansion; but, as Prof. Turner had stated, phosphorus, sulphur and other impurities behaved similarly.

Mr. E. Adamson said that with regard to hardness tests the combined carbon was generally taken as an index of hardness; but the statement that the softest bar was obtained with 0.97 and 1.04 of combined carbon was not in agreement with general experience; nor was the statement that hardness was increased after 2 per cent of silicon.

Mr. W. H. Hatfield remarked that he had demonstrated that the addition of silicon had no effect on the carbon until a certain percentage had been added, when the change occurred quite suddenly.

Mr. F. W. Harbord was of opinion that the hardening effect of silicon up to 1 per cent only applied to hard steels and was due to the carbon. On steel free from carbon, silicon would not have the same hardening effect.

Prof. Turner, in his reply, agreed with Mr. Stead's remarks as to the necessity of finding the true explanation of the accumulated facts, but he thought further research was necessary The view advanced by Mr. Adamson that silicon did not cause a reduction of the total carbon was at all events only right within the limit of 4 per cent.

The discussion on Mr. C. A. Edward's paper on *"The Theory of Hardening Carbon Steels"* was opened by Prof. Arnold, who said he believed the propositions put forward would meet with a general acceptance and would tend to reconcile the differences between the carbonist and the allotropist schools. Still it was necessary that theories should be based upon scientifically proved facts, and it was essential that before attempting to formulate a definite theory an agreement should be come to as to the method of taking the cooling curve of pure iron, and he trusted that this would soon be done. A curve taken this year at Charlottenburg with very delicate apparatus had given for pure iron practically a reproduction of a curve obtained at Sheffield 14 years ago.

Mr. E. H. Saniter, with reference to the suggested practical identity of austenite and martensite, asked if it were not the fact that in carbon steels austenite was much softer than martensite?

Prof Turner was inclined to question Mr Edwards' statements on the subject of twin crystallization, because evidence was lacking. Twin crystals were ordinarily produced only as the consequence of annealing, and he could not concede that pressure alone was adequate.

Dr Carpenter suggested that the details for the method of taking the cooling curves of iron should be referred to some scientific body such as the Physical Society.

Mr. J F. Stead said M. Osmond was the discoverer of what he called austenite, and he thought that his opinion should be asked with regard to twinning. He did not find anything of the sort in ordinary carbon steel; and concerning the pressure theory he asked if Mr. Edwards had realized that heating a bar of steel to redness and quenching it produced tension on

the outside, and not pressure? If Mr. Edwards' theory were true such steel should be soft on the outside and hard inside.

Prof. A. McWilliam thought that metallurgists would not accept Dr. Carpenter's suggestion to leave the decision as to the method of taking cooling curves to the Physical Society or any other society consisting of amateurs in such matters. Personally he would not accept a decision on the point from a special committee of the Iron and Steel Institute, but would take the curves in the way he found best.

The discussion was adjourned to the following day when Mr. Edwards, in the course of his reply, said he had no definite conviction as to the best method of taking the cooling curves. In view of the fact that austenite was regarded as softer than martensite it was noteworthy that while experimenting on a hardened, high-speed steel, which would be supposed to consist of pure austenite, he had obtained a hardness figure of 866 with the Brinell test, which he thought was superior to any ever yielded by any quenched carbon steel.

Mr. D. F. CAMPBELL's paper on *"Electric Steel Refining"* was then taken.

Mr. E. H. Saniter opened the discussion and expressed his opinion that, while there was a great future for the electric furnace, he did not think it was likely to come into general use for the production of steel rails; and his opinion was emphasized by the fact that at South Chicago, where the furnace had been working some 15 months on rail steel, it might have been expected that if the production had been economical more furnaces would have been put into operation to deal with the entire output of rails at those works.

Mr. L. Greiner congratulated Mr. Campbell on not having advocated any particular type of furnace.

Mr. J. H. Heap said the question was purely commercial, and he thought there were intermediate stages at which the use of the electric furnace would be advantageous. Obviously it could be applied to the refining of common Bessemer steel. It was desirable to have analyses of the finished steels produced in the last three months by the Illinois Steel Company, because statements had recently been made that those steels were only of average good quality. He suggested that the question of electric refining should be investigated by a committee of the Institute.

Mr. F. W. Harbord considered that it was beyond question that the electric furnace could compete commercially with the crucible furnace very easily. The main question was the quality of the product, and he thought it would be conceded that for 90 per cent of the purposes to which tool steel was applied the electrically produced steel fulfilled all requirements. The electric furnace was dear for melting, but refined economically. For high-priced material it could only be employed commercially when low sulphur and phosphorous were essential.

The Corrosion of Wrought Iron and of Steel.

Mr. WALTER MACFARLANE in his recent presidential address to the Staffordshire Iron and Steel Institute said engineers were now becoming impressed with the capacity of wrought iron for resisting corrosion. In breaking up old iron ships the condition of the metal contrasted very favorably with that of steel from ships which had had a less period of service than the iron vessels. During the overhauling of a railway bridge several years ago, over the Clyde, the condition of the iron members was found much better than that of the steel members. In the west of Scotland the output of wrought iron was greater to-day than ever it was before; and there was difficulty in getting enough puddlers. While it could not be expected that the output of wrought iron would keep pace with the enormous amount of mild steel for large structural work it was probable that the use of iron for special purposes would become more extensive.

High-Pressure Gas in the Casting of Metals.

The Birmingham Aluminium Casting Company has experimented with high-pressure gas for melting aluminium with satisfactory results—a charge of 100 lb. having been melted in slightly more than one hour in a furnace especially constructed for the purpose. Only aluminium has been dealt with at present, but it is proposed to extend the experiments to other metals and alloys.

Structural Steel for Ships.

The Engineering Standards Committee has issued a revised specification for structural steel for shipbuilding. The maximum tensile strength for plates for cold flanging is reduced from 32 tons to 30 tons per square inch. The minimum tensile strength and the elongation remain unaltered. For bars more than 1 in. in diameter, whether tested full size as from the rolls or after having been turned down, the employment of the four-diameter test piece is provided for.

Market Prices.

October, 1910.

Copper opened at £55.10, rose fairly steadily to 18th, when at £57, then fell off slightly, closing at £56. 13. 6.

Tin opened at £156, rose sharply to the 5th, when £162, then, with two small drops, rapidly to £170, on the 20th; thence declining as rapidly to £162 on the 25th; finally closing at £165. 15. 0.

Lead £13. 2. 6. at opening, rising steadily throughout the month to £13. 12. 6.

Haematite had fallen to 64/6 by the 1st and touched 63/9 on the 6th. Thence it rose and hardened, remaining at 65/- to 65/3 until the 25th, closing at 65/-.

Scotch Pig opened at 54/-, but rose again to 54/10½ by the 10th and to 55/7½ on the 12th, remaining in the neighborhood of this price to the 25th, and closing (26th) at 55/9.

Cleveland Warrants were down to 48/- on the 1st, recovered to 48/10 by the 10th, and have kept near that figure, closing (26th) at 49 11.

	£	s.	d
Alum, lump, loose, per ton	5	12	6
Antimony, black sulphide powder	24	0	0
Borax, British refined crystal	16	0	0
Sulphate of ammonia, f. o. b. Liverpool, per ton	13	0	0
Sal ammoniac, lump, firsts, delivered U. K. per ton	42	0	0
Copper sulphate, per ton	19	10	0
Camphor, 1-oz. tablets		1	7
Caustic soda, ash, 48 per cent ordinary, per ton	5	10	0
Shellac, standard, T. N. orange spots, per cwt.	4	2	6
Sulphur, recovered, per ton	5	0	0
Carbolic acid, liquid, 97/99 per cent. per gal.			11½
Hydrochloric acid, per cwt		5	0
Creosote, ordinary good liquid, per gal.			2⅛
Naphtha, solvent, 90 per cent at 160° C., per gal.			11
Petroleum, Russian, spot, per gal.			5¾
Copper ore, 10 per cent to 25 per cent unit	9/1½ to 10/1½		
Tin ore, 70 per cent, per ton	£104 to £106		
Mica, small slab, in original cases, per lb.	6d. to 1/-		

	£	s.	d.
Litharge, flake, per ton	15	5	0
Quicksilver, per bottle	8	7	6
Zinc sheets, V. M., f. o. b. Antwerp	29	7	6
Red lead, genuine, per ton	15	15	0
Bleaching powder, 35 per cent, per ton	4	5	0
Platinum, per oz. nominal	7	14	0

India Rubber, Para fine, is still somewhat unsteady, though more about a natural level. By the 5th it had fallen to 5/11, and risen again by mid-month to 6/3, dropping to 5/10 by the 26th, and still at this price.

Sulphate of ammonia has risen 5/-.

Copper sulphate is up 30/-.

Bleaching powder is brisk at Newcastle, being inquired for at £4. 15. 0.

Shellac has risen 1/6.

Platinum is still rising in the quotation. Tin ore has risen £9. Litharge is 5/- dearer. Red lead has risen 5/-.

Canal Zone Meeting of the American Institute of Mining Engineers.

Eighty members of the Institute, with their wives and guests, making up a party of 120, have completed a highly successful trip to Panama. The passenger accommodations of the Hamburg-American Atlas Line steamer *Prinz August Wilhelm* were reserved for the sole use of the party, going and returning, and with the Institute's cruising flag flying from the masthead an auspicious start was made from New York on Oct. 21. The general plan of the meeting was to hold sessions of the Institute for reading of papers and discussions on the steamer and at the Hotel Tivoli at Panama, the other incidentals being a week spent in inspecting the canal operations, 36 hours ashore at Kingston, Jamaica, and 24 hours at Havana.

At Havana members of the Institute and their friends received the visitors with enthusiasm, showing them the sights of the city and arranging a presentation to President Gomez.

PHOTOGRAPH OF MEMBERS OF AMERICAN INSTITUTE OF MINING ENGINEERS ON THEIR PANAMA TRIP.

At Kingston two very interesting excursions were arranged, one to the unique botanical garden at Castleton and another to the old capital, Spanish Town, and the gorge of the Rio Cobre, with a final dinner at the magnificently situated Hotel Myrtle Bank on the invitation of President Brunton, of the Institute.

But it was on the Isthmus that the interest and enjoyment reached their culmination. Colonel Goethals and his staff received the visitors with the utmost cordiality, placing themselves at their constant service during the whole week's stay. A special train conveyed the party each day to the part of the canal to be inspected, and with the "bosses of the job" to give intelligent and satisfactory answers to every question asked, the opportunities for obtaining thorough and accurate information upon every possible phase of the work were absolutely complete. The admiration expressed by the visiting engineers upon the conduct of the canal work was open and sincere, and found spontaneous and concrete expression on the homeward trip in the following resolutions, which, signed by every one of the engineers in the party, cannot but be regarded as a remarkable endorsement of the wisdom of the plan of the canal and of the wonderful energy with which the work is being prosecuted:

"We, the undersigned, members and guests of the American Institute of Mining Engineers, after a visit to the Isthmus of Panama, and inspection of the work of the United States Isthmian Canal Commission, and after full discussion of our individual impressions, find ourselves in unanimous agreement as to the following conclusions:

"The present plan of the work is clearly practicable and the best, in our judgment, that could be devised under the conditions imposed. It is perhaps a question whether by the choice of a higher level some of the difficulties and uncertainties of excavation in the Culebra Cut might not have been minimized, but a higher level has its disadvantages also, and no one now seriously proposes such a plan. On the other hand, we are convinced that a canal at a lower level, and especially at sea level, is practically out of the question; that no man can estimate its cost, or even guarantee its satisfactory completion and maintenance at any cost. We are satisfied that the sea-level canal, as proposed, if actually completed, would be inferior to the present lock canal by reason of its necessarily narrow and tortuous channel, its liability to many disturbances from which the lock canal is comparatively free, and so forth. The experience gained in the Culebra Cut throws additional light upon the sea-level plan and renders that scheme less worthy of approval by engineers than it was when with less information some eminent authorities favored it. In a word, we do not think that any prudent engineer would now recommend the deepening of the Culebra Cut below the level now fixed for it.

"The creation of the great Gatun Lake by means of the Gatun Dam seems to us to be the best possible way of dealing with the floods of the Chagres and other streams. The location of the Gatun Dam, spillway and locks is singularly favorable for such constructions, and there is, in our judgment, no reason for any anxiety as to their stability.

"The one serious remaining problem is presented by the nature of the ground in the Culebra Cut. There have been extensive slides on the sides of this excavation, and more of them

are to be expected, but they involve nothing more than the cost and delay of removing the material which they will force into the cut. They will ultimately end, and we regard as reasonable the calculation of the engineers in charge as to the time and money which they may call for. The results of these calculations are included in the estimates of the commission as to the cost of the canal and the date of its completion.

"We are unanimous in our praise of the manner in which sanitation, transportation and construction are performed with rapidity, skill and economy. A spirit of loyalty, emulation, industry and pride seems to animate employees and officers alike. This spirit, so difficult to arouse among workers in tropical climates, is due in this case to two causes: first, the inspiring example of Colonel Goethals and his associates, and, secondly, the splendid work of the sanitation department under Colonel Gorgas. The cities of Panama and Colon, though politically outside of the Canal Zone, have shared in the benefits of the sanitary administration and reflect an unwonted cleanliness, comfort and safety.

"We acknowledge the entire freedom and fullness with which everything we desired to see was shown to us, and everything we desired to know was told to us by the officers of the commission. There was evidently no wish to withhold or conceal anything. On the contrary, enquiry and criticism were frankly sought and heartily welcomed.

"This is but a meager summary of the points on which we are agreed. The details of individual opinion will appear later in the published report of our discussions. Meanwhile, we unite in this common declaration, which covers our conclusions on all main points. We think the present plan of the canal is good; that the work is in thoroughly capable hands; that it is progressing satisfactorily, and that it will be completed by the date set for it, Jan. 1, 1915, and probably earlier, provided Colonel Goethels and his associates receive the hearty support of the American people and its representatives in Congress. The canal engineers are the right men in the right place. The great work in which they are engaged is not connected with partisan politics, and citizens of all parties should combine to secure its early and triumphant completion. In that consummation every American should take greater pride than in any victory of military or political conflict."

[Signed by the 80 engineers (mining, civil, electrical, metallurgical and mechanical) in the visiting party, all endorsing the above and not one dissenting.]

The social features of the stay on the Isthmus included a reception by President Arosemena, of the Panama Republic, and a brilliant ball at the Hotel Tivoli, given by the visitors in honor of the officials in charge of the canal work, and which was also attended by the local government representatives and consuls.

On the return trip the U. S. S. *Tennessee* conveying President Taft to the Isthmus was met and the following message sent to the president:

"American Mining Engineers returning from Isthmus congratulate you upon good plan, splendid management and satisfactory condition of work."

To which the President replied as follows:

"Your message received. It is most satisfactory to have the assurance of men who are experts that the great canal construction has been well planned and is being carried on to a successful completion. I thank you and your associates for your visit and the courtesy of your kind message.

"WILLIAM H. TAFT."

We may without hesitation conclude that not only was the meeting one of the most interesting in the history of the Institute, but that it was of the highest public service in giving to the world an unbiased and competent opinion upon the status of the Panama Canal work such as could have been obtained in no other way.

The scientific sessions of the Institute were not particularly brilliant, although interesting to those participating. Abstracts of the papers submitted will be given in our Synopsis.

Chemists' Building in New York City.

The magnificent new Chemists' Building at 50-54 East 41st Street, New York City, is now nearing completion. It is a monument representative of the united democracy of American chemists. Without outside help it has been erected by the Chemists' Building Company, organized for this purpose by individual American chemists and chemical manufacturers.

While thus representing the ideals of American chemistry, it is a thoroughly practical twentieth-century structure with an excellent location and with excellent internal arrangements and equipment.

Fig. 1 gives an outside view of the Chemists' Building. It is a ten-story fireproof structure, gracefully designed in French

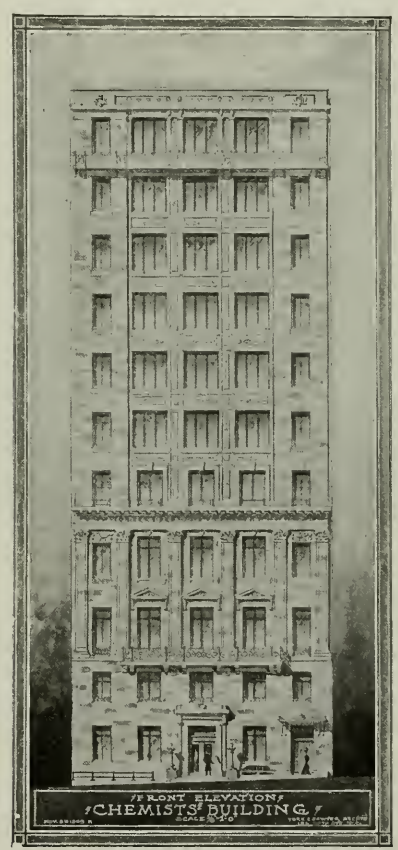


FIG. 1.—CHEMISTS' BUILDING IN NEW YORK.

Renaissance style, of the Louis XVI period. Being located in the block between Madison and Park Avenues, in 41st Street, it is close to the Grand Central Station and to the 42d Street express station of the subway, as well as to the prospective terminals of the McAdoo and Steinway tunnel systems. It is a few minutes' walk from the New York Public Library in Fifth Avenue (opposite 41st Street), from the Engineering Societies Building in 39th Street, from the Engineers' Club in 40th Street, and from the Academy of Medicine in 43d Street.

The lower five floors of the Chemists' Building are leased to the Chemists' Club and contain all the appurtenances of a social club. The first floor, a plan of which is given in Fig. 2,

contains in the rear a large auditorium with a balcony. The next floor is fitted as a lunchroom, with social rooms opening off from it. The fourth and fifth floors are to be devoted to

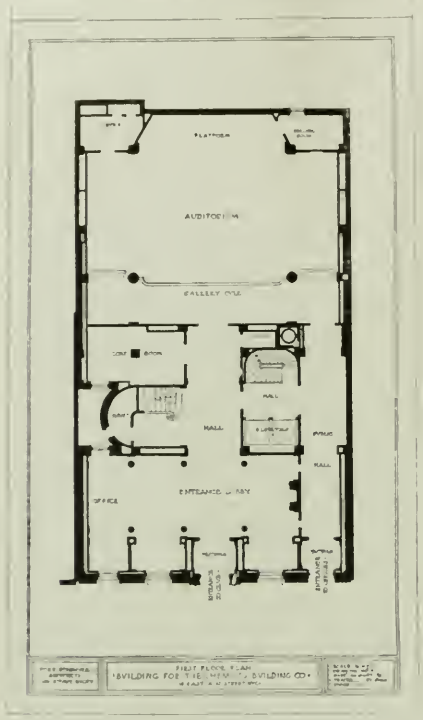


FIG. 2.—PLAN OF FIRST FLOOR.

living and sleeping rooms for the members, below which will be the library and museum.

The five upper stories from the sixth to the tenth have been especially constructed for laboratory purposes and can be rented either as entire floors or in suitable subdivisions, to analytical, commercial or research chemists, physicists, electrochemists, bacteriologists, etc., but not as manufacturing laboratories. A typical upper-floor plan with suggested subdivisions, which may, of course, be differently arranged, is shown in Fig. 3.

They are provided with ventilating flues, water, gas and electric mains, steam and compressed air lines, in suitable locations, but the tenants are expected to make their own connections and provide their own fixtures.

Rental will include janitor and elevator service, heat and electricity for lighting purposes, and the company will construct proper partition walls for the subdivision of the laboratories, according to the tenant's wishes. It is intended to charge low rentals and in many ways facilitate the prosecution of scientific and industrial research.

Among the special advantages of the internal equipment of the building are excellent light and ventilation, construction equal to that of university laboratories, command of a complete chemical library, opportunities for social and scientific intercourse, the fact that the building will be the natural meeting place of out-of-town chemists and manufacturers, for whom the Club provides excellent apartments, and finally the possibility of reduction of laboratory expenses through co-operative installation of expensive apparatus.

The value of this neighborhood for such purposes has been further enhanced by the erection of a large Physicians' and Dentists' Building alongside and the attention of physicians and pathologists is called to the advantages which the Chemists' Building laboratories could afford them for their own researches.

Until Feb. 1 communications should be addressed to Dr.

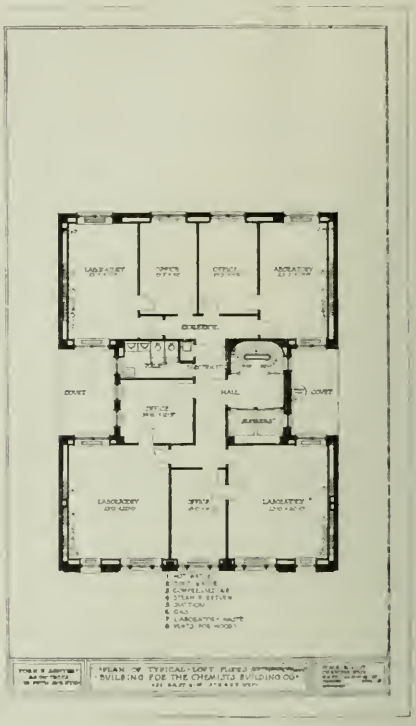


FIG. 3.—PLAN OF TYPICAL UPPER FLOORS.

Morris Loeb, President of the Chemists' Building Company, 106 West 55th Street, New York City. After that date his office will be in the building, 50 East 41st Street.

Cement Concrete Vats and Tanks.

By Albert Moyer.

Impervious, odorless, tasteless and sanitary vats and tanks for buttermilk, wine, oil, pickles, sauerkraut, etc., can be constructed of reinforced concrete, the reinforcing to be designed by a competent engineer, provided the interior surfaces are treated as follows:

After the forms are removed, grind off with a carborundum stone any projections due to the concrete seeping through the joints between the boards. Keep the surface damp for two weeks from the placing of the concrete. Wash the surface thoroughly and allow to dry. Mix up a solution of 1 part water-glass (sodium silicate) 40° Baumé, with 4 to 6 parts water, total 5 to 7 parts, according to the density of the concrete surface treated. The denser the surface the weaker should be the solution.

Apply the water-glass solution with a brush. After four hours and within 24 hours wash off the surface with clear water. Again allow the surface to dry. When dry apply another coat of the water-glass solution. After four hours and

within 24 hours again wash off the surface with clear water and allow to dry. Repeat this process for three or four coats, which should be sufficient to close up all the pores.

The water-glass (sodium silicate) which has penetrated the pores has come in contact with the alkalies in the cement and concrete and formed into an insoluble hard material, causing the surface to become very hard to a depth of ⅛ in. to ½ in. according to the density of the concrete. The excess sodium silicate which has remained on the surface not having come in contact with the alkalies is soluble, therefore easily washed off with water. The reason for washing off the surface between each coat and allowing the surface to dry is to obtain a more thorough penetration of the sodium silicate.

It is obvious that concrete surfaces so treated, if hard, impervious and insoluble, have been made impervious, tasteless, odorless and sanitary.

A New Radiation Pyrometer.

A new type of radiation pyrometer embracing several novel features has recently been devised by Mr. Richard P. Brown, of Philadelphia. As is well known, the radiation pyrometer is especially useful for measuring high temperatures, beyond the range of the thermoelectric pyrometer.

It is true, in the radiation pyrometer a sensitive thermocouple is also used, but this is not placed at the hot place the temperature of which is to be measured. The temperature is measured from a distance by means of a telescope which holds at its rear end the sensitive thermocouple. A concave mirror focuses the heat rays entering the tube of the telescope on the thermocouple, which is connected by wire to a millivoltmeter graduated in temperature degrees. The radiation pyrometer, therefore, has no part directly subjected to the high heat and there is in consequence no part to be destroyed by the furnace gases or by the high temperature. The radiation pyrometer has also the advantage of the readings being instantaneous, the slightest changes in temperature being shown immediately.

Radiation pyrometers heretofore manufactured have been of two types, the adjustable-focus and the fixed-focus instruments. In the radiation pyrometer with fixed focus to secure a correct reading of the temperature, the pyrometer tube or telescope should not be more than 30 in. distant when the furnace opening (from which the heat rays come to the telescope) is 3 in. in diameter, or 60 in. distant when the door of the furnace is 6 in. in diameter; in other words, the pyrometer telescope or tube should not be distant more than 10 times the diameter of the furnace opening, or piece of steel, the temperature of which is being measured. As long as the pyrometer tube is within these distances the temperature can be read correctly, but to determine this the distance has to be measured.

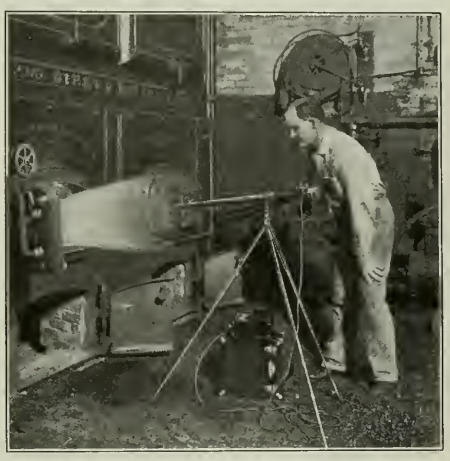
This is unnecessary with the new Brown radiation pyrometer with fixed focus, since a finder somewhat similar to those used on kodaks is placed on the tube. By means of this the tube can be readily pointed directly at the furnace opening, and at the same time the finder acts as a measure of distance, as it is only necessary that the bright red of the furnace opening takes up the whole view in the finder. If some of the dark outside wall of the furnace is showing around the bright-red opening, the tube is too far distant, and should be moved closer until the bright red only is showing. It is impossible for errors to occur with this new feature, and the instrument is stated to be accurate under ordinary working conditions within 1 per cent or 30° at 3000° Fahr.

The instrument is calibrated for measuring the temperature of a "black body," or the temperature of the walls or parts inside of furnaces which are practically "black bodies" in the thermodynamical sense. In using the pyrometer for measuring the temperature of molten metals or highly polished surfaces which reflect, a correction is made.

Another new and particularly desirable feature of the Brown radiation pyrometer is in the collapsible tube made like a telescope, which permits of the pyrometer tube, indicator, tripod and wiring being fitted in a small leather carrying case weighing complete only 15 lb. This feature is particularly desirable to engineers and superintendents who may have to take the pyrometer from one point to another occasionally for tests.

In using a radiation pyrometer focussed on a brick kiln or furnace it is frequently inconvenient, if tests of long duration are to be made, to leave the door or opening open. A firebrick tube is, therefore, inserted in the opening and projects into the kiln or furnace, the inside end of the tube being closed. The pyrometer telescope is focussed on the inner end of the firebrick tube and for permanent installation a bracket is used instead of the tripod to bolt on the wall and hold the tube.

The instrument will be adaptable particularly for measuring the temperatures about a blast-furnace plant in the open-hearth furnaces of steel works where high temperatures are employed,



RADIATION PYROMETER.

in brick kilns for burning firebrick and very refractory materials, in rotary cement kilns where the temperature is running about 3000° and must be measured 20 ft. inside the furnace, and by engineers in testing the temperature of boiler furnaces and in research work.

This new radiation pyrometer is being placed on the market by the Brown Instrument Company, of Philadelphia, with offices in Pittsburgh and Chicago, and patents have been applied for covering the advantageous features embraced in this instrument.

Lignite as Fuel.

"North Dakota Lignite as a Fuel for Power-Plant Boilers" is the title of Bulletin No. 2, just issued by the Bureau of Mines. This bulletin describes a series of tests at the pumping plant of the United States Reclamation Service at Williston, N. D. The Reclamation Service has a large project there and had installed steam boilers with furnaces designed to burn a "brown lignite" that was mined on adjacent government land.

The furnace is of the semi-gas-producer type and has an external resemblance to the so-called Dutch oven. The most striking features in its construction are the deep-set grate and the construction of the space between the bridge wall and the end of the prolonged fire brick arch. The furnace is designed to work on the gas-producer principle. The solid fuel is gasified on the grate and the gas passes through the space under the arch into the combustion chamber where most of the gaseous combustible burns.

The results of the tests on the lignite show that this fuel, though generally considered unsatisfactory, may be used with fair economy under boilers that generate their full rated capacity. In fact, when the number of heat units available is considered, the results compare favorably with those of better grades of fuel.

The tests are deemed important because the lignite deposits of the Northwest are so extensive and the distance of the region from other coal fields is so great that a large portion of the United States, including parts of North Dakota, South Dakota and Montana, may be greatly benefited by any improvements in the methods of utilizing this local fuel supply. The lignite in this field is low in heating value, some of it containing nearly 45 per cent of its weight in moisture, and it is difficult to burn in the furnaces commonly used for the better grades of coal, but the tests have shown the possibility of designing suitable furnaces for burning it profitably.

The tests were conducted by the Technologic Branch of the Geological Survey, which is now a part of the Bureau of Mines. The authors of the bulletin are D. T. Randall and Henry Kreisinger. The bulletin will be of interest to fuel engineers, especially to those located in lignite territory. It may be obtained by addressing the Director of the Bureau of Mines, Washington, D. C.

A Modern Air-Moistening Apparatus.

The moistening of air for ventilation and the removal of dust from the atmosphere is the object of the Koerting moist ventilators, an outside view of which is given in Fig. 1. The motive power in this apparatus is a fine water spray produced by forcing water (for instance, from the city water mains) through a spray nozzle (Fig. 2). The air is sucked in by this water spray and is thereby mixed with the atomized water and saturated with moisture and then forced into the discharge pipe. By this means the air in a room may be continuously passed through the apparatus and returned to the room uniformly moistened. No drops occur.

FIG. 1.—MOIST VENTILATION.

As this apparatus has no moving parts it is not subject to wear and tear. It is started by simply turning on the water. The pressure water can be taken from the city water works or can be provided by a special pump. Using a special pump the same water may be used over and over again, only the part atomized and passed into the air has to be replaced. In Fig. 1 the water enters at *C* and leaves at *D*, the air enters at *A* and leaves at *B*.

These air moisteners are used to advantage in various industrial establishments. In cutlery, cut-glass and other works where grinding, sharpening and cleaning are part of the process, the dust thrown off from the wheels is immediately drawn away by the application of these moisteners. There is the further advantage that any particles of metal or dust are precipitated as fine mud and carried away with the waste water from the exhauster and the gases are absorbed.

The moist-ventilator has also proved successful for humidifying the cheese cellars of dairies and the storerooms of dextrine or paper factories, the storerooms of cigar and tobacco factories and green houses. The regulation of the moisture is perfect, a hygrometer in the room always indicates the degree of numidity and the adjustment of the apparatus can be made accordingly. These ventilators are built with capacities from 1000 cu. ft. to 20,000 cu. ft. of air per hour by the Schutte & Koerting Company, of Philadelphia, Pa.

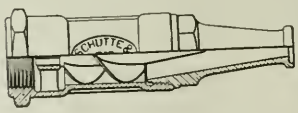


FIG. 2.—SPRAY NOZZLE.

A New Caustic Potash and Chlorine Plant at Niagara Falls.

The new plant of the Niagara Alkali Company which is to take the place of the old Roberts Chemical Company's plant is nearly completed and is expected to be in operation by Jan. 1.

The Billiter diaphragm cell, as developed under the auspices of the Siemens & Halske Company at Aschersleben, Germany, and Bruehl, Austria, will be employed for the electrolysis of potassium chloride solutions for the production of caustic potash and chlorine. The cell is "of a simple horizontal diaphragm construction, operating by percolation, thus doing away with the necessity of having a circulating brine system." It is stated that the cathode liquor of caustic potash is drawn off at a fairly high concentration, about 18 per cent, and the amp-hour efficiency in commercial operation is claimed to be about 97 per cent. It is stated that with a cell operating at 3000 amp and about 3¼ volts the production per 24 hours is 330 lb. of solid 90 per cent KOH per cell.

From the anodic chlorine gas together with the hydrogen evolved at the cathode, the Niagara Alkali Company will manufacture muriatic acid under the old Roberts patent, producing muriatic acid free from sulphur and arsenic. Part of the chlorine gas is liquified; the liquid chlorine branch of the business is handled by the Electro Bleaching Gas Company, of 24 East Twenty-first Street, New York City. It is also the intention to manufacture tin chloride, carbon tetrachloride, potassium chlorate and bleaching powder. Another branch of the business will be the manufacture of acetylene tetrachloride, chlorides and oxychlorides of metalloids and zinc chloride, and various other chlorine products.

Mr. H. D. Ruhm is president and general manager of the Niagara Alkali Company.

NOTES

Electrolysis of Water.—Hydrogen gas of practically 100 per cent purity and oxygen gas of 99.5 per cent to 99.9 per cent purity are produced from water by electrolysis by means of the Garnti process at the works of the American Oxhydrate Company, in Milwaukee, Wis. This is the only American company pushing an oxy-hydrogen welding process, while the oxy-acetylene process has found quite extended application in this country. The oxygen and hydrogen are shipped under high pressure in portable, solid-drawn, seamless, steel cylinders of 200 cu. ft. to 250 cu. ft. capacity.

The Arizona Copper Company produced during September, 1910, 1336 tons of copper.

Elmore Process.—The plant at the Sulitelma mines, Norway, produced during the month of August, 1910, 645 tons of concentrates carrying 7.38 per cent copper.

The Ray Consolidated Copper Company has issued, under date of Sept. 27, 1910, its second annual report. The company is one of those preparing to operate large deposits of low-grade copper ores in Arizona, and is building a concentrator and smelter to treat its own output. The former will be in operation early in 1911, and the latter a few months later. Coarse

crushing of the ore will be done at the mine, and ore bins will be provided at the mill to insure against interruptions. The mill will be in 8 units of 1000 tons capacity each, and will be modeled along the same lines as the large mills in Utah and Nevada. The fine grinding machinery will comprise rolls and chilian mills; the concentrating machinery of roughing and cleaning tables for the coarser sizes, and corrugated and smooth belt vanners for the finer. The entire plant will contain 16 sets of 42-in. rolls, 24 6-ft. chilian mills, 96 Garfield roughing tables, 48 Wilfley tables, and 768 6-ft. vanners, together with the necessary equipment of screening, classifying and settling apparatus. With the plant running at full capacity on a normal grade of ore the production will be not less than 80,000,000 lb. of copper per year, at a cost of 9 cents per pound.

Southern Water Power Plant.—Contract has been awarded to Messrs. J. G. White & Co., of New York, for the engineering and construction of their hydro-electric plant for the Eastern Tennessee Power Company. The dam and hydro-electric power station will be constructed on the Goose River, at Parksville, Polk County, Tenn., about 12 miles from Cleveland, Tenn. The power equipment will consist of four main units of normal rated capacity of 3000 kw each, of 200-kw exciter units and the necessary step-up transformers, etc. Each main unit will consist of one tandem horizontal turbine in enclosed case direct connected to a three-phase, 60-cycle, water-wheel type, alternating-current generator. Each generator will have mounted on its shaft an exciter having a capacity sufficient for its excitation.

The Eureka Stone & Ore Crusher Company, of Cedar Rapids, Iowa, has just published a new catalog descriptive of their stone crushing machinery and equipment. Information is given on the different uses to which the Mitchell "improved crusher" can be put, which is made in fifteen different sizes with capacities ranging from one to thirty tons an hour. The Mitchell "Eureka crusher" is also described, being a machine of all-steel construction, designed to reduce 3-in. and 4-in. rock or ore to sand in one operation. The Eureka Stone & Ore Crusher Company equips plants complete with the necessary elevators, screens, bins and dump cars. They have just recently located in a large new factory building to enlarge their facilities for manufacturing.

The **Ideal Electric & Manufacturing Co.,** of Mansfield, Ohio, announces that the Machinery and Supply Company, of 61 Freemont Street, San Francisco, Cal., has been appointed its Western agent. The Machinery and Supply Company now carry a full line of "Ideal" apparatus, including alternating and direct-current generators, induction motors, both of the slip-ring and squirrel-cage types, special elevator motors and also a complete line of direct-current motors.

Spring Fitting Furnaces.—The W. S. Rockwell Co., of New York City, has issued an illustrated pamphlet of the Rockwell spring-fitting furnace which is designed to heat the steel uniformly and cleanly and to meet the convenience of the fitters so as to turn out a large number of springs per day with the least hard work. The furnace can be operated either with oil or gas fuel.

The **Golden West Pump Co.,** of San Francisco, Cal., has issued a leaflet giving prices and capacities of centrifugal pumps, also tables stating power required for operating. The pumps are intended principally for irrigation and are listed in sizes from 1 in., 25 gal. per min. to 50 in., 100,000 gal. per min. and for heads from 5 to 75 ft. An interesting method of rating is given for irrigation work, for instance, the 12-in. size delivering 4500 gal. per min. will cover 1 in. deep in one hour 10 acres of land; or 6 in. deep in 100 days, pumping 12 hours a day, 44,444 acres.

Pumping Machinery.—The Deming Co., of Salem, O., has issued a new and handsomely illustrated "Catalog H" describing in 190 pages their line of pumping apparatus, including single and double-acting triplex pumps for various services, deep-well power-working heads, artesian well cylinders, rotary and centrifugal pumps. The standard Deming pumps for general service are triplex and arranged for belt, electric, gas engine or water-wheel drive. Direct-acting steam pumps are not shown as the separate-driven type is regarded as much more efficient. They are also self-contained and require no special foundation. A number of artistic illustrations show the pumps in operation in mine and irrigation work, boiler feeding, paper and pulp mills and in general industrial work, in brine circulating, hydraulic pressure accumulators, fire and water supply service, artesian wells, etc. One of the special designs which has found favor with mine managers is a portable, electrically driven triplex pump which is permanently mounted on a mine truck and readily hauled to the section of the mine which needs pumping out. In addition to the pumps a line of small compressors is shown and valves and fittings are illustrated. The catalog also contains some useful technical information, data on compression, capacities of pumps, tables of power, transmission and friction losses and other hydraulic calculations.

Messrs. **Tate, Jones & Company,** of Pittsburgh, have received from the Mogollen Gold & Copper Company, of Cooney, N. M., an order for oil-burning equipment to be applied to their boilers. Their plant is 90 miles from a railway.

Personal

Mr. **H. B. Abbott** has assumed the position as Western representative of this journal, with headquarters at Pittsburgh.

Mr. **Victor M. Braschi,** of Mexico City, was in New York recently.

Mr. **Max Drott,** metallurgist for the Krupp Co., Germany, was in New York recently en route to the mining districts of Central America.

Mr. **Ernest Gayford,** secretary of the General Engineering Company, of Salt Lake City, recently made a trip through Montana and Idaho.

Mr. **John Gross,** of Draper & Gross, Denver, is on a professional trip to Sonora, Mexico.

Mr. **Edmund B. Kirby,** St. Louis, Mo., recently made a trip to the Skeena River district in British Columbia.

Mr. **Mark R. Lamb** is on an extended professional trip through South America for the Allis-Chalmers Company.

Messrs. **James M. McClave** and **Warren B. Lippincott** have recently entered into partnership and opened offices in the Ideal Building, Denver, where they will engage in practice as metallurgical and chemical engineers, giving special attention to the treatment of complex ores. Both of these gentlemen have been identified with the zinc industry of the West for some years.

Mr. **Kenneth Seaver** has been appointed chief engineer of the Harbison-Walker Refractories Company, Pittsburgh.

Dr. **Edgar F. Smith,** the distinguished professor of chemistry and electrochemistry at the University of Pennsylvania and for a series of years vice-provost of the university, has been elected provost on the retirement of Mr. Harrison.

The many friends of Mr. **Frank J. Tone,** works manager of the Carborundum Company of Niagara Falls, will be delighted to hear that he is now quickly recovering from his recent serious illness. He is now going to Colorado or New Mexico for the winter and expects to be back at work in Niagara Falls about May first.

Mr. **G. E. Webber,** who has been in the Transvaal for 17 years, has resigned his position as manager of the Rand Mines, Ltd., and will return to California.

Mr. **O. F. Westlund,** formerly manager of the Aguascalientes plant of the American Smelting & Refining Company, has resigned that position and is now general manager of the Mines Management Company, of New York, with offices in Mexico City.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

Electric Furnaces (Continued).

688,364. Dec. 10, 1901. Edward R. Taylor, of Penn Yan, N. Y.

Resistance or arc type, especially for the production of carbon bisulphide. The furnace comprises a vertical stack of iron lined with fireclay, the lower portion being enlarged and receiving two or more pairs of horizontal opposed carbon electrodes. The electrodes pass through packing glands, and are insulated with alternate layers of mica and asbestos. In operation, coke or broken carbon is fed down through hoppers onto the electrodes to protect the latter from excessive wear. Arcs may be sprung between the electrodes, or enough broken carbon may be fed in between them to serve as a resistor. The central shaft of the furnace is charged with charcoal. Surrounding the middle portion of the shaft is an annular chamber or jacket which is charged with sulphur. The lower enlarged portion of the furnace is also surrounded by four concentric annular chambers receiving sulphur. In operation the sulphur melts and feeds into the zone of heat, there becoming vaporized and rising through the charcoal and forming carbon bisulphide vapor, which is led from the top of the stack to a condenser. The sulphur being a non-conductor can be used to regulate the electric current by varying its rate of supply. The charge ingredients in the annular chambers intercept the escaping heat and return it to the working zone.

688,393. Dec. 10, 1901. Ramon Chavarria Contardo, of Sèvres, France.

Arc type. The furnace has the general shape of the ordinary Bessemer converter, or of a converter having two pouring mouths at opposite sides. The refractory walls are cooled by inclosed pipes carrying water or air. The trunnions are hollow and receive the opposed electrodes, which pass through water-cooled stuffing boxes. The charge, introduced through the mouth or mouths, is distributed over the whole of the bed and under the arc by tilting the furnace to an angle dependent upon the size and shape of the component grains. If the charge becomes pasty and does not descend freely the furnace is tipped further. Charging and smelting is continued until the molten mass reaches nearly to the electrodes, whereupon the furnace is tipped to pour. The single-mouth converter has a tap hole in its rear end. During fusion the exposed walls and tap hole are raised to a temperature often exceeding that of the molten material, superheating the latter as it runs over them when the furnace is tilted to pour.

688,861, Dec. 17, 1901, Charles Albert Keller, of Paris, France.

Resistance type. The two electrodes or terminals are carried by carriages, movable to and from each other by cables and pulleys. Each carriage comprises a vaulted floor, carrying four square blocks of carbon, connected to the electric mains by thin copper sheets. Small carbon sticks projecting upward from the blocks, and the spaces around them, are filled with refractory material, giving an inclined surface. Between the carriages is a truck having a brick hearth to receive the charge. The carriages and truck are inclosed and covered by a chamber having a stack for escaping gases. In operation both the carriages and hearth are covered with loose carbon, which acts as a resistor. When the carriages are drawn apart the charge breaks at the junction of the carriages and hearth in the zone of greatest heat and where the charge has reached the highest degree of fusion. In the production of chrome iron, to prevent its carburization, the carriages and hearth may be covered by a layer of particles of iron, acting as a resistor.

NEW BOOKS.

Allen, J. R., and **Bursley, J. A.** Heat engines: steam, gas, steam turbines, and their auxiliaries. 288 pages, illustrated. Price, $3. New York: McGraw-Hill Book Company.

Carhart, H. S. Physics for college students. 623 pages. Price, $2.25. Boston: Allyn & Bacon.

Ennis, W. Duane. Applied thermodynamics for engineers. 438 pages, diagrams, 316 illustrations. Price, $4.50. New York: Van Nostrand Company.

Kohlrausch, Friedrich. Gesammelte Abhandlungen. Edited by W. Hallwachs, A. Heydweiller, K. Strecker, P. Wiener. Vol. I. Elasticity, heat, optics, absolute electric measurements and miscellaneous. With the portrait of Kohlrausch as frontispiece. 1108 pages, 1 table and 117 illustrations. Price, bound, 27 marks (in New York, $9, retail). Leipzig: J. A. Barth.

Mineral Industry for 1909. Vol. 18. Edited by W. R. Ingalls. 878 pages. Price, $10. New York: McGraw-Hill Book Company.

Ostrup, J. Christian. Standard specifications for structural steel, timber, concrete and reinforced concrete. 99 pages, illustrated. Price, $1. New York: McGraw-Hill Book Company.

Stoddard, J. T. Introduction to general chemistry. Price, $1.60 net. New York: Macmillan Company.

Wellcome Photographic Exposure Record and Diary, 1911. (This well-known annual appears in time for Christmas, neatly bound in red cloth, and contains again a great deal of useful information on developing and printing, but especially on the problem of correct exposure.) 271 pages. Price, 50 cents. New York and London: Burroughs, Wellcome & Company.

BOOK REVIEWS.

Technologie der Holzverkohlung unter besonderer Berücksichtigung der Herstellung von samtlichen Halb und Ganzfabrikaten aus den Erstlingsdestillaten. By **M. Klar,** chief chemist of F. H. Meyer, engineers for wood distillation plants. Second edition (1910). 430 pages, 49 illus. Price in New York, $3.70. Berlin: Julius Springer.

Wood distillation has become a very important chemical industry and is still rapidly growing. But exact knowledge of the engineering features and of the commercial conditions under which financial success may be hoped for was for a long time restricted to a comparatively small number of specialists.

Mr. Klar's book, the first edition of which appeared in 1903, was the first reliable modern work based on practical experience and giving exact data. The new edition has been practically rewritten to give to the work more the character of a handbook, and it has been considerably enlarged to cover the progress which has been made in the past six years in the methods of wood disillation and construction of apparatus. In this way a thoroughly up-to-date handbook has been produced based on the practical experience of the author.

There are 20 chapters, dealing with the following subjects: History of wood distillation; raw materials, the chemical changes in wood during distillation; products of distillation, condensing gases, their composition, properties and uses; wood vinegar and its composition, tar and its constituents; charcoal, preparation of wood for distillation; distillation apparatus for large pieces of wood, for wood scrap and sawdust; apparatus for cooling the products of distillation; treatment of non-condensing gases; manufacture of crude wood alcohol and acetate of lime, manufacture of acetic acid and acetone; tar oils, formaldehyde, production of charcoal briquets; analysis of the various substances; list of patents.

Mr. Klar's book may be sincerely recommended to anyone interested in any way in wood distillation.

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